MADSEN GOLD PROJECT TECHNICAL REPORT FEASIBILITY STUDY FOR THE MADSEN DEPOSIT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE FORK, RUSSET SOUTH AND WEDGE DEPOSITS RED LAKE, ONTARIO, CANADA

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NOTICE

JDS Energy & Mining Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Pure Gold Mining Inc. The quality of information, conclusions and estimates contained herein is 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.

Pure Gold Mining filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

This report summarizes the results of the Feasibility Study (FS) completed by JDS Energy & Mining Inc. (JDS) as commissioned by Pure Gold Mining Inc. (Pure Gold or the 'Company') for the Madsen Gold Project (the Project or Madsen) and was prepared in accordance with the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as National Instrument (NI) 43-101.

This report also contains the results of a Preliminary Economic Assessment (PEA) of gold mineralization at the Fork, Russet South, and Wedge deposits. The results of the PEA analysis are not included in the Feasibility Study. The PEA has been prepared in accordance with the NI 43-101.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the project presented in the PEA will be realized.

The Madsen Gold Project is a precious metals resource project located in the Red Lake district of Northwestern Ontario. Based on Probable Reserves, a subset of the Indicated Resource category, the Project will develop a mining and processing operation with mining from five zones near the historical Madsen Mine over a 12-year production period by way of a ramp and an overhauled shaft access underground mine, an expanded and refurbished cyanidation gold extraction processing facility capable of processing ore at an average rate of 800 tonnes per day (t/d), a cyanide destruction circuit, an upgraded tailings management facility, a water treatment plant and related infrastructure.

1.2 Property Description and Ownership

The Madsen Gold Project, which is 100% owned by Pure Gold Mining Inc. is centered around the historical Madsen Mine, which produced 2.5 million ounces at an average grade of 9.7 g/t gold (7.9 million tonnes) between 1938 and 1976 and again from 1997 to 1999. The Madsen Gold Project comprises a contiguous group of 251 mining leases, mining patents and unpatented mining claims covering an aggregate area of 4,648 hectares (46.5 km^2). Infrastructure includes paved highway and secondary road access, a mill and tailings facility and access to power and water.

The Project is located in the Red Lake district of Northwestern Ontario, approximately 440 km Northwest of Thunder Bay, Ontario, 260 km east-northeast of Winnipeg, Manitoba and 10 km south-southwest via provincial highway ON-618 S from the town of Red Lake. The Project is adjacent to the community of Madsen at approximately 93.91 degrees longitude west and 50.97 degrees latitude north. Access to the Project site is via the Mine Road off ON-168 S and access to Red Lake is via ON-105 N from the Trans-Canada Highway / ON-17 and via commercial airline flying into the Red Lake Municipal Airport.

1.3 History, Exploration and Drilling

Gold was discovered in the Red Lake area in 1925 and the first claims were staked in the Madsen area in 1927. Initial development at the Madsen Property was focussed on the Madsen No. 1 Vein where a shaft was sunk and underground exploration commenced in 1936. The Madsen deposit was discovered in 1937 and promptly went into production a year later with sinking of the Madsen No. 2 shaft which ultimately reached a depth of 1,273 m with production from 27 levels. 8 Zone was discovered in 1969. Production in the mine was halted in 1974 and the mine was placed into Temporary Suspension. Production at Madsen Mine to this time totalled 2.43 million ounces from 7.6 Mt at a recovered grade of 9.91 g/t gold. Little work occurred at Madsen until 1997 when exploration and development resumed. In 1998 Claude Resources purchased the project and in 1998–99, produced about 22,000 ounces of gold from the Madsen shaft and the newly developed McVeigh portal but ceased production in October, 1999 due to low gold prices and low head grades resulting from excessive mining dilution. From 1999 to 2013 Claude focussed on exploration of the property and compilation and conversion of an extensive hardcopy historical record to digital formats.

Pure Gold (then Laurentian Goldfields) purchased the project in 2014 and embarked on a property-wide geoscience and exploration program. Work focussed on integrating new geologic mapping and geochemical data with the geological learnings from the 38 years of mining development into a new property-wide geological framework. From 2014 to 2018, Pure Gold drilled 904 core holes (for 210,645 m), re-logged 595 historical drill holes (for 271,000 m), developed a new geological model for mineralization which formed the basis for a new Madsen Mineral Resource estimate and discovered and published maiden Mineral Resource estimates for three new deposits (Fork, Russet South, and Wedge) delineated through systematic exploration of the broader Madsen gold system. In 2017, Pure Gold reconditioned the McVeigh portal and completed underground exploration and delineation drilling of the bulk sample area. In 2018 the company completed ongoing environmental baseline and feasibility-level studies and collected a 7,096 tonne bulk sample from the McVeigh Zone.

1.4 Geology and Mineralization

The Madsen Project is located within the Red Lake Greenstone Belt (RLGB) of the Archean Superior Province craton of the Canadian Shield. The RLGB is approximately 50 km by 40 km and comprises 2.99-2.70 Ga deformed and metamorphosed supracrustal (volcanic and sedimentary) rocks intervening between three main granitoid batholiths. The RLGB boasts a prolific history of gold production over a 90 year history. All major deposits in the RLGB are hosted within the ca. 2.99-2.96 Ga Balmer Assemblage which includes the belt's oldest volcanic rocks that are predominantly of submarine mafic thoeliite and ultramafic komatiite composition. Gold deposits in the RLGB are classified as orogenic gold deposits (Groves et al., 1998) and characterized by an association with crustal-scale fault structures. Gold deposition in orogenic gold deposits is typically syn-kinematic and syn- to post-peak metamorphic and is largely restricted to the brittle-ductile transition zone.

Rock units of the RLGB are multiply deformed and metamorphosed. On the Madsen Property, this complex deformation history is most readily explained through an early phase of tight upright folding (D1) and an overprinting minor folding event and associated widespread foliation development (D2). Significantly, the Madsen, Fork, Russet South, and Wedge deposits all occur within planar structures (shear zones) that developed generally axial-planar to property-scale D1 folds. These early planar mineralized structures are the main targets for further gold exploration at the Madsen Gold Project and although they have been strongly overprinted by penetrative D2 deformation and metamorphism, they can be effectively identified by a distinct series of mineral phases (alteration), vein styles (blue-grey, deformed quartz veins) and quartz porphyritic intrusions that pre-date gold mineralization but are very common within the D1 shear zones.

In some ways, the Madsen deposits seem atypical in that they are strongly overprinted by deformation and metamorphism rather than being syn- to post-peak metamorphic. The age of D1 deformation and gold mineralization, however, is poorly constrained and if the overprinting deformation is unravelled from the Madsen deposits, they closely fit the orogenic gold deposit model including an association with crustal scale structures and an association with pervasive structurally-controlled carbonate alteration and quartz-carbonate veining.

1.5 Mineral Processing and Metallurgy

The most recent metallurgical program, completed in 2018 in support of this Feasibility Study, was carried out with the primary objective of confirming the flowsheet and design criteria using a combination of new testwork, historical data and the existing plant design. Drill core and underground samples from the four Madsen deposit zones, McVeigh, South Austin (including the new A3 domain), Austin and 8 zones, was sent to Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC for test work that included sample preparation, interval assaying, mineralogy, gravity concentration, cyanide leach and cyanide destruction. Process optimization test work was also conducted to further the understanding and optimization of the processing characteristics in support of this Feasibility Study and evaluate the existing equipment in the plant.

The test program was done in four phases: Variability Scoping Composites, Year Composites, Tailings Generation and Variability Composites. The first phase was scoping variability tests on 12 composites from the five zones (The A3 domain was split out from the South Austin zone) to evaluate the metallurgical response using the existing plant flowsheet and historical data. Phase 2 included test work on the three year composites that represented the years -1 to 1, 2 to 3 and 4 to 7 of the mine schedule at a target head grade of 8 g/t. The final phase tested the optimized flowsheet using 30 variability composites representing the five zones of the deposit.

The mineralogy indicated that the sulphur content is mainly associated with Pyrrhotite and Pyrite and trace amounts of Arsenopyrite. The comminution test work included Crushing Work Index (CWi), SAG Mill Comminution (SMC), Bond Ball Mill Work Index (BWi) and Abrasion Index (Ai). The results indicate the material is soft to moderately hard with a BWi ranging from 9.5 to 17.1 kWh/tonne with an average of approximately 14.5 kWh/tonne and an average Ai of 0.266 g. A correlation between gold extraction and head grade was not observed. The variability composite results averaged 96.6% gold extraction and gravity gold recovery of 45.7%.

Based on the results from Base Met (2018), gold doré can be produced with a primary grind size of 80% passing (P_{80}) 75 µm followed by gravity concentration, 2 hour pre-oxidation, 250 g/t lead nitrate, a 24 hour cyanide leach at a cyanide concentration of 500 ppm and a pH of 10.5, 6-hour carbon-in-pulp (CIP) adsorption, desorption and refining process. The blended average of the samples tested, based on the mine plan, using this method is estimated to achieve a recovery of 96% Au.

1.6 Mineral Resource Estimate

The Madsen Mineral Resources were estimated by Ginto Consulting Inc. (Ginto) The Madsen resource drill hole database comprised 14,822 holes totaling 1,220,042 m of drilling, with a cut-off date of January 16, 2018. 549 of these holes were drilled by Pure Gold from 2014 to 2018. The mineral resources were estimated with an ordinary kriging technique on composited and capped gold assays.

The Fork and Russet South deposits were both updated as part of the current mineral resource estimates, with drill hole cut-off dates of July 10, 2018 and August 10, 2018 respectively. The mineral resource estimate at Fork considered data from 45,525 metres of drilling from 122 drill holes, including five new drill holes since the December 2017 estimate. At Russet South, the mineral resource estimate considered data from 26,567 metres of drilling from 105 drill holes, including 24 new drill holes completed since December 2017.

Mineral resources at Wedge, located approximately three kilometres by road south of the Madsen milling infrastructure, were estimated for the first time in this Mineral Resource. Surface drilling in the Wedge area was conducted by Pure Gold in late 2017 through 2018, establishing strongly mineralized zones with good continuity that remain open for expansion. In total, the Wedge mineral resource estimate considered 52,238 metres of drilling from 201 drill holes, with a cut-off date of August 29, 2018.

The estimation methodology for the Fork, Russet South, and Wedge deposits used the same approach to that of the Madsen deposit.

Table 1-1 summarizes the Mineral Resource with an effective date of February 5, 2019.

Table 1-1: Madsen Gold Project Mineral Resource Statement – Effective February 5, 2019

Zone	Indicated			Inferred		
	Tonnes	Au Grade (g/t)	Ounces (Au)	Tonnes	Au Grade (g/t)	Ounces (Au)
Madsen	6,429,000	9.0	1,857,000	889,000	8.4	241,000
Fork	203,000	6.6	43,000	331,000	5.8	61,000
Russet South	241,000	7.2	56,000	352,000	7.5	85,000
Wedge	322,000	10.3	107,000	307,000	8.0	79,000
Total	7,196,000	8.9	2,063,000	1,880,000	7.7	467,000

Notes:

1) The Qualified Person for the Mineral Resource estimate is Marc Jutras, P.Eng. of Ginto Consulting Inc.
2) Mineral resources are reported at a cut-off grade of 4.0 g/t gold based on US$1,275 per troy ounce gold and gold metallurgical recoveries of 95 percent.
3) Mineral resources are reported at a cut-off grade of 4.0 g/t gold based on US$1,275 per troy ounce gold and gold metallurgical recoveries of 95 percent.
4) 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 estimated will be converted into mineral reserves. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
5) The CIM definitions were followed for the classification of indicated and inferred mineral resources. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred mineral resources as an indicated mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated mineral resource category
6) All figures in Table 1-1 have been rounded to reflect the relative accuracy of the estimates.

Source: Ginto Consulting Inc. (2019)

1.7 Mineral Reserve Estimate

The effective date for the Mineral Reserve Estimate is February 11, 2019 and was prepared by JDS. All Mineral Reserves in Table 1-2 are Probable Mineral Reserves. The Mineral Reserves are included in the Mineral Resources.

The Qualified Person (QP) has not identified any risks including legal, political, or environmental that would materially affect potential Mineral Reserves development.

Table 1-2: Madsen Gold Project Mineral Reserve Estimate

Class	Diluted Tonnes (kt)	Au Grade (g/t)	Au Ounces (koz)
Probable	3,512	8.97	1,013
Total	**3,512**	**8.97**	**1,013**

Notes:
1) The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.
2) Effective date: February 11, 2019. All Mineral Reserves have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under NI 43-101.
3) Mineral Reserves were estimated using a US $1,275 /oz gold price and gold cut-off grade of 4.75 g/t for all mining zones with the exception of McVeigh which used a cut-off grade of 4.0 g/t. Other costs and factors used for gold cut-off grade determination were mining, process and other costs of $226.00/t, transport and treatment charges of $6.00 /oz Au, CAD:USD exchange rate of 0.78 and a gold metallurgical recovery of 95%.
4) Tonnages are rounded to the nearest 1,000 t, gold grades are rounded to two decimal places. Tonnage and grade measurements are in metric units; contained gold is reported as thousands of troy ounces.
5) Rounding as required by reporting guidelines may result in summation differences.

Source: JDS (2019)

1.8 Mining Methods

Madsen is proposed to be mined as an underground operation using a combination of longhole stoping (LH) with unconsolidated rock fill, conventional cut and fill (CCF) and mechanized cut and fill (MCF) both using hydraulic and rock fill. A target production rate of 800 t/d is envisioned over an operating mine life of 12-years that will extract 3.5 Mt of ore. LH Stoping, CCF and MCF will account for 25%, 59% and 16% of the total ore production respectively. The Madsen deposit will be accessed by extending the current underground ramp and tying into the existing mine levels spaced approximately every 50 metres. Ore and un-mineralized mine rock will initially only be trucked out of the mine, with shaft hoisting being utilized to bring ore to surface from year 4 onwards. A mix of fully electric and diesel mining equipment will be utilized to reduce ventilation requirements through the mine life.

The ventilation network will require a combination of 3.7 meter and 2.4 meter diameter raises, as well as the ramp to distribute fresh air. The tight-lined ventilation partition of the shaft will be utilized as an exhaust path for the ventilation network. Mine air heaters will be installed at both fresh air intakes.

Dewatering of the current workings will be done with a network of pumps utilizing the Madsen shaft. The mine is currently dewatered to approximately 240 m below surface.

The mine production and development schedule is shown in Table 1-3.

Table 1-3: Mine Production and Development Schedule

Parameter	Unit	Totals	Year													
			-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Ore Mined	kt	3,512	13	265	293	293	290	293	293	293	293	293	293	293	266	43
Mining Rate	t/d		88	726	802	802	793	802	802	802	802	802	802	802	728	483
Ore Feed	kt	3,512	-	278	293	293	290	293	293	293	293	293	293	293	266	43
Mill Rate	t/d		748	802	802	802	802	802	802	802	802	802	802	733	119	748
Au Feed Grade	g/t	9.0	-	7.0	8.5	10.5	8.2	13.7	13.2	10.7	8.6	7.5	6.8	6.4	6.7	6.3
Milled Au	koz	1,012	-	62	80	99	77	129	124	100	81	71	64	60	57	9
Ore - LH	kt	862	13	156	134	133	71	56	76	59	24	23	39	24	53	1
Ore - CCF	kt	2,078	-	71	135	106	206	169	171	174	181	197	223	231	186	28
Ore - MCF	kt	564	-	38	24	53	12	67	46	60	88	71	30	36	26	14
Ore - Access Dev	kt	7	-	-	-	1	-	2	2	-	1	1	-	-	-	-
Lateral Capital Dev	km	60.9	2.7	8.3	7.5	7.8	7.6	9.4	8.0	1.8	1.3	1.4	1.8	1.9	1.1	0.1
Lateral Un-mineralized Dev	km	28.6	-	0.9	1.3	1.7	1.0	3.3	3.7	2.5	2.9	3.1	2.3	2.9	2.0	0.8
Lateral Slashing	km	12.6	0.8	2.8	3.3	3.2	1.7	0.8	0.1	-	-	-	-	-	-	-
Vertical Un-mineralized Dev	km	2.1	-	0.3	0.3	0.5	0.2	0.5	0.2	-	-	-	-	-	-	-
Vertical Stope Dev	km	23.4	-	1.8	1.8	1.7	2.4	1.6	1.9	2.2	2.1	2.6	2.0	2.4	1.1	-
Hydraulic Fill Placed	k m^3	951	-	34	60	50	68	77	69	85	105	105	99	104	80	14
Rock Fill Placed	k m^3	438	-	68	52	56	28	34	49	35	22	21	23	20	27	4

Note: Dev=Development
Source: JDS (2019)

1.9 Recovery Methods

The processing plant will have a capacity of 800 t/d. The plant consists of the following unit operations:

- Single stage crusher (new);
- New 600 tonne storage bin and reclaim system;
- SAG Mill (existing) and Ball Mill (new);
- Gravity separation and intensive leach system (new);
- Pre-leach thickening to 50% solids;
- 2 hour pre-oxidation (new), 24 hour leach and 5 hour Carbon in Pulp;
- 1 tonne carbon plant (existing) and gold recovery in a new refinery;
- Cyanide destruction (new) utilizing 2 tanks (1 standby); and
- Tailings pumping to the tailings facility and/or to the underground as hydraulic backfill.

The leach circuit will have a retention time of 24 hours and sodium cyanide (NaCN) consumption is expected to be in the range of 0.3 kg/t to 0.4 kg/t to maintain a cyanide concentration of 500 ppm. Cyanide will be destroyed using the SO_2/Air process. The slurry will be agitated in one of two tanks for 1 hour using copper sulphate ($CuSO_4$) as a catalyst, maintaining a 30 mg/L concentration in solution and sodium metabisulphite (SMBS) solution will be dosed into the system as the source of SO_2.

The grinding circuit target size is approximately P_{80} of 75 µm. The crushing circuit will 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% processing 800 t/d.

1.10 Project Infrastructure

The following infrastructure items will be installed to support the Project:

- High voltage (HV) electrical substation, electrical site distribution network, and diesel emergency power generators;
- Primary crusher, discharge conveyor, and crushed ore bin;
- Process plant improvements with new equipment and refurbishment;
- Assay laboratory;
- Buildings for offices, mine dry and warehousing;
- Backfill plant for underground hydraulic backfill;
- Tailings management facility upgrading;
- Mine rock management facility;
- Overburden and soil stockpiles for reclamation; and

- Water treatment plant (WTP).

The project site as an existing mine site, is already developed and has site infrastructure and existing facilities that will be utilized for the project. The site is accessible by paved highway from Red Lake, ON, and is connected to the electrical grid and municipal water and sewage systems, which will serve the project throughout operations.

During pre-production and into year one of operations a temporary mine rock facility will be used to stockpile material for processing.

Excess un-mineralized mine rock from the mining activities will be stored in the mine rock management facility (MRMF) and the tailings management facility (TMF). The TMF will be an upgrade of the existing tailings management facility. All of the disturbed areas on site will drain to the TMF catchment, minimizing the need for surface water management controls for the project site. The existing polishing pond will be used throughout the operation life to collect and reclaim water from the TMF, and all disturbed area drainage.

A WTP facility will be constructed next to the polishing pond. It will be designed and constructed to ensure the release of all water is within the effluent limits and requirements of the applicable project permits and regulations.

1.11 Environment and Permitting

Pure Gold has prepared an updated Project Definition (January 2019) based on this feasibility study. The Project as described in the Project Definition will not be classed as a designated project under the Canadian Environmental Assessment (CEA) Act (2012) as the thresholds defined are not triggered. Pure Gold has prepared a screening document (February 2019) for submission to the Canadian Environmental Assessment Agency to confirm that this is not a designated project.

The Ontario Ministry of Environment, Conservation, and Parks (MECP) has indicated that there will be no requirement for the submission of a Provincial Environmental Assessment based on the presentation of the March 2017 Project Definition and the materials presented in December 2018 for the new Industrial Sewage Permit. It is however anticipated that the Project will require approvals from the MECP, the Ontario Ministry of Energy, Northern Development, and Mines (ENDM), the Ontario Ministry of Natural Resources and Forestry (MNRF), the Ontario Ministry of Labour (MoL) and the Ontario Energy Board including: Environmental Compliance Approval (ECA) Industrial Sewage Works Permit; ECA for Air and Noise; Permit to Mine and a Mine Closure Plan update amongst others.

1.11.1 Consultation and Engagement

Pure Gold has committed to engagement and consultation with local First Nations, Metis Nation of Ontario, provincial and federal governments, the public, and stakeholders throughout all stages of Project planning, regulatory review, and construction. The intent is to provide all interested parties with opportunities to learn about the Project, identify issues, and provide input with the goal of positively enhancing Project planning and development.

The Project 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. ENDM will undertake and fulfill the Crown's Duty to Consult. Currently, the primary role of Pure Gold with First Nations is to ensure that appropriate information sharing occurs. Pure Gold maintains contact and dialogue with the Treaty 3 First Nations of: Wabauskang First Nation; Lac Seul First Nation; Wabaseemoong First Nation; Grassy Narrows First Nation; and Naotkamegwanning First Nation. This information is also shared with the Métis Nation of Ontario. Pure Gold has reached terms, and expects to finalize a Project Benefit Agreement with Wabauskang and Lac Seul First Nations in 2019.

Pure Gold has had ongoing communication with Red Lake Municipality, community and local business community who have an interest in the Project. Pure Gold's plan to reopen the Madsen mine is largely seen as a benefit to the local community, which has suffered economically from the decline in mining over recent years. Regulators from, the Canadian Environmental Assessment Agency (CEAA), MECP, MoL, ENDM, MNRF and Red Lake Municipality have been involved with Pure Gold at the mine or in evaluation of proposed reopening activities at the mine.

1.11.2 Mine Reopening from Temporary Suspension

For reopening of the mine, as defined in the Feasibility Study and Project Definition, Pure Gold intends to establish an upgraded tailings management facility, a mine rock management facility, overburden and soil stockpiles, and an improved road haulage network, as well as a refurbishment of the existing processing plant. Cyanide detoxification (destruction) and a water treatment facility will be added. Based on the recent independent noise and air quality baseline study and modelling, noise and dust management features have been designed that are anticipated to eliminate any effect to local communities. All new features have been designed to occur within the existing catchment, so no new impacts are expected.

Pure Gold has expended considerable effort, at their own cost, to clear legacy waste, monitor for new impacts, and reclaim the surface as well as manage both public and mine-related activities at the site. This expenditure has contributed to a much safer and better working environment and is expected to reduce closure costs at the end of mine life.

In order to manage the water that is utilized as a result of mine operation, Pure Gold will implement several management measures to safe guard water quality downstream of the TMF, including: recirculation and reuse of tailings system water; cyanide destruction technology; a water treatment plant to clean water to be released from the TMF; pump back, if required, of any seepage from the TMF, facilitating the Red Lake Municipality to eliminate Madsen community sewage inputs to the TMF; and progressive closure of the tailings pond to reduce water flows and the remobilization of arsenic and to dry out the upper portions of the TMF that could contribute to seepage.

1.11.3 Mine Closure Plan

Pure Gold 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 2014. Reopening of the mine will require an update of the closure plan and closure bond reassessment. These actions will facilitate final closure by allowing Pure Gold to fund further progressive reclamation of the Project site.

1.12 Operating and Capital Cost Estimates

1.12.1 Capital Cost Estimate

Life-of-mine (LOM) project capital costs are estimated to total $327 M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to an average of 800 t/d underground production rate. Initial capital costs total $95 M (including $8 M contingency), which will be expended over a 15 month pre-production design, construction and commissioning period;
- Sustaining Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs are estimated to be $220 M (including $2 M contingency). Sustaining costs are expended in operating Years 1 through 13; and
- Closure Costs – includes all costs related to the closure, reclamation, and salvage value, post operations. Closure costs total $12 M after salvage credits. Closure costs are primarily incurred in Year 14, with costs extending into Year 15.

All capital estimation for the Madsen project has been completed to an accuracy range of -15%/+15% which represents an Association for the Advancement of Cost Engineering (AACE) Class 3 estimate.

The capital cost estimate (CAPEX) is summarized in Table 1-4.

Table 1-4: Summary of Capital Cost Estimate

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
Mining	31.2	209.2	240.4
Site Development	0.8	-	0.8
Mineral Processing	17.4	-	17.4
Tailings Management	4.3	7.9	12.1
Site Services	17.5	0.5	18.0
Project Indirects	6.1	-	6.1
EPCM	7.0	-	7.0
Owner's Costs	2.6	-	2.6
Subtotal	**87.0**	**217.6**	**304.5**
Contingency	8.1	2.1	10.2
Closure	-	16.8	16.8
Salvage	-	-4.2	-4.2
Total Capital Costs	**95.1**	**232.2**	**327.3**

Source: JDS (2019)

1.12.2 Operating Cost Estimate

The operating cost estimate (OPEX) in this study includes the costs to mine, process ore to produce doré, and general and administrative expenses (G&A). These items total the operating costs and are summarized in Table 1-5. The total operating unit cost is estimated to be $223/t processed.

The target accuracy of the operating cost is -10%/+15%. No allowance for inflation or contingency has been applied to operating costs.

Table 1-5: Summary of Operating Cost Estimate

Operating Costs	$/t milled	LOM ($M)
Mining	168.80	592.8
Processing	32.30	113.3
G&A	21.80	76.7
Total	**222.90**	**782.9**

Source: JDS (2019)

The main OPEX assumptions are outlined in Table 1-6.

Table 1-6: Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.100
Total operating load	MW	6.55
Demand load (all facilities in year 5)	kWh/t processed	9.87
Diesel cost (delivered)	$/litre	1.072
LOM average workforce (including contractors, excluding corporate)	employees	400

Source: JDS (2019)

1.13 Economic Analysis

1.13.1 Main Assumptions

An economic model was developed to estimate annual cash flows and sensitivities of the Madsen Project. All costs, metal prices, and economic results are reported in Canadian currency ($CDN) unless stated otherwise.

Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

This Technical Report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting cash flows as part of this study. The mill head grades are based on sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, to obtain major equipment or skilled labour on a timely basis, or to achieve the assumed mine production rates at the assumed grades may cause actual results to differ materially from those presented in this economic analysis.

The reader is cautioned that the gold prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The price of gold is based on many complex factors and there are no reliable methods of predicting the long-term gold price.

Table 1-7 outlines the LOM summary and the basis for the economic analysis while Table 1-8 highlights the economic assumptions.

Table 1-7: LOM Summary

Parameter	Unit	Value
Ore Processed	Mt	3.51
Mill Average Daily Production	kt	0.8
Mill Average Annual Production	Mt	0.29
Average Gold Mill Grade	g/t	8.97
Gold Contained	koz	1,013
Gold Recovered	koz	970
Gold Recovery	%	95.8
Average Gold Production	koz/year	79
Initial Capital Cost	$M	95
Sustaining Capital Cost	$M	232
Life of Mine Capital	$M	327

Source: JDS (2019)

Table 1-8: Economic Assumptions

Off-site Costs and Payables	Item	Unit	Value
Payables for Doré	Gold	%	99.97
Doré Refining Costs	Gold	US$/payable oz	0.38
Transportation Costs		US$/payable Au oz	1.35
Royalties		% NSR	-

Source: JDS (2019)

Table 1-9 outlines the metal prices and exchange rates used in the economic analysis.

Table 1-9: Net Smelter Return Assumptions

Assumptions	Unit	Value
Au Price	US$/oz	1,275
FX Rate	US$:C$	0.75

Source: JDS (2019)

1.13.2 Results

The economic results for the Project, based on the assumptions outlined above are presented in Table 1-10.

Table 1-10: Economic Results

Parameter	Unit	Pre-Tax Results	After-Tax Results
$NPV_{0\%}$	$M	536	383
$NPV_{5\%}$	$M	353	247
IRR	%	42.9	35.9
Payback Period	Production years	3.0	3.4

Source: JDS (2019)

1.13.3 Sensitivities

A simple sensitivity analysis was performed to determine which factors most affect the Project economics and is discussed in Section 23. Each variable evaluated was tested using the same sensitivity values, although some may be more likely to experience significantly more fluctuation in value over the LOM (i.e. CAPEX versus metal prices). The confidence attributed to each variable in this study does not factor into the sensitivity analysis, the inter-correlation between certain variables, and for this reason is considered a simplistic approach to determine which variable would most affect the economic results of the Project.

Sensitivity analyses were performed on metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 15% independently while all other variables were held constant. The Project shows the most sensitivity to metal price and head grades. The results of the sensitivity analyses are shown in Table 1-11.

Table 1-11: Sensitivities Analyses

Variable	After-Tax $NPV_{5\%}$ (M$)			Pre-Tax $NPV_{5\%}$ (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Metal Price	125	247	368	177	353	529
Mill Head Grade	125	247	368	177	353	529
OPEX	304	247	190	436	353	271
CAPEX	287	247	207	394	353	313

Source: JDS (2019)

1.14 Conclusions

Results of this Feasibility Study demonstrate that the Madsen Project warrants development due to its positive and robust economics.

It's the conclusion of the QPs that the Feasibility Study (FS) summarized in this technical report contains adequate detail and information to support a Feasibility Study analysis. Standard industry practices, equipment and design methods were used in this FS and, except for those outlined in this section, the report's authors are unaware of any unusual or significant risks, or uncertainties that would affect project reliability or confidence based on the data and information made available.

Risk is present in any development project. Feasibility engineering formulates design and engineering solutions to reduce risk common to every mining project, such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, political risks, environmental and social risks, and labour sourcing. Associated project risks are deemed manageable; and identified opportunities can provide enhanced economic value.

The recommended development path is to continue efforts in obtaining the required environmental permits to start construction and to continue dewatering the underground workings in advance of mine development, while concurrently advancing key activities that will reduce and de-risk the project execution schedule.

1.14.1 Risks

The most significant risks are summarized below:

- **Surface Geotechnical Conditions** - There is a potential for additional foundation improvement measures to be required at the Tailings Pond Dams to meet stability Factors of Safety. There is a potential for additional overburden excavation at the critical stability footprint area of the MRMF which could lead to additional associated costs. This risk can be managed by completing additional geotechnical investigations and studies on those areas identified in the FS ahead of developing these areas.
- **Underground Geotechnical Conditions** – Less favorable ground conditions than modelled may be encountered in the 8 Zone. Squeezing or closure around drifts may be more dramatic or frequent than anticipated requiring slower mining rates and increased ground support and maintenance requirements. The risk can be managed by additional geotechnical drilling and laboratory testing. It is recommended that this study be carried out as soon as possible within the 8 Zone hanging wall and mineralized zone.
- **Electrical Load & Supply Availability** - Adequate electricity supply from Hydro One is a concern in Northwestern Ontario and while the recent capacity study completed by Hydro One shows adequate power will be available, the initial capacity of 7 MW is very close to the estimated operating demand load. The completion of the upgrade of the Pickle Lake section of transmission line by 2022, and additional power capacity required for the hoisting remains an area of uncertainty for the Madsen Project. The risk will be managed by additional power supply from generators to mitigate any potential power shortages but could come with some additional operating cost.

- **Shaft Conditions** - Conditions in the shaft could be more deficient than anticipated which could increase rehabilitation costs and/or delay getting the shaft operational. Delay in shaft hoisting would be mitigated by increased truck haulage albeit at a higher operating cost. An increased dewatering rate would allow additional time for assessment and mitigation / rehabilitation.
- **Historical Backfill Failure** - Saturated backfill from mined stopes could become unstable and fail. Backfill failures pose a health and safety risk, could increase operating costs to clean-up and cause mine schedule delays. The risk can be managed by increased dewatering ahead of when existing levels are required to allow old stopes to decant and assessments with drones and/or probe holes from safe areas well in advance of accessing or mining activities.
- **Ore Body Complexity** - The complexity of the ore body could potentially lead to increased mining dilution. Grade control and proper mining execution will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs. Definition drilling in advance of mining to assist stope design, particularly in long hole stopes will mitigate the risk.

1.14.2 Opportunities

The most significant opportunities are summarized below:

- **Underground Water Management** - Flows from the Madsen Underground Mine are currently being managed in the TMF all year round, with an assumption that winter discharge is not possible. This results in an accumulation of water in the TMF, during winter months. A WTP with a large design treatment rate is therefore required to manage the surplus water in the TMF. Managing the Madsen Mine underground dewatering flows outside the TMF would considerably reduce the surplus volumes of water to be treated and reduce the design treatment rate of the WTP. This would lead to significant savings in Project Capital Costs. Additional water quality studies are ongoing to investigate the possibility of releasing the underground flows bypassing the TMF and WTP.
- **Power Unit Cost** – Comparable Hydro One customers in the region have been able to achieve significant reductions in their electricity costs by closely managing their electrical load through applying for Northern Industrial Rebate Program (NIER), reduction in Global Adjustment (GA) Payments and Shifting Peak Demand to Off-Peak Hours. These three mechanisms could reduce the unit power rate and ultimately operating costs.
- **Mined Stope Buffer Zones**- Known high grade mineralization has been modelled within the 5 m buffer zones around mined stopes. This mineralization has been removed from the Mineral resource. Further drilling and geotechnical study could potentially lead to some of this material being reclassified as Mineral Resources and if proven economic as Mineral Reserve.
- **Resource Conversion & Expansion** - Through additional exploration drilling, there is potential opportunity to convert Inferred Resources to Indicated Resources and to discover additional mineralized zones. If successful this could increase production rates or extend mine life. There is no certainty that all or any part of the Inferred Resources will be converted to Indicated Mineral Resources or converted into Mineral Reserves.

- **Ore Sorting** - The deposit may be amenable to ore sorting. If successful, ore sorting would remove external and planned dilution increasing the grade to the process plant. Further physical properties study is required.

1.15 Recommendations

Based on the Feasibility Study results, it's recommended to advance the Project to construction and development, and then production. The recommended development path is to continue efforts in obtaining the required environmental permits to start construction and continue dewatering the underground workings in advance of mine development, while concurrently advancing key activities that will reduce and de-risk the project execution schedule. Associated project risks are deemed manageable; and identified opportunities can provide enhanced economic value.

The project exhibits robust economics with the assumed gold price, currency exchange rates, and consumables pricing. Value engineering and recommended fieldwork should be advanced in conjunction with preparation of permit amendments / applications to de-risk the construction schedule and minimize costs.

From the identified project risks and opportunities, the following activities are noted as critical actions that have the potential to strengthen the project and further reduce risk:

- Complete additional diamond drilling to potentially convert Inferred Mineral Resources into Indicated Resources, and potentially Reserves;
- Continue underground dewatering;
- Confirm TMF and MRMF foundation design parameters with additional site investigation, complete TMF foundation assessment and optimize the water balance and water management strategy;
- Develop a full closure plan for the project based on the final design configurations;
- Conduct initial physical properties test work and assess the viability of employing ore sorting technology as a method of rejecting low grade plant feed and increasing head grade in the process plant; and
- Investigate potential for purchasing used equipment to reduce project capital costs.

2 Introduction

2.1 Basis of Technical Report

JDS was commissioned by Pure Gold to carry out a NI 43-101 Feasibility Study (FS) and Technical Report for the Madsen Gold Project, which is based around the Madsen Mine, currently in Temporary Suspension and owned by Pure Gold in the Red Lake mining district of Northwestern Ontario.

This report presents the results of the FS, in accordance with the Canadian Securities Administrators' National Instrument (NI) 43-101 and Form 43-101F1, collectively NI 43-101, guidelines. Two previous technical reports were prepared by Nordmin Engineering Ltd. (Nordmin) for The Project: a Preliminary Economic Assessment (PEA) dated October 27, 2017, and a re-stated PEA and an initial resource report dated January 29, 2018. Both technical reports were filed on the System for Electronic Document Analysis and Retrieval (SEDAR).

Pure Gold Mining Inc. is a junior exploration and development company listed on the TSX Venture exchange with its head office located at Suite 1900, 1055 West Hastings Street, Vancouver, British Columbia, Canada.

2.2 Scope of Work

This technical report summarizes the work of several consultants with the scope of work for each company listed below, which combined, comprises the total Project scope.

JDS Energy & Mining Inc. (JDS):

- Establishing an economic framework for the FS;
- Mine engineering, design and scheduling;
- Geotechnical recommendations for underground mine design;
- Development of conceptual flowsheet, detailed flowsheets, specifications and selection of process equipment;
- Design oversight related to site infrastructure, access road, power line, plant facilities and other ancillary facilities;
- Estimating mining, process plant, G&A and site services OPEX and CAPEX;
- Establishing gold recovery values for doré production on-site;
- Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses;
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks; and
- Developing and compiling the technical report and integrating sub-consultant report sections.

Ginto Consulting Inc. (Ginto):

- Mineral Resource Estimate.

Equity Exploration Consultants Ltd. (Equity):

- Describe geology, mineralization, exploration, drilling, sample quality control & analysis.

Knight Piésold Ltd (KP):

- Tailings management facility and mine rock management facility design;
- Overall project water balance; and
- Water management, including design of ditches, channels and ponds for storm water controls.

Nordmin Engineering Ltd. (Nordmin):

- Development of process plant detailed engineering / material take-offs (MTOs) associated with the processing facilities;
- Surface and underground electrical power distribution design; and
- Shaft and hoisting system design.

MineFill Services Inc. (MineFill):

- Tailings and hydraulic backfill fill characterization; and
- Hydraulic fill plant and distribution system design.

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this report are specialists in the fields of mining engineering, geology, metallurgy, exploration, Mineral Resource and Mineral Reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in Pure Gold and neither are insiders, associates, or affiliates. They are independent of Pure Gold. The results of this 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 Pure Gold and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions / associations. The QPs are responsible for the specific report sections as follows in Table 2-1.

Table 2-1: QP Responsibilities & Site Visits

Qualified Person	Company	QP Responsibility / Role	Site Visit	Report Section(s)
Michael Makarenko, P.Eng.	JDS	Project Management, Environment & Permitting, Mineral Reserve Estimate, Mining Methods, CAPEX, OPEX, Economic Analysis	December 7, 2017	1.1, 1.2, 1.7, 1.8, 1.11-1.15, 2-3, 4.3, 4.4, 12.7, 15, 16 (except 16.3, 16.8), 19-21, 22 (except 22.4), 24, 26 (except 26.2, 26.3.1, 26.4, 26.5, 26.8.2), 27-30
Michael Levy, P.E.	JDS	Underground Geotechnical	February 7-9, 2017 March 14-15, 2018	16.3, 26.3.1
Kelly McLeod, P. Eng.	JDS	Metallurgy, Recovery Methods	December 7, 2017	1.5, 1.9, 12.6, 13, 17, 22.4, 26.2, 26.4, 26.8.2
Richard Boehnke, P.Eng.	JDS	Infrastructure	April 17-19, 2018	1.10, 18 (except 18.9-18.12), 23, 26.5 (except 26.5.8-11)
Marc Jutras, P. Geo.	Ginto	Mineral Resource Estimate	August 30, 2017	1.6, 14
Darcy Baker, P. Geo.	Equity	Geology, QA/QC	Numerous times since 2014, most recently February 11-15, 2019	1.3, 1.4, 4 (except 4.3, 4.4), 5-11, 12.1-12.5, 25
Daniel Ruane, P. Eng.	KP	TMF & MWRF design, Water Management	December 7, 2017 July 16, 2018	12.8, 18.9-12, 26.5.8-11
David Stone, P.E.	MineFill	Hydraulic backfill	No site visit	16.8

Source: JDS (2019)

David Stone, QP for hydraulic backfill, did not personally inspect the site as it was not deemed critical for his work.

2.4 Sources of Information

This report is based on information collected by the QPs during site visits and on additional information provided by Pure Gold throughout the course of JDS's investigations. Other information was obtained from the public domain. JDS has no reason to doubt the reliability of the information provided by Pure Gold. This technical report is based on the following sources of information:

- Discussions with Pure Gold 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 Pure Gold;
- Previous studies completed on the Project;

- New testwork completed during the course of this study by Pure Gold or by the QPs or their designates; and
- Additional information from public domain sources.

2.5 Units, 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., ounces (oz.) for the mass of precious metals, US gallons per minute (gpm) for water flow rates).

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

Frequently used abbreviations and acronyms can be found in Section 30.

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.

3 Reliance on Other Experts

The QPs opinions contained herein are based on information provided by Pure Gold and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are listed below, along with the extent of their involvement and sections of the report to which their input applies.

- AJ MacDonald M.A.SC., P. Eng. – Vice President, Operations Integrated Sustainability Consultants Ltd.
    - Water treatment plant design.
- Wentworth Taylor, CPA, CA – Principal Accountant with W.H. Taylor Inc., Public Accounting Corporation
    - Provided input to the cash flow model related to Ontario and Canadian taxation requirements summarized in Section 24.4.
- Alan Martin, P. Geo., R.P. Bio. – Principal, Lorax Environmental Services Ltd.
    - Geochemistry, hydrogeology and environmental & permitting.

The QPs used their experience to determine if the information relied upon was suitable for inclusion in this technical report and adjusted information that required amending. The QPs take responsibility for the work provided by the experts listed above.

4 Property Description and Location

4.1 Mineral Tenure

The Madsen Gold Project comprises a contiguous group of 251 mining leases, mining patent claims and unpatented mining claims covering an aggregate area of 4,648 hectares in northwestern Ontario (Figure 4-1). 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. Claim data is summarized in Table 4-1 and shown in Figure 4-2.

Figure 4-1: Madsen Gold Project Location Map

Source: Pure Gold (2019)

Pure Gold owns 100% of all mining leases, patents and unpatented claims comprising the Madsen Gold Project. 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. None of the royalties described apply to tenure covering the current mineral resource statement.

Unpatented mining cell claims confer title to hard-rock mineral tenure only, and claims must be converted to leases before mining can take place. Their boundaries are defined by the new provincial grid physical except where encumbered by pre-existing lease and/or patent claim boundaries. Annual assessment work valued at $400 per cell claim unit ($200 per boundary cell claim) must be carried out to maintain unpatented mining claims in good standing. Significant work credits exist on these claims or adjacent patent claims which can be used to renew the unpatented cell claims. Prior to conversion to cell claims in 2018 under new mining regulation, the physical location of all claims posts for unpatented claims were located on the ground and surveyed (Stechishen, 2018).

Patented mining claims (“patents”) confer fee-simple rights to hard-rock mineral tenure and allow extraction and sale of minerals. Most of the Pure Gold 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. Typically, boundaries of mining patents are defined by legal surveys done prior to patenting. Patents do not require assessment work but are subject to an annual Mining Land Tax of approximately $4/ha.

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 an annual rent of $3/ha.

Table 4-1: Tenure Data

Claim No.	No. of Claims	Area (Ha)	Type
Madsen Mine			
PAT-7791 - PAT7826	61	1151	Patented
11509A	1	18	Patented
12527A	1	19	Patented
PAT-8993 - PAT-8995	3	53	Patented
MLO-13528	1	15	Patented
Grouping Total	67	1256	
Starratt - Olsen			
PAT-28016 - PAT-28036	21	330	Patented
PAT-28038 - PAT-28051	14	282	Patented
12881A – 12882A	2	30	Patented
12642A – 12644A	3	55	Patented
Grouping Total	40	697	
Russet			
PAT-7668 - PAT-7681	14	258	Patented
Grouping Total	14	258	
My-Ritt			
PAT-7501 - PAT-7502	2	39	Patented
PAT-7505 - PAT-7510	6	103	Patented
Grouping Total	8	142	
Newman-Heyson			
PAT-48726 - PAT-48745	20	386	Patented
MLO-10670 - MLO-10671	2	20	Patented
Grouping Total	22	406	
Aiken*			
PAT-8158 - PAT-8193	36	666	Patented
20586A – 20587A	2	63	Patented
Grouping Total	38	729	
Nova Co			
PAT-9013 - PAT-9020	8	149	Patented
Grouping Total	8	149	
Hager			
124250	1	6	Unpatented
135653	1	14	Unpatented
140530	1	14	Unpatented
188266	1	3	Unpatented
194127	1	0	Unpatented
216940	1	2	Unpatented
231394	1	7	Unpatented
263367	1	2	Unpatented
303646	1	18	Unpatented
LEA-107157	1	51	Leased
Grouping Total	10	117	
Derlak			
PAT-8024 - PAT-8034	11	219	Patented
Grouping Total	11	219	
Ava			
PAT-7839 - PAT-7857	19	291	Patented
Grouping Total	19	291	
Killoran			
LEA-109514	1	108	Leased
LEA-109622	1	98	Leased
Grouping Total	2	206	
Mills			
PAT-7827 - PAT-7838	12	178	Patented
Grouping Total	12	178	
Grand Total	**251**	**4648**	

Source: McMillan (2018)

Table 4-2: Summary of Royalty Agreements Affecting Madsen Tenure

Claim No.	No. Claims	Royalty Holder	Royalty
18728, 18729, 19367, 19368, 19687, 19688, 19720, 20169–20171, 20585–20588, 20585A–20588A, 21273, 21274, 21275–21278, 21280, 21281, 21316–21318, 21316A, 21378, 12726–12728, 12820–12824, 19181, 19182, 19235–19237, 19328	44	Franco-Nevada Corporation	1% NSR to a maximum of C$1 million
18728, 18729, 19367, 19368, 19687, 19688, 19720, 20169–20171, 20585–20588, 20585A–20588A, 21273, 21274, 21275–21278, 21280, 21281, 21316–21318, 21316A, 21378, 12726–12728, 12820–12824, 19181, 19182, 19235–19237, 19328	44	Canhorn Mining Corporation	1% NSR to a maximum of C$1 million
13060–13062, 13069, 13241–13244, 13255, 13554, 13659, 13660, 13068, 13082–13084, 13254, 13475–13477, 407, 408, 456–461, 1444–1452, 1476	38	Sandstorm Gold Ltd.	0.5% NSR
13060–13062, 13069, 13241–13244, 13255, 13554, 13659, 13660, 13068, 13082–13084, 13254, 13475–13477	20	Franco-Nevada Corporation	1.5% on first 1M oz-equiv; 2% on production beyond first 1M oz-equiv
407, 408, 456, 457, 458–461	8	My-Ritt Red Lake Gold Mines Ltd	3% NSR
K 1445–1452	8	Camp McMann Red Lake Gold Mine Ltd.	3% NSR
12746–12756	11	Fechi Inc.	3% NSR, 1% purchasable for C$1M

Source: Pure Gold (2018)

4.2 Surface and Other Rights

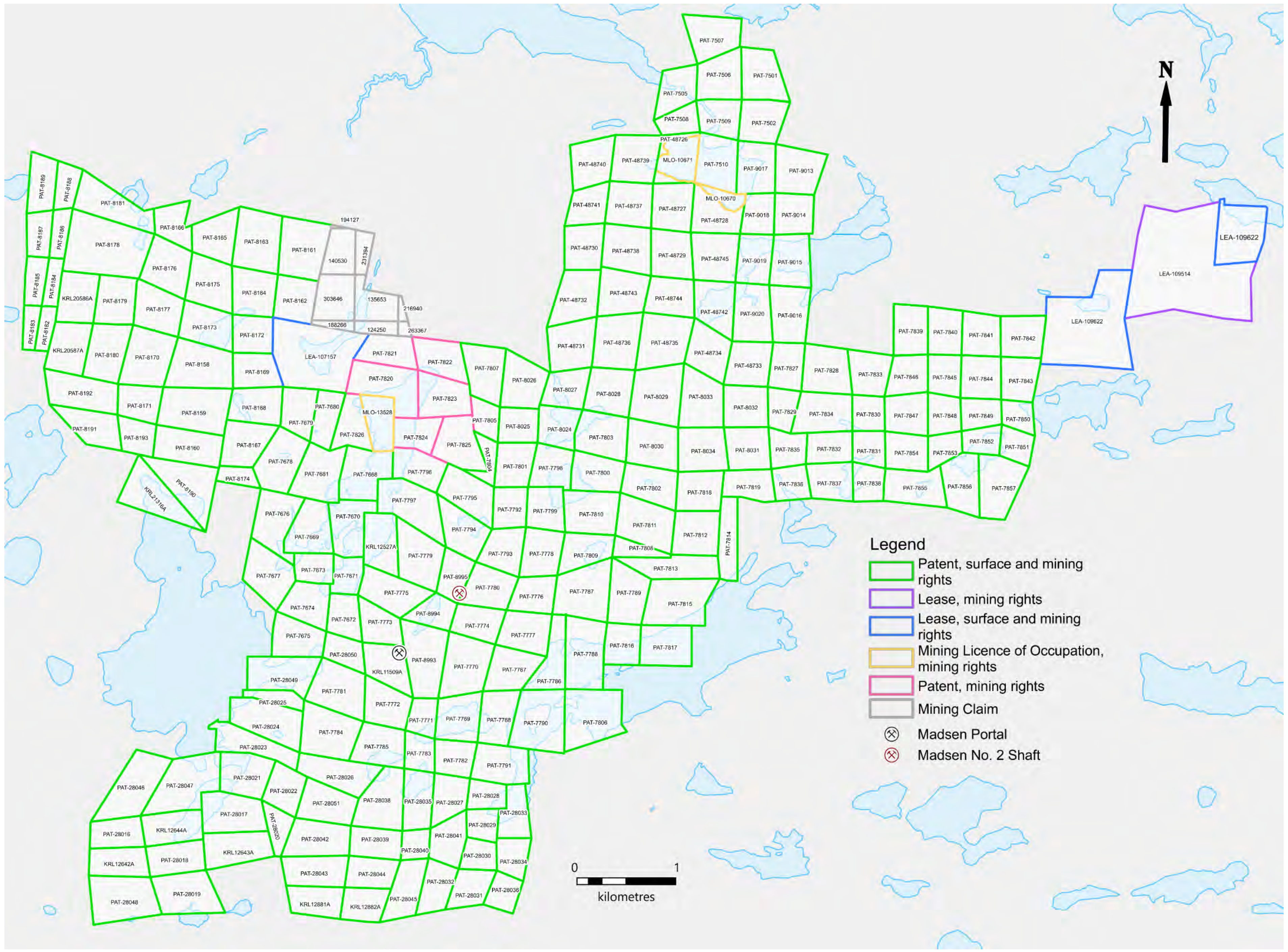
Table 4-3 shows surface rights ownership for Madsen Gold Project claims, patents and leases. Pure Gold owns surface rights as indicated in the table. Where Pure Gold does not hold surface rights they are predominantly held by the Crown, as administered by the Province of Ontario. Timber rights are reserved to the Crown and water rights are held for the public use. A First Nations owned timber company has obtained a timber harvest licence covering the northwestern portion of the property and has been granted access by Pure Gold through an agreement. A single trapping tenure is held over the entire property and Pure Gold maintains good relations with the tenure holder. A local outfitter has a Bear Management Area licence over the Property which requires consent of the landowner for access. 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. Claims	Disposition Type
1445–1452, 407–408, 456–461, 11502–11509, 12521–12529, 12601–12605, 12638–12648, 12658–12669, 12673–12684, 12704–12706, 12726–12728, 12730, 12746–12756, 12758–12760, 12764–12766, 12820–12824, 12836–12838, 12858–12866, 12875–12883, 12921–12922, 12953–12955, 12963–12965, 13024, 13060–13061, 13068–13069, 13082–13084, 13241–13244, 13254–13255, 13475–13477, 13554, 13659, 13660, 16672–16673, 18728–18729, 18778, 19181–19182, 19223–19226, 19235–19238, 19247–19254, 19278–19281, 19306–19313, 19367–19368, 19428–19430, 19684–19688, 19719–19720, 19788, 20169–20171, 20585–20588, 21273–21278, 21280–21281, 21316–21318, 21378, 11509A, 12527A, 12642A–12644A, 12881A–12882A, 20585A–20587A, 21316A, 2890	229	Patent, surface and mining rights
50992, 50993, 51018, 51019, 51020, 51021, 51287, 51288, 51289, 51290	10	Lease, surface and mining rights
4229, 4231, 4902	3	Crown retained surface rights
13062, 38093	2	Licence of Occupation, surface and mining rights
47990, 47991, 47992, 47993, 47994, 47995, 47996	7	Lease, mining rights only
36016, 36017, 36018, 36019, 38091, 38092, 38094	7	Patent, mining rights only

Source: Pure Gold (2018)

Figure 4-2: Madsen Gold Project Tenure Map



Source: Pure Gold (2018)

4.3 Environmental Liabilities

Pure Gold 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 2014. Pure Gold has also undertaken, at its own expense, a site cleanup that has seen significant amounts of waste removed from the site, off-site tire and metal recycling, derelict building removal, PCB storage site decommissioning and closure, security and road improvements and revegetation of some of the disturbed areas. Reopening of the mine will require an update of the closure plan and closure bond reassessment. These actions will facilitate final closure by allowing Pure Gold to fund further progressive reclamation of the Project site, as well as areas within the Madsen town site. (Vendrig, 2019).

4.4 Permitting

Pure Gold has worked to maintain the permits that existed for the Madsen Mine under previous operators. As the project has advanced, operational enhancements and regulatory changes require some of the permits to be updated. Permit status has been confirmed with both ENDM and Ministry of the Environment and Climate Change (MOECC) and the following permits and authorisations are in good standing:

- **Permit to Take Water (0202-AHJL45):** This permit was updated in 2017 and allows for the pumping of 6.5 million litres of water per day from the mine workings. No pumping is currently being undertaken.
- **Advanced Exploration Closure Plan:** In 2016, the Madsen Portal Advanced Exploration area was moved from Temporary Suspension to Advanced Exploration status to facilitate underground access for feasibility level studies. The closure plan for these activities was accepted by the regulators along with additional closure funding for the Advanced Exploration program closure. This plan is in good standing and closure funding can be rolled over into a new mining operation or mine closure if a new operation does not proceed.
- **Advanced Exploration Closure Plan Amendment:** In 2018, Pure Gold requested an amendment to the Madsen Portal Advanced Exploration Closure Plan to allow for the taking of a bulk sample. This amendment was acceptable to ENDM and local First Nations and required nominal additional closure funding. This plan is in good standing and closure funding can be rolled over into a new mining operation or complete mine closure.
- **SAR Exemption and Benefit Program Under Clause 17(2)c of SARA for Endangered Bats:** Myotis species bats were encountered in the underground portal area in 2017. A permit allowing Pure Gold 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 of 2017. The permit is in good standing and MNRF has accepted two annual reports, 2017 and 2018, under this permit.
- **Registered Hazardous Waste Generation Site:** As required by the Ontario Environmental Protection Act Regulation 347, Pure Gold maintains its registration as a hazardous waste site, primarily focussed 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. This permit is renewed annually and is in good standing for 2019.

- **PCB Storage Site Closure:** Pure Gold has decommissioned and closed a legacy PCB storage site established by a previous operator and met all conditions set by MOECC. A closure confirmation letter has been received from MOECC.

The following existing permits are currently in good standing but would require updating due to process changes or regulatory changes:

- **Environmental Compliance Approval (ECA) Industrial Sewage Works Permit:** Pure Gold currently has an Amended Certificate of Approval for Industrial Sewage Works (0047-7V9PW9) that allows for the release of suitable quality water from the tailings facility. This ECA allows Pure Gold 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 conducted and annual reports are presented to MECP. The permit is in good standing.

  Changes in the effluent regulations since the mine and mill facility last operated as well as the water management described in the Project Definition will require an updated Industrial Sewage Works Certificate of Approval. Pure Gold is in the process of updating this ECA for future operations and has undertaken baseline studies focussed on optimizing water resource usage, recovery and recycling and has presented an updated operation general arrangement in the Project Definition. JDS, Knight Piésold and Integrated Sustainability, were contracted to provide tailings and process water treatment designs and Lorax Environmental and Minnow the baseline studies for this work, which is aimed at bringing Pure Gold into compliance with the Metal Mining Effluent Regulation, Ontario Water Resources Act, Regulation 560/94 (Municipal Industrial Strategy for Abatement (MISA) Regulation) and the Metal Mining Effluent Regulation. Designs, baseline reports and predicted water quality modelling have been provided to MECP and ENDM as the first part of updating this permit. Full submission for a new permit application is expected by mid-2019 and a new permit is expected to be issued in 2019 to early 2020.

- **ECA for Air and Noise:** With the proposed use of new equipment and some operational changes to enhance power, energy and water usage, a new air and noise ECA is required. DST Consulting and Aeroacoustics were contracted to provide the baseline study for this work and a new permit application was submitted in 2018. A new ECA for Air and Noise is expected to be issued in 2019.

- **Mine Closure Plan, ENDM:** A Mine Closure Plan was submitted to ENDM on May 24, 1995 with Closure Plan Amendments in July 2011 and April 2014. Current funding is considered to be adequate for closure of the current mine in temporary suspension. Pure Gold has also submitted closure plans for the Madsen Portal Advanced Exploration. It is anticipated that funding from these plans will be rolled into a further Amended Madsen Mine Closure Plan in advance of mine reopening and that additional closure funding may be required for reopening.

  It should also be noted that the mine is currently in a state of Temporary Suspension and considerable site cleanup has been undertaken by Pure Gold outside of the closure plan funding. Given the new Project Definition which is focussed on progressive reclamation and the considerable effort that has been made to clean up the site at Pure Gold's own cost it is anticipated that the 2019 Madsen Mine Closure Plan will only require limited additional funding.

- **Permit to Mine, ENDM:** A Notice of Project Status was received and acknowledged by ENDM on April 24, 2007. A new notice of Project Status will need to 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:** Any construction/relocation of a transmission line for work on Crown land or for in-water works.
  - **Plans and Specifications Approval:** For construction of dams or berms, including those associated with tailings facilities and/or new ponds and ditches.
  - **Forest Resource License:** Annual license for clearing of any Crown merchantable timber.
  - **Aggregate Permit:** Aggregate Resources Act For extraction of any aggregate for dam construction.
  - **Leave to Construct:** For approval to construct a transmission line.
  - **Notice of Construction:** Notice is required before any contractor or construction activities take place.
  - **SARA Approvals:** Although the area falls within the range of several terrestrial species at risk these have not been encountered on or near the site. FRi has been retained to evaluate habitat suitability and to undertake surveys and develop management plans prior to any construction and operational activities associated with mine reopening.

4.5 Other Factors and Risks

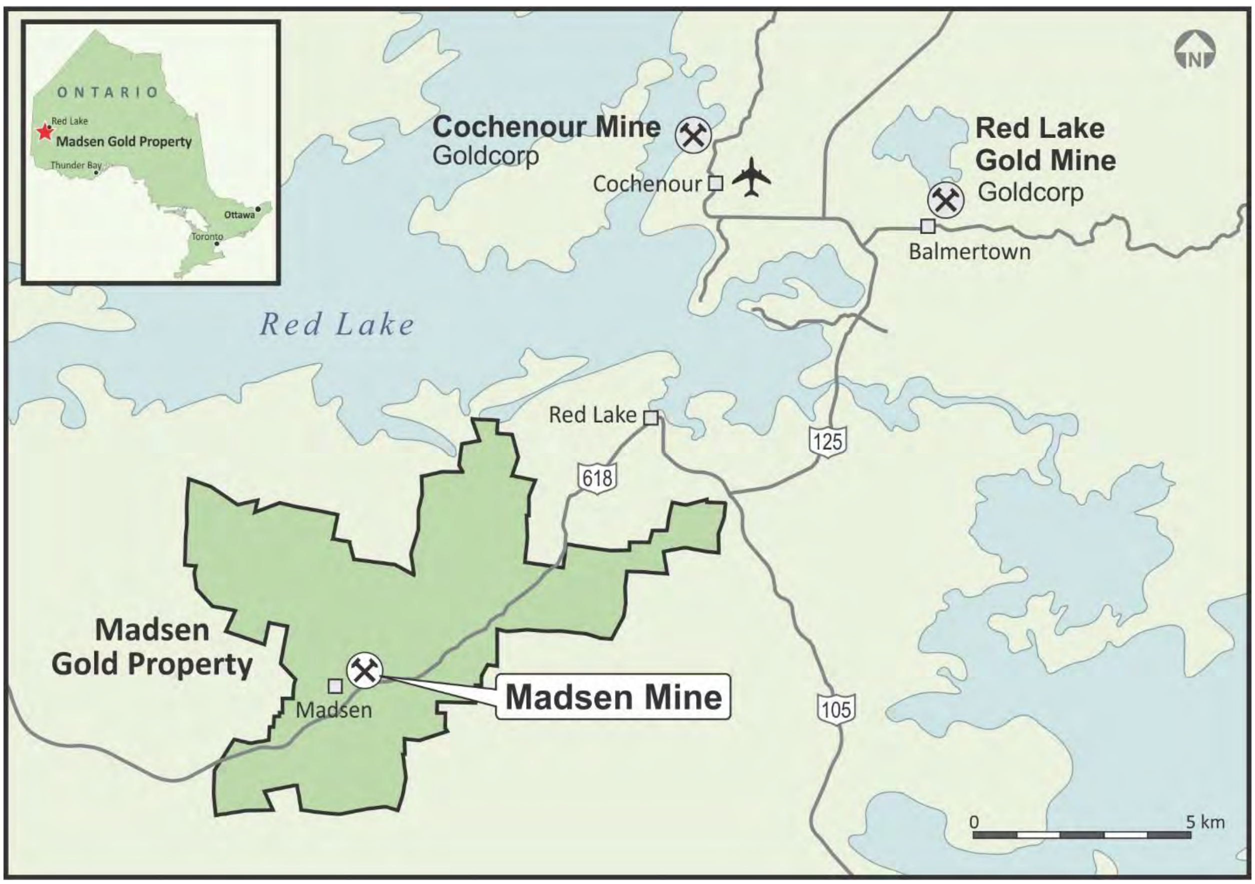
The authors are not aware of any other significant factors and risks that may affect access, title or the right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Madsen Gold Project is located adjacent 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 (Figure 4-1). Red Lake is also serviced with daily flights from Thunder Bay and Winnipeg by Bearskin Airlines.

The Madsen Gold Project 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 Madsen Mine site is 10 km southwest 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.

Figure 5-1: Property Location



Source: Pure Gold (2019)

5.1 Local Resources and Infrastructure

The Red Lake Municipality, population 4,170 (Statistics Canada 2016 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, with production mainly from Goldcorp's 3,100 tonne/day Red Lake gold mine with 1,067 employees in 2019 (Goldcorp, 2019). 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. Many of the Madsen Gold Project staff live in the surrounding area and out of town employees stay in local accommodations in Red Lake.

The Madsen Mine site is serviced by a 44kV Ontario Hydro transmission line. Water is supplied via a municipal treatment facility from nearby Russet Lake. The project site is connected to municipal wastewater service.

5.2 Physiography and Climate

The terrain within the Madsen Gold Project is gentle to moderate (Figure 5-2) 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.

Figure 5-2: Typical Landscape Surrounding the Madsen Gold Project

Note: Aerial photograph looking southeast over Russet Lake with a drilling rig positioned at the Russet South deposit at lower left. Madsen No. 2 shaft and mill complex is shown at upper right.

Source: Pure Gold (2017)

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, which includes a total of about two metres 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. Intensive exploration of the district followed discovery in 1925 of the gold showings which 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 Mine (Lichtblau et al., 2018).

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

The Madsen Gold Project can be divided into five claim groups with separate histories of mining and exploration prior to amalgamation with the Madsen 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; and
- 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).

Exploration conducted by Pure Gold after acquisition of the Madsen Gold Project in 2014 is described in Section 9.0.

Table 6-1: Exploration and Mining History of the Madsen Gold Project

Year	Activity
1925	Gold discovered at Red Lake
1927	First claims staked in Madsen 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 1273 m with 27 levels
1938	Madsen mill facility initiates 36 year 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
1997	Madsen Gold Corp. commences mining and milling at Madsen after moving Donna Lake mill to Madsen. Produces 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. Produces 8,930 ounces gold
1998–2000	Claude drills 230 holes (~21,000 m) on the Madsen Gold Project
1999	Claude mines and mills from the Madsen shaft until October 1999
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
2014	Laurentian, renamed Pure Gold, purchases the Newman-Madsen Property from Sabina and amalgamates it into the Madsen Gold Project. SRK restates 2009 resource

Year	Activity
2014–2018	Pure Gold drills 904 core holes (210,645 m) on the Madsen Gold Project
2016	Nordmin completes positive Preliminary Economic Assessment (PEA) for Pure Gold
2017	Pure Gold opens Madsen 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 Madsen Gold Project
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 at Madsen deposit
2019	JDS completes positive Feasibility Study for Pure Gold

Source: modified after Cole et al. (2016) by D. Baker (2019)

6.1 1925 – 1980

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. Initially, the work 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.

Total recorded production from 1938 to 1999 at the Madsen Mine is 7,872,679 tonnes at an average recovered grade of 9.69 g/t Au. This production accounted for 2,452,388 ounces of gold (Lichtblau et al., 2018). Annual production for the period 1938 to 1976 is summarized in Table 6-2 (excludes data from certain periods).

Table 6-2: Madsen Mine Gold Production (1938 - 1976)

Year	Gold Production (ounces)	Tonnage Milled (tons)	Year	Gold Production[1] (ounces)	Tonnage Milled (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			

Note: Production figures extracted from available Madsen Mine annual reports, 1938 to 1976. n/a = data not available.
Source: Cole et al. (2016)

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 Madsen 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 (Lichtblau et al., 2018).

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 Mineral Holdings

('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 the ST Zone 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 the ST Zone. 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. In 1957, the company name was changed to Starratt Nickel Mines Limited.

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 South

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 between 1953 and 1966 (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.

6.2 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 with three holes intersecting 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 comparable mineralization to the 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 South

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 to 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 the acquisition of the Madsen Mine and surrounding property from Madsen Gold in April 1998, Claude began mining portions of the McVeigh and Austin deposits with access from the Madsen portal and ramp and eventually conducted aggressive exploration 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 Madsen Mine and mill complex was put on care and maintenance in October of 1999. Total recorded production for the Madsen Mine property, 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 (Lichtblau et al., 2018).

Following acquisition of the property in 1998, Claude compiled all historical geophysical, geological, geochemical, and drilling data on the Madsen Gold Project (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 6-3 summarizes the extent and distribution of drilling on the Madsen Gold Project, between Claude's purchase of the property in 1998 and its sale to Pure Gold in 2014.

Table 6-3: Distribution of 1998 - 2013 Drilling on the Madsen Gold Project

Operator		Area						Total
		Madsen	Starratt	Fork	Russet South	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	**545**
	metres	115,598	24,174	51,339	6,774	37,929	13,051	**248,810**

Source: D. Baker (2017)

At the McVeigh West area, approximately 750 m west of the Madsen 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 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 was 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 Madsen project. 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 km2 airborne magnetic/gravity survey were also completed.

From 2001 to 2005, Placer Dome drilled 98 holes (Table 6-3) to test the footwall stratigraphy of the main Madsen auriferous zones within a mafic-ultramafic sequence up-dip of various targets on the property, including: 8 Zone, Starratt, Treasure Box, Russet South, 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 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, 2004). 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 Madsen Gold Project in 2006, they focused mainly on drilling (Table 6-3), historical data compilation, and dewatering and rehabilitation of the Madsen Mine. Mine dewatering commenced in 2007 and was discontinued in late 2013. Claude drilled 108 holes in the Madsen 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 (Lichtblau et al., 2008), 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 which 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 Madsen 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 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 adjacent Balmer Assemblage volcanic rock.

In June 2014 Sabina sold the Newman-Madsen Property to Pure Gold, who amalgamated it into the Madsen Gold Project. Table 6-4 summarizes historical drilling on the former Newman-Madsen Project.

Table 6-4: 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 Madsen 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 ductile structure interpreted from field mapping, airborne magnetic survey data and a metamorphic petrology study. 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-like 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, 2004).

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. In 2009, additional drilling attempted to demonstrate continuity along the interpreted mineralized structures and define the limits of the known mineralization (Cole et al., 2010). Modelled continuity was poor and no resource estimation was completed.

6.4 Previous Mineral Resource and Mineral Reserve Estimates

6.4.1 Pre-NI 43-101 Mineral Resource and Mineral Reserve Estimates

Annual estimates of mineral resource and mineral reserves inventories for the Madsen Mine were undertaken internally by mine staff. Typically, sampling from exposed development and stoping was used to estimate proven reserves, whereas closely spaced core drilling data were used for estimating probable reserves. Indicated and Inferred resources were extrapolated from wider spaced drill holes. Independent audits were undertaken by ACA Howe in 1998 and 1999 (Patrick, 1999). A qualified person has not completed enough work to classify the historical estimates as current mineral resource or reserve and therefore they should not be relied upon. The historical mineral resource and mineral reserve estimates are superseded by the Mineral Resources and Reserves reported in Sections 14 and 15 of this report.

6.4.2 Previous NI 43-101 Compliant Mineral Resource and Mineral Reserve Estimates

In 2008, Claude commissioned SRK Consulting (Canada) Inc. (SRK) to prepare a resource estimate to NI 43-101 standards (Cole et al., 2010). This resource estimate was restated for Pure Gold (then Laurentian Goldfields) in 2014 (Weiershäuser et al., 2014) and used to support a Preliminary Economic Assessment in 2016 (Cole et al., 2016). An updated resource estimate was completed by Pure Gold in 2017 (Jutras et al., 2017) which was used to support an new Preliminary Economic Assessment (Baker et al., 2017) with a different mine design and this Preliminary Economic Assessment was restated along with new resources from the Fork and Russet South deposits (Baker et al., 2018). These mineral resource and mineral reserve estimates are superseded by the Mineral Resource Statement reported in Sections 14 and 15 of this report.

7 Geological Setting and Mineralization

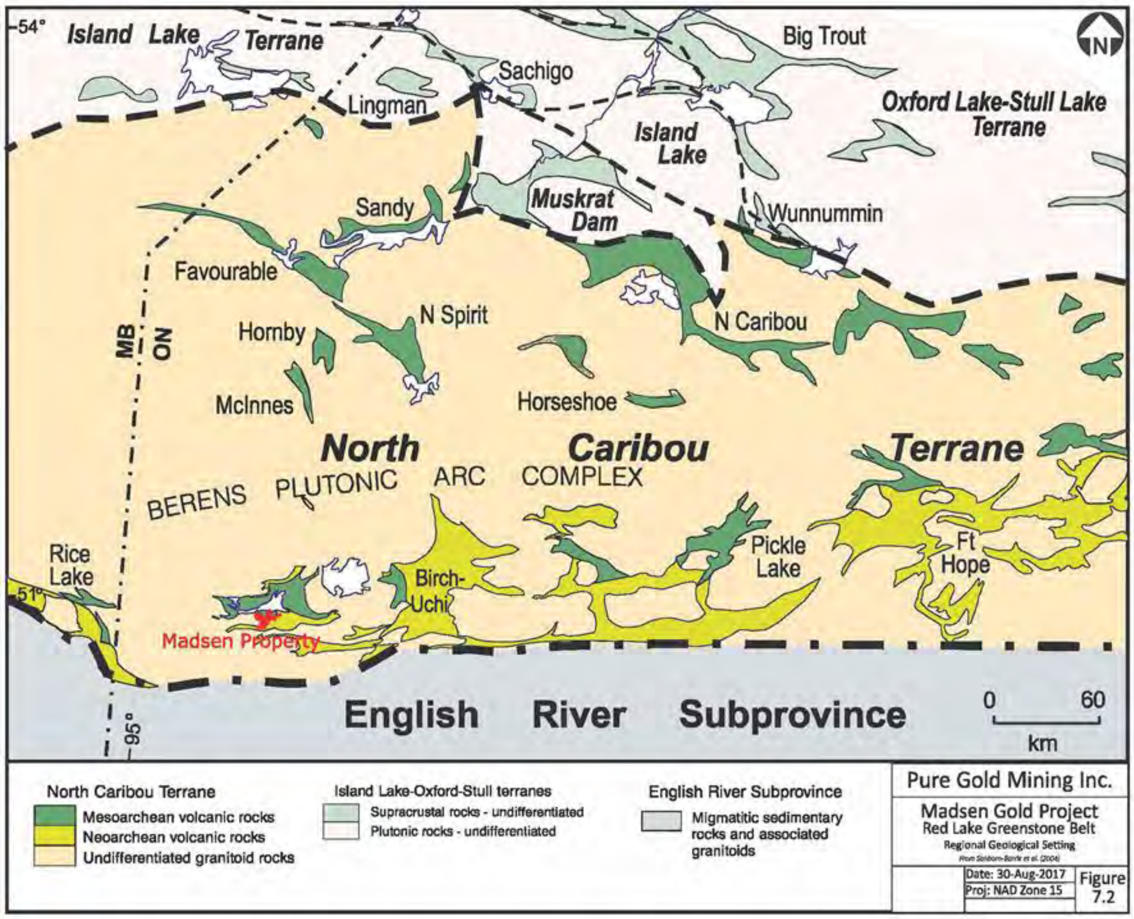
7.1 Regional Geology

The Madsen Gold Project is located within the Archean Superior Province craton of the Canadian Shield. It occupies part of a regional geologic domain characterized by volcanic-plutonic rocks termed the Uchi Subprovince which is bound to the north by the Berens River Subprovince (pluton dominated) and to the south by the English River Subprovince (metasedimentary rock dominated). These three subprovinces amalgamated through tectonic processes at ca. 2700 Ma during the Kenoran orogeny (Stott et al., 1989).

7.1.1 Uchi Subprovince

The Uchi Subprovince (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 rocks, predominately volcanic and subordinate metasedimentary rocks. Continuously trending packages of supracrustal rocks are referred to as greenstone belts. Globally, such Archean belts are responsible for about 18% of historical gold production (Roberts, 1988) and the Uchi Subprovince 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 29.2 million ounces of gold production to the end of 2017 (Lichtblau et al. 2018).

Figure 7-1: Geology of the Uchi Subprovince



Note: Madsen Property is shown as red polygon.
Source: drafted by Equity Exploration Consultants Ltd. (2017) after Sanborn-Barrie et al. (2004b).

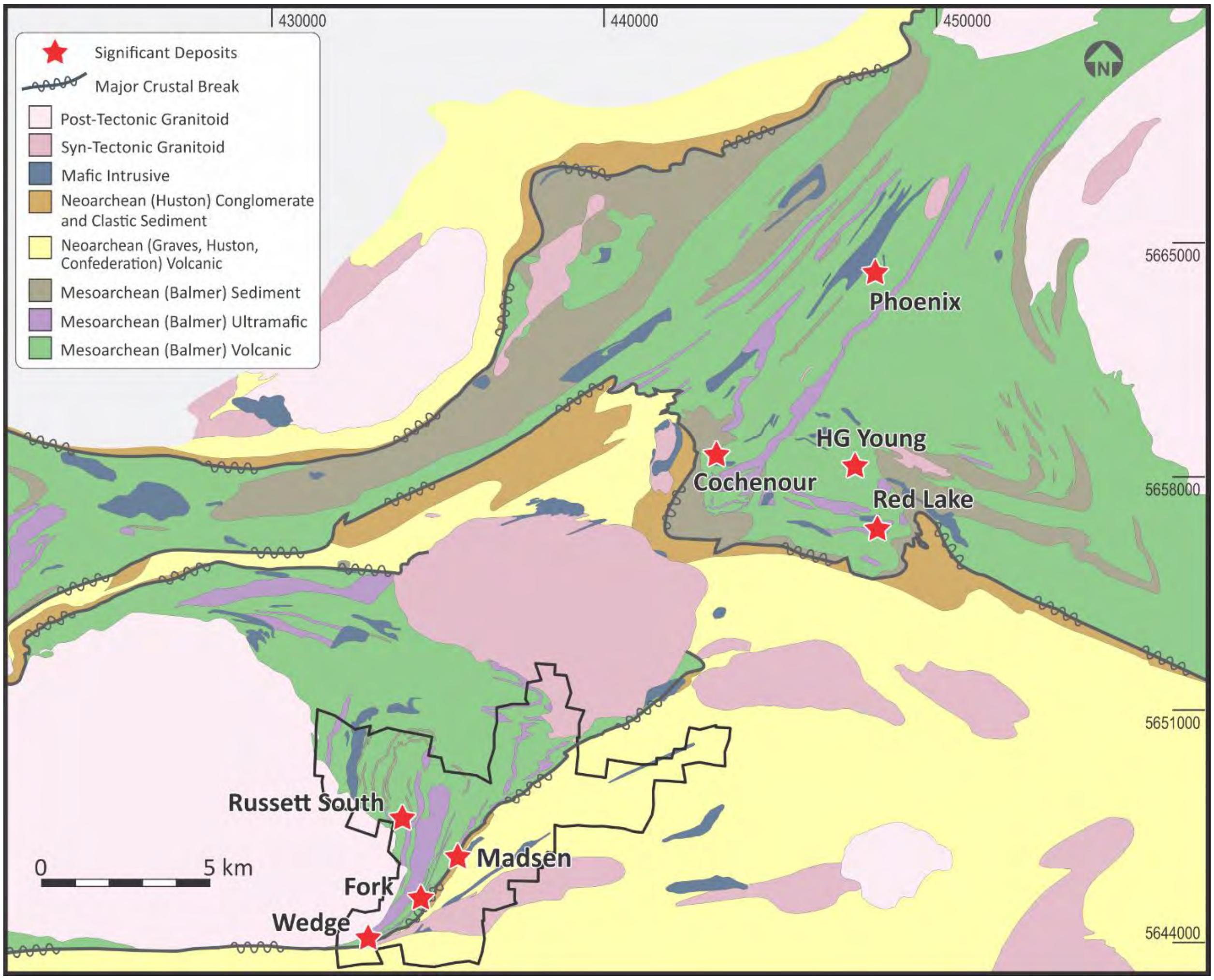
7.1.2 Red Lake Greenstone Belt

The Red Lake Greenstone Belt is approximately 50 by 40 km and comprises a series of ca. 2.99–2.70 Ga supracrustal rocks intervening between three main 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).

7.1.2.1 Balmer Assemblage

The oldest volcanic rocks in the Red Lake greenstone belt comprise predominately tholeiitic mafic and komatiitic ultramafic rocks of the ca. 2.99–2.96 Ga Balmer Assemblage. Significantly, all of the belt's major gold deposits are hosted in the Balmer Assemblage. 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.

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



Note: Madsen Property is shown with black outline.
Source: drafted by Pure Gold (2019) after Sanborn-Barrie et al. (2004b).

7.1.2.2 Ball Assemblage

Underlying the northwestern portion of the Red Lake Greenstone Belt is the ca. 2.94–2.92 Ga Ball Assemblage, comprising a thick sequence of metamorphosed intermediate to felsic calc-alkaline flows and pyroclastic rocks.

7.1.2.3 Slate Bay Assemblage

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).

7.1.2.4 Bruce Channel Assemblage

A thin (<500 m) sequence of calc-alkaline dacitic to rhyodacitic pyroclastic rock, clastic sedimentary rock and banded iron formation is dated at ca. 2.89 Ga 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.

7.1.2.5 Trout Bay Assemblage

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. 2.85 Ga age for this assemblage.

7.1.2.6 Confederation Assemblage

Following an approximately 100-million-year hiatus in volcanic activity, the Confederation assemblage records a time of widespread calc-alkaline volcanism from ca. 2,748–2,739 Ma. A ca. 2,741 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 Madsen area, the strata of the Confederation and Balmer assemblages depict an angular unconformity with suspected 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).

7.1.2.7 Huston Assemblage

Following deposition of the Confederation Assemblage, the Huston Assemblage (deposited approximately between 2,742–2,733 Ma) records a time of clastic sedimentary rock deposition varying from immature conglomerates to wackes. The Huston Assemblage has been compared to the Timiskaming conglomerates commonly associated with gold in the Timmins camp of the Abitibi greenstone belt (Dubé et al., 2003).

7.1.2.8 Graves Assemblage

The ca. 2.73 Ga 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.

7.1.2.9 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.

Post-volcanism plutonic activity is also evident from granitoid rocks such as the McKenzie Island stock, Dome Stock, and Abino granodiorite (2,720–2,718 Ma), which were host to past producing gold mines. The last magmatic event recorded in the belt is from about 2.7 Ga 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 Madsen property at Flat Lake.

7.1.2.10 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 at the Madsen Gold Project has prompted a modified structural history which is discussed in the next section.

The earliest deformation event (denoted as D0) 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 Madsen and within central Red Lake.

The first stage of penetrative deformation (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 F1 folds including a NNE-trending fold that trends through the centre of the Madsen 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 (D1) structures are E to NE-trending D2 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). The timing of

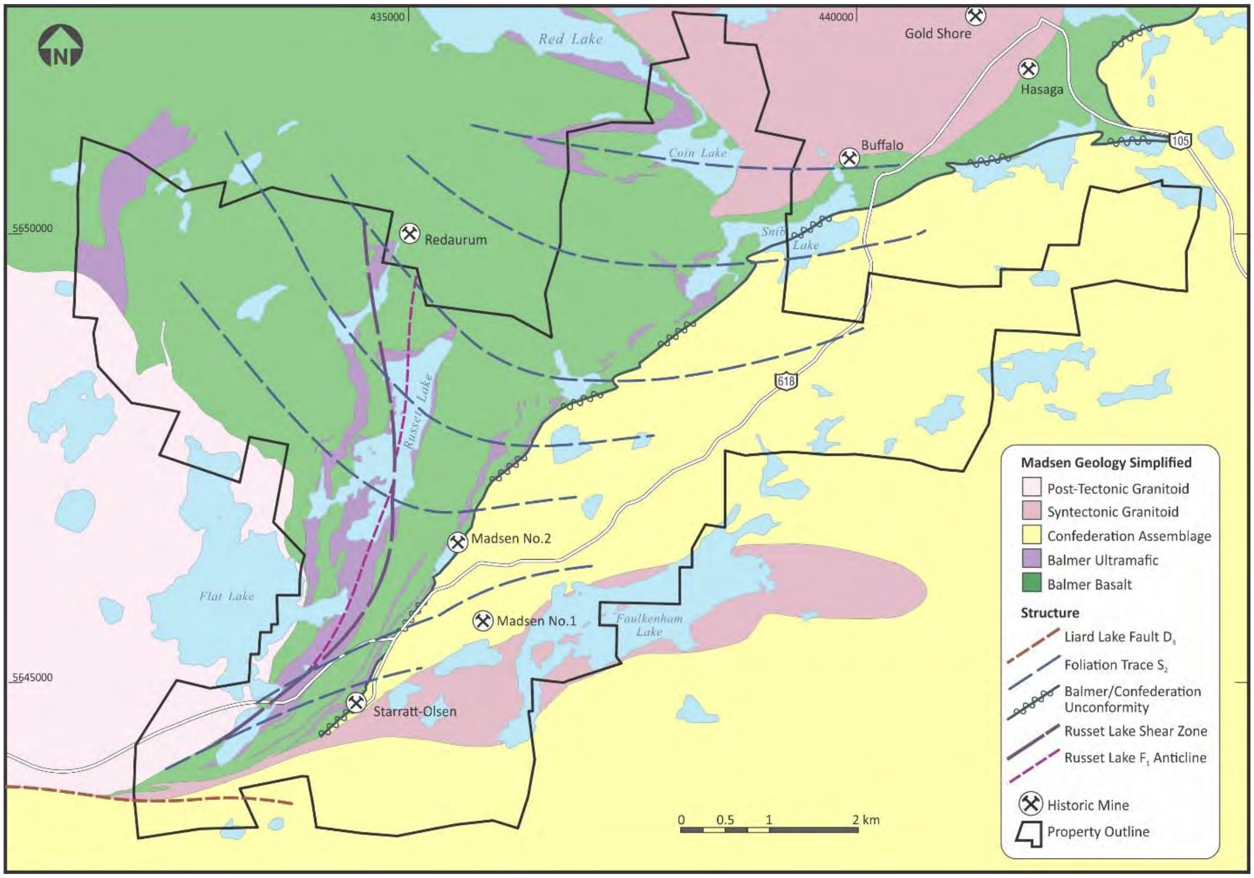
D2 strain is constrained by the ca. 2.72 Ga Dome Stock which exhibits a weak S2 fabric but country rock xenoliths within the pluton also exhibit a penetrative S2. Sanborn-Barrie takes this to mean that the deformation largely predated Dome Stock but continued after stock emplacement which brackets the 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 S2 fabric, D2 was seemingly a protracted event.

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 S2 foliation.

7.2 Property Geology

The Madsen Property is underlain by Balmer, Confederation and possibly Huston Assemblage supracrustal rocks (Figure 7-3). These rocks are cut by a series of plutonic rocks (post-tectonic Killala-Baird batholith to the west and syntectonic Dome Stock to the east) and associated smaller sills and dykes. Although detrital zircon geochronological data suggests that epiclastic rocks near the Madsen Mine belong to the Huston or English River Assemblages (Lichtblau and Storey, 2015; Lichtblau et al., 2012; Sanborn-Barrie et al., 2004b), the sequence of these units based on drilling data suggests that the Huston Assemblage underlies the Confederation Assemblage. This is difficult to reconcile with regional relations, so these units are herein grouped with the Confederation Assemblage until more geochronological data is available.

Figure 7-3: Simplified Geology Map of the Madsen Property



Note: Area showing traces of the main structural features. An F1 axial trace trends along the core of the Russet Lake ultramafic volcanic unit (purple unit underlying Russet Lake). Minor displacement along the Russet Lake Shear Zone likely occurred during D1 and is linked with the gold event at 8 Zone which occurs at depth along this structure. A regional overprinting S2 foliation cuts obliquely across Balmer and Confederation Assemblage rocks and overprints gold mineralization at Madsen and Starratt.

Source: Drafted by Pure Gold (2018) after geology by D. Baker (2016).

7.2.1 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 Madsen Gold Project has been a focus of surface exploration work since the property was acquired by Pure Gold (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 Madsen host stratigraphy, gold-bearing structures and deformation features.

Based on outcrop, underground exposure and drill core analysis, 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. 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) readily developed foliations owing to a bulk chemistry that encourages phyllosilicate (mainly sericite) growth during stress-related recrystallization and metamorphism. As such, use of a qualitative determination of “intensity” of strain must be used with caution because this is a rock composition dependant feature. The focus of Madsen surface work has been on recording structures from outcrop to outcrop with the understanding that a given foliation in Confederation Assemblage felsic volcanic rock will manifest itself much differently in adjacent Balmer basalt across the regional unconformity.

The deformation history of the Madsen Gold Project is here interpreted to include three deformation events: D1, D2 and D3. Identification and interpretation of these structures is not unambiguous everywhere, but these sets of structures seem to show the best continuity across the geological datasets.

D1 deformation is confined to Balmer-aged rocks and equates to the D0 event of Sanborn-Barrie et al. (2004b). Outcrop-scale evidence for this event is sparse because, as described above, the unaltered mafic and ultramafic rocks of the Balmer Assemblage do not readily develop penetrative foliations. The strongest evidence, however, 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 8600 showing; both confirming the overprint of two folding events (F1 and F2). Although no widespread penetrative foliation developed during the F1 folding event, strain was seemingly partitioned into generally axial-planar shear zones such as the Russet Lake Shear Zone, an interpreted ductile shear zone that is concordant with the F1 fold axis which trends along the core of the Russet Lake ultramafic body. Importantly, gold mineralization of the 8 Zone lies within the Russet Lake Shear Zone. Other gold mineralization on the property including Austin, South Austin, McVeigh and Starratt are strongly linked to this generation of structure since they all show similar alteration and mineralization styles. The current interpretation is that these individual, planar deposits all formed within pre-D2, early planar structures, likely forming during D1 deformation.

S1 deformation fabrics are difficult to identify and only locally are S1/S2 overprinting relationships observed. Within these structures, the rock has been strongly overprinted by D2 deformation and metamorphism such that most D1 structures are obliterated. As such, characterizing these structures is difficult because of later strain and metamorphism so it is unclear if these structures behaved in a ductile (i.e. shear zones), brittle (i.e. fault or breccia zones) or a combination of both strain types. What is clear, however, is that these structures and associated gold mineralization pre-date the penetrative regional S2 foliation.

The strongest evidence for these being early structures is their planar geometry and scale as well as their low angle oblique relationship to lithological contacts and the Balmer / Confederation unconformity. This is based on systematic geological interpretation and contradicts early descriptions that suggest the Madsen deposit is stratabound and occurs parallel to the unconformity (Dubé et al., 2000). In detail, the Madsen deposits trend oblique to the unconformity and locally (e.g. the Austin deposit) terminate at the unconformity surface. Significantly for exploration, this opens up a voluminous sequence of prospective host rocks in the footwall of the unconformity. Stated another way, gold prospectively does not decrease away from the unconformity and indeed the high-grade 8 Zone is an example of significant gold mineralization down section – 900 m away in plan view – from the unconformity.

Little evidence exists that Confederation assemblage rocks were affected by F1 folds such as the Russet Lake anticline or other F1 folds in the eastern part of the Red Lake Greenstone Belt (Sanborn-Barrie et al., 2004b). Generally, regional map patterns show Confederation rocks as non-participants in F1 folds. As such, the current interpretation is that D1 predates 2,744 Ma.

The second generation of structural fabric development at Madsen includes a conspicuous, penetrative, regional foliation (S2) which is generally consistent with the D2 structural trends of Sanborn-Barrie et al. (2004b). This foliation has been described as parallel to the Balmer / Confederation unconformity and to represent a major transcurrent, regional shear zone (e.g. Hugon and Schwerdtner, 1988), however, detailed data shows that this fabric consistently transects the unconformity with no displacement of lithological contacts. Minor (10s of metres scale) S-shaped folds are defined by lithological contacts and also by historical outlines of Madsen orebodies on numerous original underground level plans (Horwood, 1940). The S2 foliation is axial-planar to these small folds, linking the S-folds to D2. Numerous small-scale folds of quartz porphyry intrusions and diopside veins within the 2018 McVeigh Zone bulk sample area show a similar, consistent orientation to these larger folds (plunging about -60º towards 110º).

The latest deformation to affect the Madsen Gold Project is localized brittle faulting. Such faults are rare across the property, particularly in the Madsen Mine area but are common at Starratt where they are characterized by metre-scale intervals of fault breccia and fault gouge recovered in recent drill core. These are most likely 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.

Generally, the deformation history outlined for the Madsen Gold Project above is consistent with that for the Red Lake Greenstone Belt (Sanborn-Barrie et al., 2004b). Sanborn-Barrie interprets a tilting-only event to explain the angular relation between Balmer and Confederation Assemblage rocks but the pre-Confederation deformation event appears to be much more significant and involved broad folding (F1 folds of Sanborn-Barrie et al., 2004b) of Balmer rocks across the belt. Early, planar structures host gold mineralization at Madsen and Starratt and may coincide with F1 folding since they are at least locally axial-planar to F1 structures. Significantly, Confederation rocks are not affected by these folds, so the interpretation herein is that D1 predates 2,744 Ma. Therefore, only minor differences between the deformation history of Sanborn-Barrie et al. (2004b) and the working deformation history for the Madsen Gold Project are required. These include the intensity of Balmer-only deformation and the lack of requiring a cryptic, tilting-only D0 deformation as this tilting is accomplished via F1 folding.

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

7.2.2 Balmer Assemblage Rocks

The oldest rocks underlying the Madsen Gold Project 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 lithologies are described below.

7.2.2.1 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 where original textures are preserved, both primary intrusive and extrusive features have been identified. 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. Generally, PRDT is not a host rock for gold mineralization at Madsen. The exception is the 8 Zone which is hosted in quartz veins hosted in a D1 shear zone cutting the Russet Lake Ultramafic. Other major concentrations of gold at Madsen are spatially associated with PRDT sills.

7.2.2.2 Pyroxenite

Medium- to coarse-grained pyroxenite (PXNT) occurs within composite sills with PRDT (Section 7.2.2.1) 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).

7.2.2.3 Iron Formation

Thin (0.1–1 m) iron formation (IRFM) occurs exclusively within the Balmer Assemblage in the Madsen area within rare clastic sedimentary packages or more commonly between individual basalt flows. Three types are recognized at Madsen: 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.

7.2.2.4 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.

7.2.2.5 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 Madsen Property, particularly where D1 shear zones have cut PRDT / BSLT contacts.

7.2.2.6 Gabbro

Dark grey, massive, equigranular, medium- to coarse-grained gabbro (GBRO) cuts basalt rocks and shows relatively high ratios of Mg, Ni and Cr relative to younger Confederation gabbro (O'Connor-Parsons, 2015). Gabbro is not prospective for gold mineralization at Madsen.

7.2.3 Confederation Assemblage Rocks

7.2.3.1 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 Madsen.

7.2.3.2 Intermediate Volcanic

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

7.2.3.3 Quartz Crystal and Lithic Rhyolite Tuff

A quartz crystal-rich lithic-crystal tuff (QPXL) forms the majority of the lowest Confederation Assemblage in the Madsen 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 at Madsen. A sample of QPXL collected near the Madsen Portal was dated ca. 2,741 Ma (Lichtblau et al., 2012).

7.2.3.4 Conglomerate

Locally, a pebble-cobble conglomerate (CONG) demarcates the lowermost Confederation Assemblage, underling the lithic-quartz crystal tuff. However, a similar unit is also found locally elsewhere in the stratigraphic sequence and in the Balmer Assemblage and is described as Huston Assemblage by Sanborne-Barrie et al. (2004b).

7.2.3.5 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.

7.2.3.6 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.

7.2.3.7 Gabbro

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

7.2.4 Veins

7.2.4.1 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.

7.2.4.2 Early Carbonate-Magnetite 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. These veins are overprinted (metamorphosed) by amphibole-bearing veins and are locally nearly completely replaced by diopside (see below). This vein type has a close spatial association with gold mineralization, and although they seemingly predate the gold mineralizing event in detail, broadly they seem part of the same hydrothermal system. The consistent and locally intense metamorphic overprint of these veins resembles a skarn assemblage as suggested by Dubé et al. (2000) but these mineral phases were formed through the metamorphism of a pre-existing carbonate alteration.

7.2.4.3 Diopside Replacement Veins

Light green, massive, coarse crystalline diopside-quartz-amphibole-calcite veins (VNDI) represent the partial to total replacement of early carbonate-magnetite veins (VECB). These veins are highly prospective for gold mineralization, especially along vein margins where VNDI is in contact with altered country rock, quartz veins or quartz porphyry.

7.2.4.4 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.

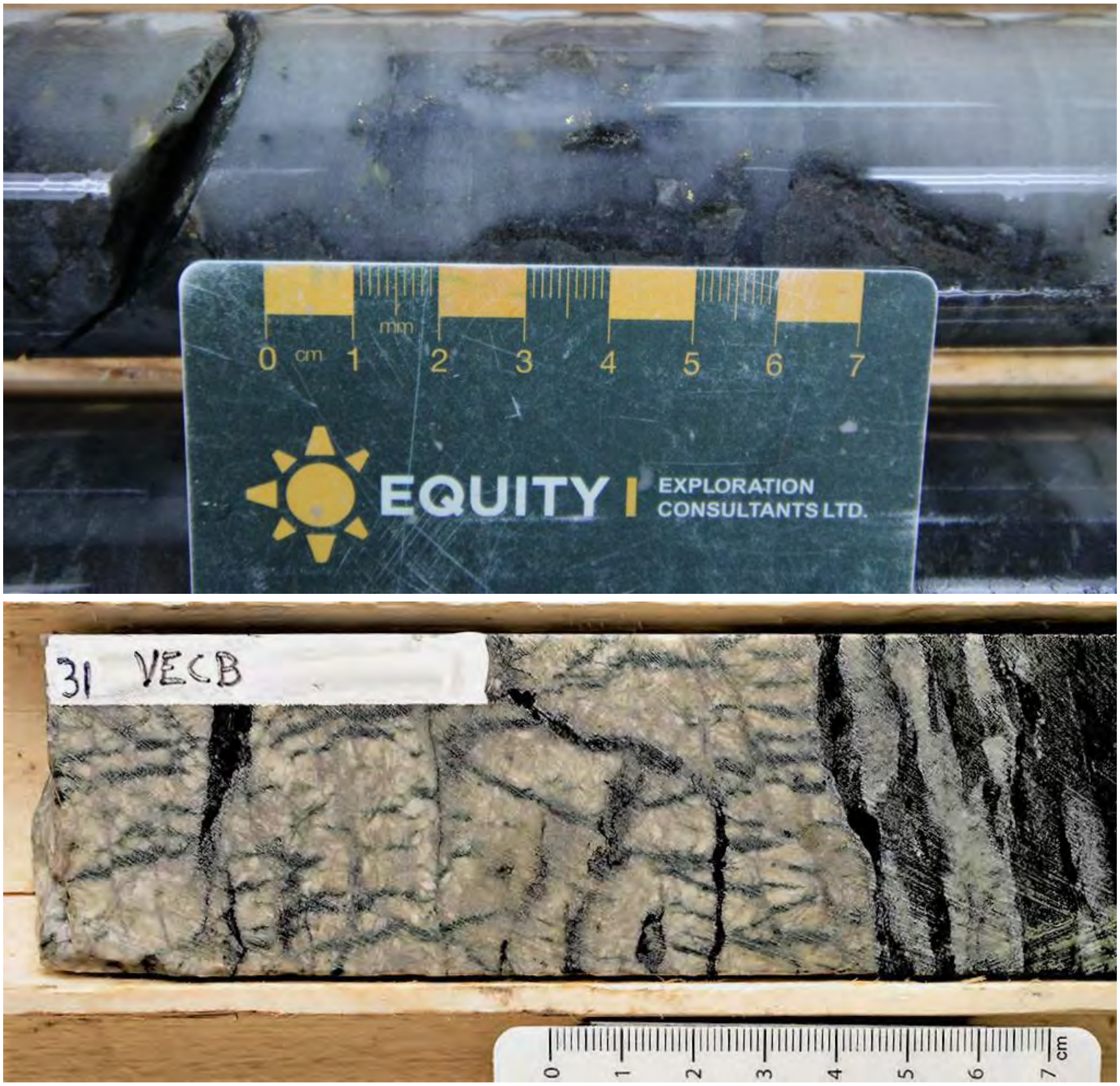
7.2.4.5 Blue-Grey Quartz Veins

Blue-grey to white, massive, recrystallized quartz veins (VBGQ; Figure 7-4) are associated with gold mineralization at both the Russet South deposit and 8 Zone and have only been identified within Balmer rocks. This vein set is folded and/or boudinaged and clearly pre-dates D2 deformation, consistent with a D1 timing for gold mineralization. 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 selvages to these veins. This vein set may relate to more pervasive silicification within the main Madsen deposits.

7.2.4.6 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 S2 foliation so are tectonically late consistent with their more brittle style.

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 with is characteristic of these veins and evidence of an 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 at Madsen 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 (top) and sawn half (bottom) drill core.

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

7.2.5 Metasomatized Rocks

Balmer Assemblage rocks vary from unfoliated and undeformed volcanic rocks with well-preserved fine-scale primary volcanic features (e.g. pillow structures), to mafic and ultramafic rock that has been pervasively altered, deformed and metamorphosed to the point that no primary features are discernible. Such rocks are 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). Two important points need to be made about these metasomatized volcanic rocks.

Firstly, it was recognized that metasomatized, or hydrothermally altered, Balmer rock locally forms coherent, planar units (coincident with suspected early, gold-associated structures as described above) and that it was advantageous for drill targeting to delineate these intervals with codes that highlight this alteration. Strictly, these intervals are probably mostly basalt but the original protolith has been modified to the point that it is inappropriate to log or map this rock as basalt. So, three codes were developed that are defined by secondary mineral assemblages as described in the following sections.

Secondly, 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 indiscernible at Madsen. Regional metamorphism (synchronous with D2 deformation) has overprinted the Madsen gold systems to the degree that the host rocks to the gold-bearing zones are characterized by a seemingly complex mineral assemblage that has grown during regional metamorphism and, arguably, should not be described as alteration. But, because the rock surrounding the Madsen gold systems was altered before metamorphism, an assemblage of abundant metamorphic biotite, garnet 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 a halo surrounding gold mineralization. The three metasomatic rock assemblages identified are described in the following sections.

7.2.5.1 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 unrecognizable (Figure 7-5). These domains of intense alteration and strong foliation overprint are 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.

In the Madsen deposit area, 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. The sulphide content of SAFZ is highly variable but ranges up to 20% pyrite-pyrrhotite-chalcopyrite-arsenopyrite. There is limited to no correlation between sulphide content and gold values. SAFZ typically contains abundant VECB and VNDI veins which are commonly transposed into the main fabric of the foliation.

7.2.5.2 Pervasively Altered Basalt

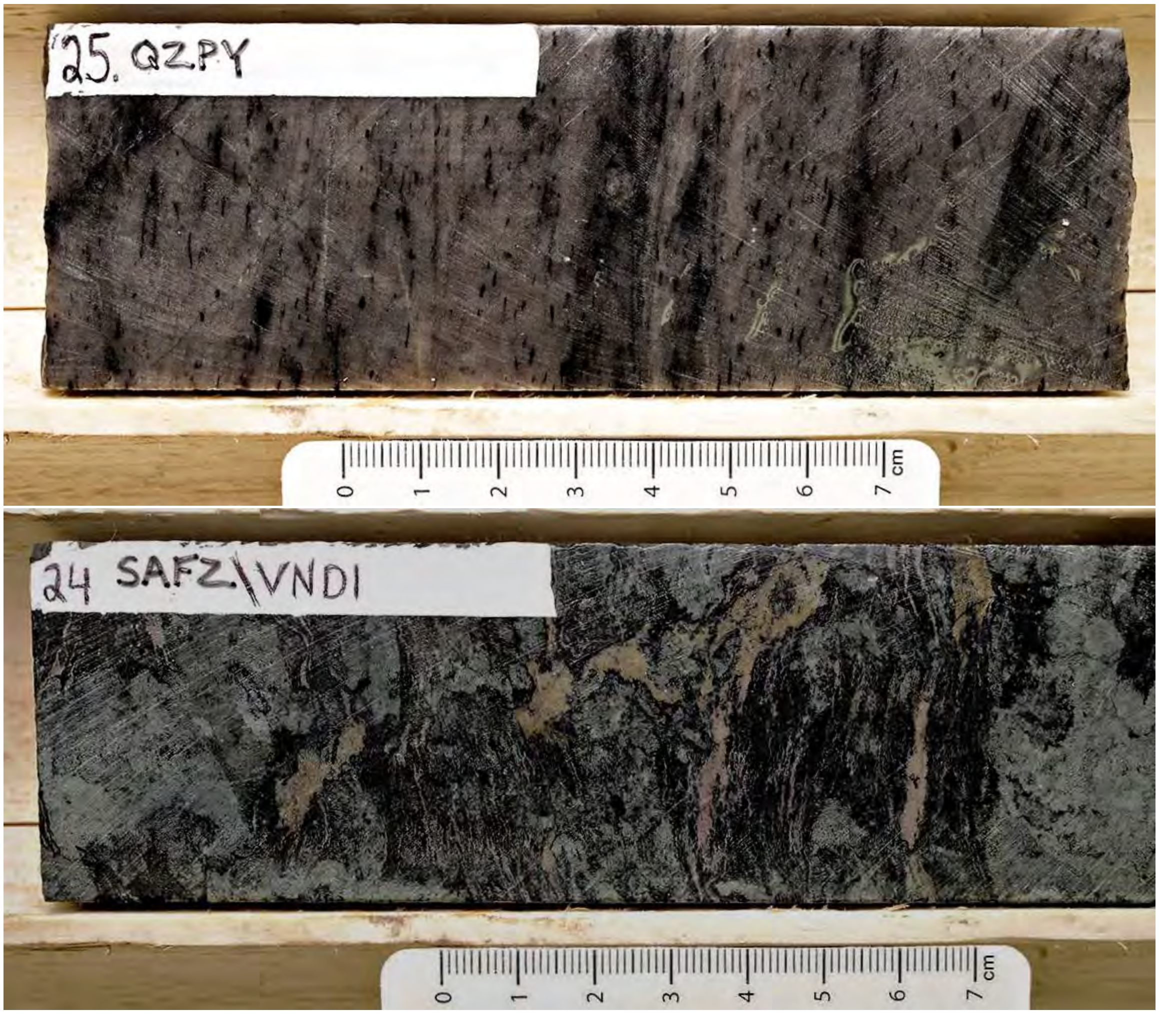
Pervasively altered basalt (BSLA) refers to coherent domains of weakly to moderately foliated, pervasively biotite-amphibole altered rock that is generally interpreted to have a mafic volcanic protolith. BSLA is transitional from the more strongly altered SAFZ and primary textures are locally preserved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

These domains of weak to moderate alteration and foliation represent the marginal envelope of structural corridors that were exploited by both early hydrothermal gold mineralising fluids. Where this unit is proximal to an ultramafic sill or SAFZ unit, it has moderate potential to host gold mineralization.

7.2.5.3 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. Significantly, PRBA represents D1 shear zones that were exploited by gold bearing fluids as evidenced by 8 Zone and the common elevated to significant gold values returned from PRBA samples.

Figure 7-5: Key Rock Types from Madsen



Note: Photographs of two distinct rock types that show close relationship to auriferous zones at Madsen. Quartz porphyry (QZPY, top) is characterized as light to dark grey, fine-grained and weakly quartz-phyric. QZPY bodies are concordant to planar gold-bearing zones interpreted to be early shear zones. Strongly altered and foliated zones (SAFZ, bottom) are characterized by texturally destructive biotite, diopside and potassium feldspar mineral phases that are considered to be resultant from a metamorphic overprint of a more typical carbonate altered basalt. SAFZ is the main gold host at the Madsen deposit. Photos of and sawn half drill core.

Source: D. Baker (2018) from photographs supplied by Pure Gold (2018).

7.2.6 Plutonic Rocks

7.2.6.1 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.

7.2.6.2 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.

7.2.7 Dykes and Sills

7.2.7.1 Intermediate Intrusive

Intermediate intrusive (IINT) are grey, undeformed, fine- to medium-grained dykes that cross-cut both Balmer and Confederation group rocks and cut gold mineralization at Madsen and Starratt. These dykes have sharp, chilled margins and locally tend to strike concordant to the property-wide foliation suggesting they exploited the S2 structural grain. IINT have been dated at ca. 2698 Ma from underground at Madsen and ca. 2696 Ma from the Wedge - CK zone and provide a minimum age for Madsen 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.

7.2.7.2 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 thought to be Proterozoic aged.

7.2.7.3 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 seemingly post-tectonic and therefore are not prospective for gold.

7.2.7.4 Hornblende Feldspar Porphyry

Hornblende-feldspar porphyry (HFPY) dykes are intermediate, dark grey-pink and are common in the Russet Lake area but are unknown in the Madsen 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.

7.2.7.5 Quartz Porphyry

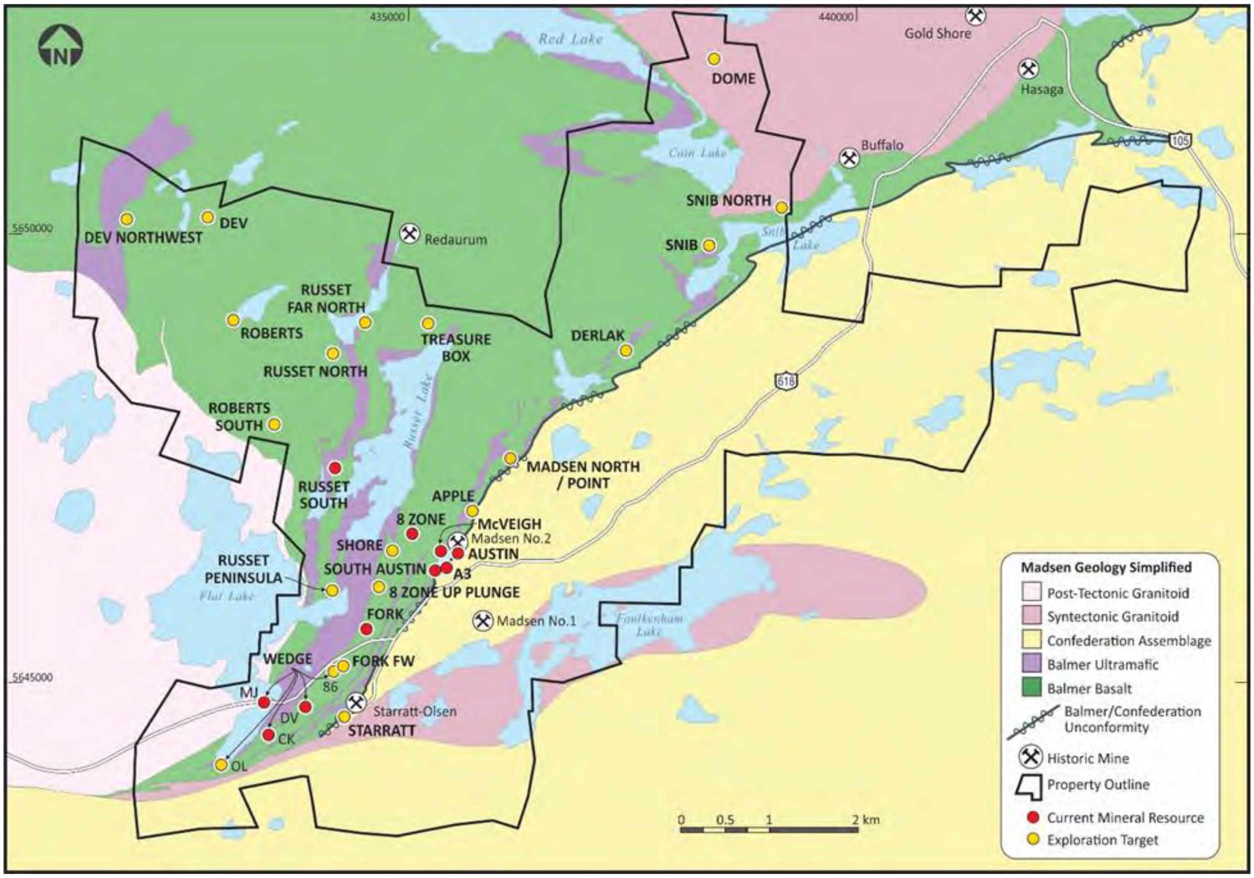
Quartz porphyry (QZPY) refers to a set of felsic, light to medium grey, foliated, quartz-phyric or fine-grained (aphyric) dykes that, significantly, are spatially associated with gold-bearing zones (Figure 7-5). Porphyritic examples contain a few percent rounded, quartz phenocrysts and foliation-parallel 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 Madsen gold event and QZPY dykes typically are concordant with D1 shear zones indicating that these dykes exploited the same early structures that controlled gold-associated hydrothermal systems. Exposures in the 2018 bulk sample area of the

McVeigh Zone show that QZPY dykes are locally tightly folded with the penetrative S2 foliation axial planar to the folds.

7.3 Property Mineralization

The following sections summarize the geology, geometry and style of the significant gold-bearing zones and associated targets present on the Madsen Property (Figure 7-6). A closer view of the projected distribution of the Madsen deposits is shown in Figure 7-7.

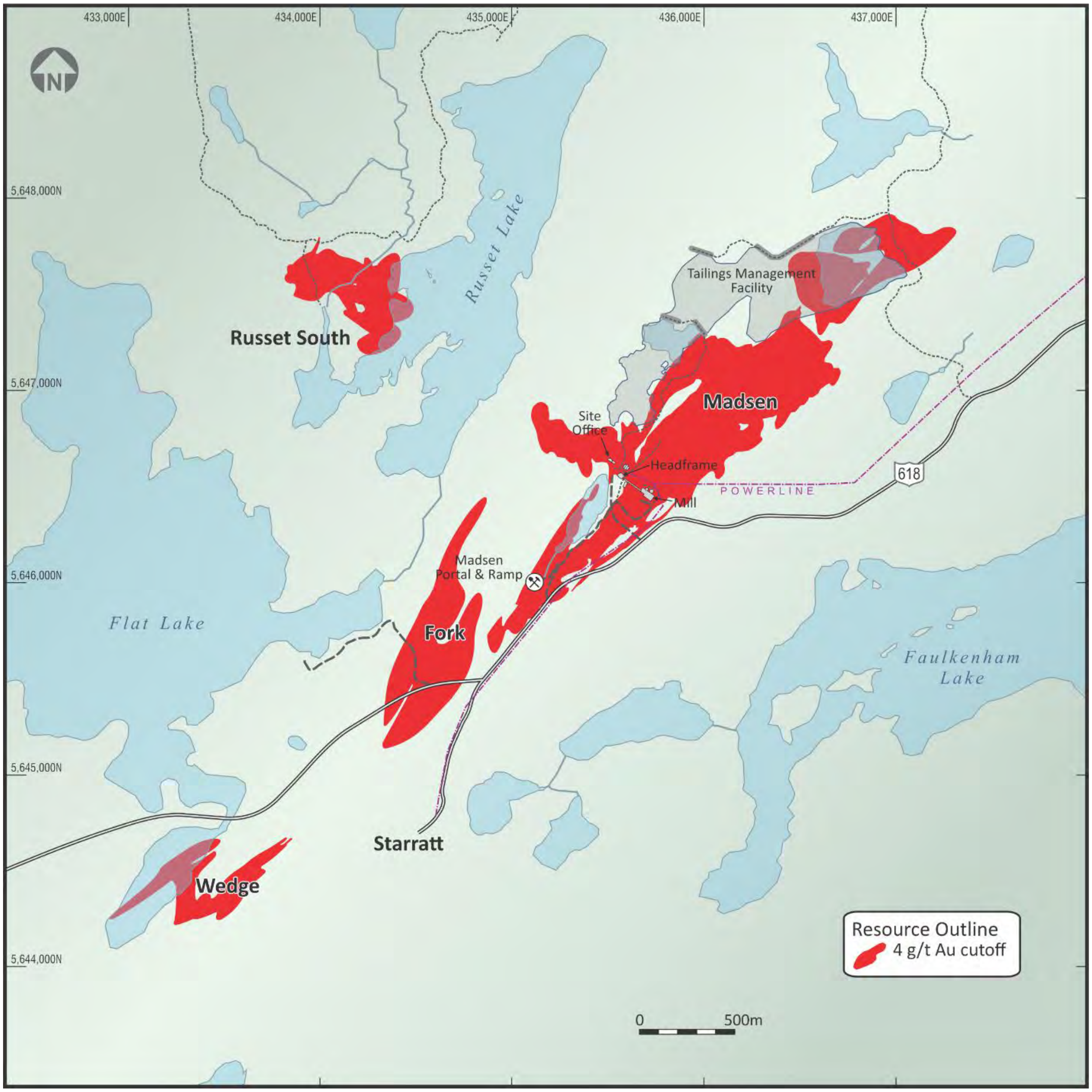
Figure 7-6: Historical Mines, Deposits and Exploration Targets



Note: Summary geological map of the Madsen Property showing the targets of recent exploration work by Pure Gold. Property outline shown by solid black line.

Source: Pure Gold (2018)

Figure 7-7: Generalized Plan Map of Madsen Gold Project Resources



Source: P. Smerchanski (2019)

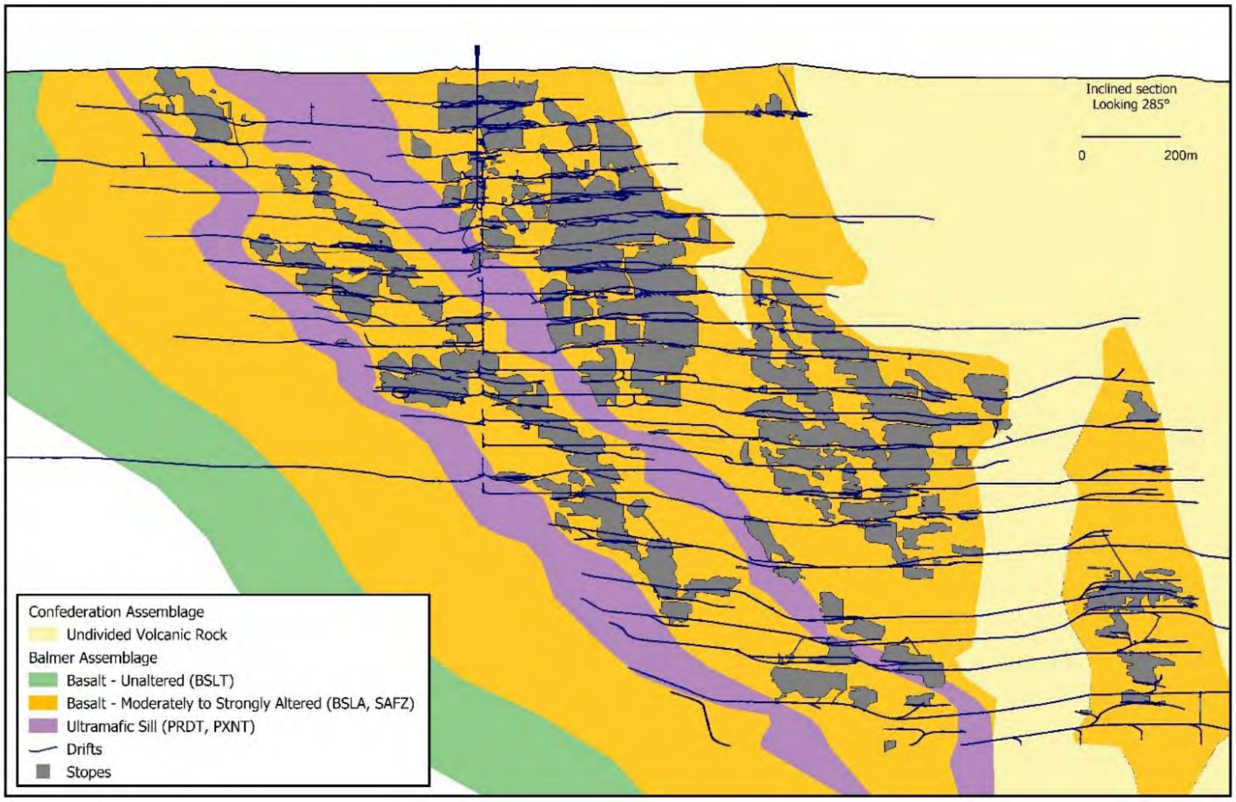
7.3.1 Madsen Deposit – Austin, South Austin (including A3 Domain) and McVeigh Zones

Most historical gold production and most of the current reserves and resources at the Madsen deposit are within the Austin, South Austin, and McVeigh zones which, along with the 8 Zone, comprise the Madsen deposit. At the property-scale, these zones all lie within much broader, kilometre-scale planar zones that are considered early (D1) shear or fault zones. The distribution of gold within these planar structures is controlled principally by the intersection of the structures with basalt/ultramafic lithological contacts (Figure 7-8). A secondary plunge control is locally defined by the intersection of these structures with Confederation Assemblage rocks. These controls are both evident in Figure 7-8: the more dominant (-40^{o}) plunge is defined by the intersection lineation of Balmer ultramafic sills (purple) and the early shear or fault zone. The subordinate, steeper plunge control is defined by intersection with Confederation rocks (yellow) and also locally by small-scale S-shaped folds.

The Austin and South Austin Zones are open down plunge from the reserves. Historical drilling intersected 14.3 g/t Au over 2.0 m in 2011 at 825 metres below past mining in the Austin Zone and drilling in 2017 returned 34.6 g/t Au over 4.3 m at 240 m below past mining in the South Austin Zone. Core review and geological modelling has confirmed alteration and host rock continuity at these depths.

The mineralized structures are oriented at low angles to both the Balmer Assemblage stratigraphy (as defined by narrow metasedimentary units) as well as to the dominant, overprinting regional foliation (S2). The geometry of the ore zones is the result of intersection between the early (D1) mineralizing structures and prospective stratigraphy. Primary lithological control on gold distribution includes proximity to the contacts of an ultramafic unit within the upper several hundred metres of the Balmer Assemblage. Note that the ultramafic rocks are interpreted as intrusive sills and extrusive volcanic units therefore are used as a proxy for stratigraphy. Most gold mineralization is located within approximately 100 m of one of these ultramafic contacts, in rock characterized by strong to intense biotite, garnet and diopside alteration. As such, the linear Austin and South Austin zones occur on the same mineralizing structure, with a poorly mineralized zone of ultramafic rock separating them (Figure 7-8).

Figure 7-8: Inclined Long Section through the Austin and South Austin Zones with Projected Geology

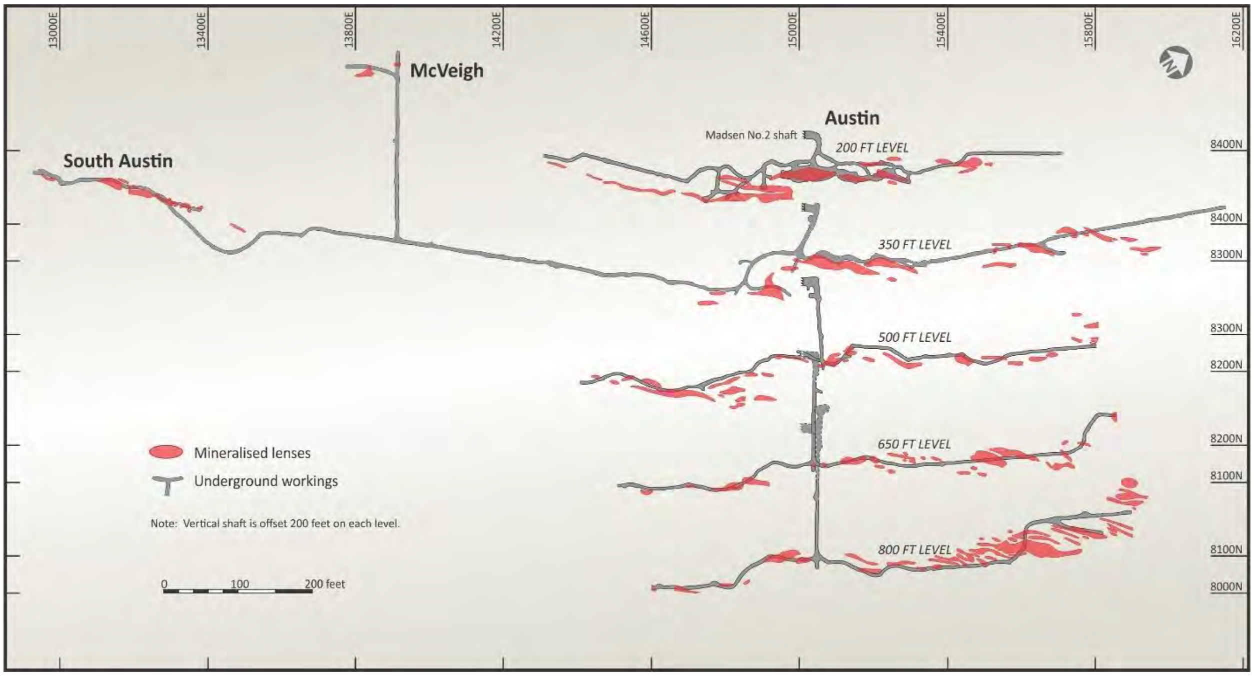


Note: Mined historical stopes (grey) demonstrate gold-bearing zones which clearly show a strong northeast plunge at about 40°. The projected geology shows the strong proximal alteration zones (orange) surrounding high grade gold mineralization (here represented by mined stopes) and, significantly, that 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 Madsen Mine. Note that the McVeigh and 8 Zones lies in the footwall to this long section and is not shown.

Source: Equity Exploration Consultants Ltd. (2019) from drawing by D. Baker (2017)

Viewed in more detail (Figure 7-9), the Austin and South Austin Zones are comprised of a series of tabular gold-bearing zones that are concordant with the penetrative S2 foliation. Locally, gold-bearing bodies show minor s-shaped fold geometries (e.g. the 350 foot Level in Figure 7-9) consistent with small-scale folds mapped at surface and interpreted in geological level plans. These geometries and the transposition of the gold-bearing zones into foliation-concordant bodies is consistent with the overprint of the Madsen deposit by the belt-scale D2 deformation event. Importantly, D2 is not a major shearing event but rather it has superimposed the prominent axial-planar foliation with negligible lateral offset as evidenced by the significant strike lengths of the Madsen deposit zones.

Figure 7-9: Historically Mined Gold-Bearing Shapes from Original Level Plan Maps



Note: Composite level plan maps of Madsen gold-bearing lenses drawn from mining control plans during the earliest periods of mining at Madsen. The gold-bearing zones shown by red shading were demarcated by early mining geologists as high-grade zones. Note that these shapes have two orders of control. At a smaller-scale, gold occurs within a series of concordant, right-stepping lenses that are transposed within the S2 foliation. At a larger-scale, these lenses collectively define planar structures that are continuous for many 100s of metres. Levels are offset for effect.

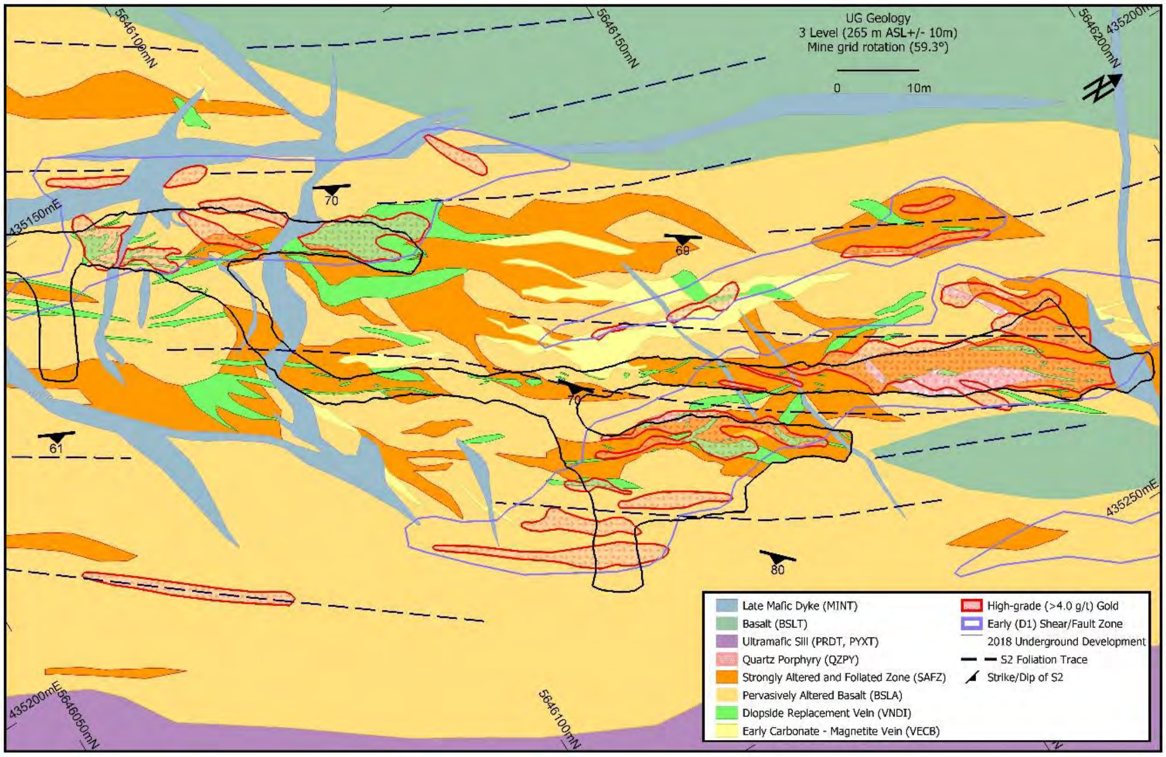
Source: Pure Gold (2018) after Horwood (1940)

The 2018 bulk sample project provided significant, detailed information on a small portion of highly mineralized McVeigh Zone (Figure 7-10) and a test for the geometries and relations described above for the Austin and South Austin Zones. Detailed structural observations and data collection has confirmed the relationships between gold-bearing lenses and the penetrative foliation (S2). High-grade gold lenses form a similar pattern to that recorded on historical level plans whereby individual, tabular lenses show a right-stepping pattern but overall, the zone is continuous across the S2 fabric.

The penetrative S2 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 grid) which is similar to the orientation of map-scale folds interpreted from level plan geological interpretations. The S2 foliation as defined by metamorphic segregation layers is itself locally folded which is likely a result of progressive deformation rather than a superposition of an unrelated deformation event.

Figure 7-10: 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 7-9) that is characterized by high-grade gold-bearing lenses that are aligned and transposed parallel to the penetrative S2 foliation but that overall are contained in planar zones that are oblique to S2.

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

In drill core or at underground face exposures, gold-bearing zones at Madsen are best identified visually by fine (sub-millimetre) grains of free gold. Generally, all high-grade intervals contain visible gold but there are numerous examples of high-grade assays returned from samples in which no visible gold was initially identified on the core surface but has been later explained by gold 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 diopside veining are the best indicators that a given interval is within a high-grade lens within the mineralized structure.

As discussed in Section 8.0, controls on mineralization at Madsen 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 Madsen is an unusual or end-member type of orogenic gold system. For example, Dubé et al. (2000) conclude that Madsen 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 shares many characteristics with typical orogenic gold deposits, including the Red Lake Mine.

7.3.2 Madsen Deposit: 8 Zone

The geology and mineralization style at the 8 Zone is somewhat distinct from that of other known zones within the Madsen deposit. Gold at the 8 Zone occurs within strongly altered and veined peridotite of the Russet Lake Ultramafic (see Section 7.2.5.3 for description of the PRBA unit). By contrast, most gold at McVeigh, Austin and South Austin Zones is hosted within mafic host 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 by 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 which is generally on the order of tens of metres wide. The more intense zones of alteration are marked by near-total replacement by 1–10 cm intervals with 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 has 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 8 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 Madsen Property.

7.3.3 Russet South Deposit

Gold at the Russet South deposit is hosted within folded and/or boudinaged blue-grey quartz veins that are similar to those characteristic of the 8 Zone. At Russet South, 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 hangingwall 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 D2 folds in the Russet Lake area. 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 three resource zones (DV, CK and MJ) and two mineralized zones that remain 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, has shown that these historical zones all lie within a series of northeast-striking D1 shear zones (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 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. Recent 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 selvages 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 (i.e. the early shear zone) are discordant to the main S2 foliation of the host rocks. This is consistent with gold being associated with the earlier, D1 deformation event (Baker et al., 2017).

The OL Zone exploration target lies about 550 m from the edge of the CK Zone resource shape in an area characterized by deformed gold-bearing quartz veins exposed in historical trenches. Recent drilling by Pure Gold at OL Zone intersected strongly altered rock, quartz porphyry sills and gold.

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 South, the Wedge deposit exhibits similar high-level characteristics to the Madsen deposit (same alteration and structural timing), however gold tends to be hosted directly in deformed quartz veins rather than disseminated within intervals of silicified rock, which is more like mineralization at the 8 Zone of the Madsen deposit and the Russet South deposit where gold is hosted within blue-grey quartz veins (VBGQ).

The apparent plunge of mineralization along the D1 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 host D1 shear zones and mafic/ultramafic contacts.

7.3.5 Fork Deposit and Fork Footwall Target

The Fork deposit lies within two concordant D1 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 (the equivalent resource domain is Hanging Wall Domain in Section 14.2.2). This lens occurs along a D1 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 seems to be variously controlled by interaction with thin (~10 m to 20 m wide) ultramafic sills and iron formation in an otherwise basaltic host rock.

The lower lens has been referred to as the Fork Footwall Zone (the equivalent resource domain is Footwall Domain in Section 14.2.2) 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 so controls on gold deposition are unclear since the host rock is generally homogenous. Potentially significantly, the Fork Footwall Zone occurs within the same shear zone that hosts 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 perhaps it is a short lower-order splay.

Both Fork deposit structures are cut by late, discordant felsic, intermediate and mafic dikes as at Madsen. 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 D1 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. 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 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 only been tested by 11 drill holes.

7.3.6 Starratt

Gold mineralization at the Starratt target is very similar to that 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 Madsen whereby mineralized zones occur in planar bodies that cut at low oblique angles across the same ultramafic sills that occur at Madsen. As at Madsen, 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, Starratt and Madsen 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 10 m to 15 m.

Interestingly, since the Starratt and Madsen mines were originally separated by a tenure boundary, the area between these historical mines seems particularly 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 at Madsen.

7.3.7 Dev Northwest

The Dev Northwest Target is an early-stage 700 m by 100 m gold in soil anomaly. Follow-up prospecting and mapping 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. Altered zones range from 1 m to 10 m wide and trend NW-SE. The intersection of these altered zones and the ultramafic sill to the northwest remains a priority target.

7.3.8 Dev

At the Dev Target, a large, D2 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). A program of mechanized outcrop stripping during the 2016 field season (Jones, 2016) followed up on these anomalies. Several zones of altered and mineralized sedimentary rocks cut by significant quartz veins were exposed and grab samples returned up to 59.3 g/t Au. These samples are not considered representative of the outcrops but are considered indicative of prospective veining. The 2016 work demonstrated that gold mineralization is present along chemical and competency contrasts in the Dev area, making it an attractive exploration target with similar characteristics to the Russet South deposit. Further mapping and rock sampling as well as re-logging of historical drill core from Placer Dome was completed at Dev in 2017. VBGQ veins occur at surface and in drill core which is encouraging.

7.3.9 Snib

Historically, this area was part of the Newman Madsen Property acquired by Pure Gold in 2014 and the Derlak property acquired by Pure Gold in 2017 and hosts the Newman Rajah Red Lake occurrence, which has been described as quartz veins occurring in a narrow, easterly-trending, mineralized shear zone (REF). 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 tested the unconformity between the Balmer and the Confederation Assemblages and 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. Prospecting in this area by Pure Gold has returned anomalous gold values associated with disseminated pyrrhotite and pyrite. There is approximately 1000 m of poorly tested strike length between the two drill intercepts noted above.

7.3.10 Snib North

The Snib North Target has a geologic setting analogous to the Madsen deposit. Historical drilling shows of favourable alteration, veining and structural complexity within basalt adjacent to north-east trending ultramafic sills. Strongly foliated basalt cut by quartz veins and granodiorite dykes is exposed in surface outcrops. A surface rock sample of granodiorite returned 3.2 g/t Au and anomalous gold in soil overlies the area.

7.3.11 Madsen North / Point

This 600 m by 300 m target comprises highly anomalous gold in soil values northeast of the Madsen Mine that may be related to drainage downstream from the Madsen historical tailings (Baker and Swanton, 2016). The alteration zone which hosts the Austin Zone, however, extends through this target. Limited drilling completed in 2014 by Pure Gold returned weakly anomalous gold (up to 0.4 g/t Au over 8.0 m) and while no high-grade drill intercepts have been returned from this area, access is complicated by tailings infrastructure which has made adequate drilling difficult. Recent geological modelling indicates that the geology is highly prospective, so further work is warranted.

7.3.12 Treasure Box

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 weakly deformed basalt and gabbro.

Extensive drilling by Placer Dome and Claude delineated 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. Placer Dome and Claude drill holes were re-logged during the 2017 field season. The work added several new gold intercepts to the target area and has helped define alteration in the target area.

7.3.13 Russet Peninsula

The Russet Peninsula target is located east of Flat Lake, along the footwall contact of the Russet Lake Ultramafic. Anomalous gold values from soils and grab samples form a north-south oriented trend spanning >100 m of strike length. Quartz veins hosted by altered basalt were discovered during surface reconnaissance mapping in 2015. Drilling completed by Placer Dome in 2003-2004 intercepted several anomalous gold intervals (e.g. 6.1 g/t over 3.0 m) at 170 m vertical depth below the gold in soils anomaly. Gold intercepts near surface are associated with altered basalt and sheeted quartz porphyry dikes in contact with ultramafic rocks and at depth (>600 m vertical depth) gold intercepts are associated with altered ultramafic rocks (PRBA) and deformed early blue-grey quartz veins (VBGQ). The Russet Lake Ultramafic footwall contact is a prospective target at the Wedge and Russet South deposits, so the presence of alteration and anomalous gold at this location makes it an attractive early-stage target.

7.3.14 8 Zone Up Plunge

8 Zone Up Plunge is an exploration target interpreted from an up-plunge projection of the target setting associated with the 8 Zone at depth along the Russet Lake Shear Zone to the upper (eastern) contact of the Russet Lake Ultramafic towards the Fork deposit footwall domain. Drilling in this area has shown that gold mineralization is similar style to that of the 8 Zone and is characterized by visible gold in deformed blue-grey quartz veins (VBGQ) as well as replacement-style disseminations in silicified wall rock. Soils and rock samples from this target show weakly anomalous gold. The target has not been directly tested by drilling. A fence of drill holes focused on the Shore target 300 m to the north were completed by Claude in 2007 and intercepted narrow intervals hosting VBGQ veins along the Russet Lake Shear Zone.

7.3.15 Shore

The Shore target area is located along the hanging wall contact of the Russet Lake Ultramafic, 200 m north of the 8 Zone Up Plunge target area. The target setting is the same as 8 Zone Up Plunge, Fork and the Wedge deposits east of the Russet Lake Ultramafic (that is, they all occur along the Russet Lake Shear Zone). Mineralization is associated with quartz veins within massive and altered basalts. Surface rock sampling returned several anomalous gold values with a high of 5.3 g/t Au. Historical drill intercepts returned up to 38.1 g/t over 3.0 m.

7.3.16 Apple

The Apple target is located northeast along trend of the Austin deposit. Mineralization is analogous to the Madsen deposit. It is associated with quartz veining and silicification within altered basalts along hanging wall and footwall contacts of ultramafic rocks. Extent of mineralization spans 500 m strike length and is delineated down to 400 m vertical depth. Historical and modern drilling define the foot print of the mineralized zone with sporadic well mineralized intercepts. This is a high priority target as it sits immediately adjacent to Madsen resources and planned underground development.

7.3.17 Derlak

This target is analogous to the Austin Zone, as it occurs within the hanging wall to an ultramafic sill and footwall to the unconformity and Confederation rocks. The target is underlain by anomalous gold in soil samples but historical drilling in the area produced no significant gold intercepts. Core relogging in 2017 identified similar alteration (BSLA) to the Madsen deposit over a strike length of 200 m. Alteration in basalts and structural complexity are documented in surface mapping.

7.3.18 Dome

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.

7.3.19 Roberts

The Roberts target is located along a curvilinear north-striking, east-dipping iron formation on the west side of Robert's Lake. 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 meters to the north and south of the Roberts trenches. 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. Since 2017, Pure Gold has drilled six holes testing directly beneath surface mineralization and the best intercept was 3.7 g/t Au over 5.0 m, including 9.8 g/t over 1.0 m. No testing below 50 m vertical depth has been completed. Localized clusters of anomalous gold in soils and rock grab samples suggest the iron formation-basalt contact could be mineralized over 700 m strike length.

7.3.20 Roberts South

A trenching program in 2017 targeted a roadside exposure of iron formation roughly 1.1 km south of the Roberts trenches. Anomalous gold results from 2016 channel sampling on exposed iron formation beds prompted mechanical stripping in 2017. Geologically the area is underlain by rocks similar to the Roberts target, comprising east dipping, north-south trending folded sulphide-facies iron formation and narrow silicate iron formations / metasedimentary rocks intercalated with basalt. Deformed quartz veins, sulphidized wall rock around veins and silicified sulphide-facies iron formation host gold (Leidl 2017). Channel samples yielded up to 5.9 g/t Au over 1.0 m. The Roberts South target has not been drill-tested.

7.3.21 Russet North

The Russet North target has a similar geological setting to Russet South. This target is 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.

7.3.22 Russet Far North

This target occurs north of, and along strike of Russet South and Russet North in the hanging wall to an ultramafic sill. The target extent is localized to a gold in soil anomaly spanning 200 m strike length. In 2017, Pure Gold drilling intercepted a best result of 2.7 g/t Au over 1.0 m. Gold is hosted by quartz veins within basalt subjacent to ultramafic rocks and altered basalt.

8 Deposit Types

The Madsen Gold Project is focused on identifying and delineating Archean orogenic gold deposits (Groves et al., 1998).

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 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°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 Madsen Gold Project Mineralization Model

All significant gold mineralization at Madsen is demonstrably early relative to the most significant, penetrative deformation event (D2). Quartz veins at 8 Zone, Wedge and Russet South are boudinaged, recrystallized and folded and are cut by the penetrative S2 foliation. Mineralized bodies of the Austin, South Austin, and McVeigh Zones are locally folded and transposed into S2 but this event is largely a flattening event. This geometry is defined by sampling data on numerous historical level plan maps (Figure 7-9). Thus, a major component of the deposit model is the expectation that mineralized bodies are deformed and may be dismembered and/or folded at a small scale. At the property-scale, however, these structures are conspicuous based on patterns of gold mineralization and alteration and are continuous on a kilometer-scale (e.g. Austin/ South Austin Zones are planar and continuous over an area of at least 600 by 2,000 m). Another important characteristic is that these planar structures are oblique at low-angles to host Balmer stratigraphy, the Balmer/ Confederation Assemblage unconformity and to the penetrative S2 foliation. To summarize, the Madsen gold-bearing structures are early and strongly overprinted by S2, but only minor displacement across S2 exists. Rather, the early structures are continuous at the property-scale despite the strong deformation of the gold-bearing zones.

At drill core scale, much variation in mineralogy and structural features of gold-bearing zones exists. Typically, these features are linked to the chemical and rheological features of the host rock. At least minor gold mineralization is hosted in most rock types on the property, but some important patterns form the basis of the exploration model. Principal among these patterns is the occurrence of the most significant gold mineralization within Balmer basalt adjacent to ultramafic sills. This pattern is conspicuous in level plan and cross section throughout the Madsen and Starratt Mines. Most gold precipitated within several 10s of metres of the intersection of mineralized structures and an ultramafic/ mafic contact. Likely, the relatively iron-rich basalt encouraged destabilization of gold-bearing bisulfide complexes which are generally considered the most likely gold transport mechanism (Mikucki, 1998) where the fluid interacted with the ultramafic / mafic interface. Additionally, mafic rock likely acted more competently promoting fluid pathways and permeability (e.g. Cox et al., 1986). A similar mechanism of chemical and rheological contrast was likely at work where mineralized structures intersected iron formation in close proximity to ultramafic contacts; significant gold mineralization is present in these settings at the Fork and Russet South deposits.

Within the Madsen deposit, a secondary control was apparently provided by early, metre-wide-scale, quartz porphyry intrusive bodies (lithological code QZPY). In detail, the quartz porphyry bodies are preferentially un-mineralized (or very weakly mineralized) but the basalt surrounding these bodies is locally exceptionally well-mineralized. The most extreme example of this relationship is within the Austin zone at deeper levels within the Madsen Mine. In this area the quartz porphyry body forms a largely barren core within the Austin historical ore body, the orientation of which is an effective vector to the South Austin zone on the footwall side of an ultramafic sill. The Madsen geological model interprets that these felsic intrusions emplaced along the same early structures that controlled gold mineralization. These intrusions are sodium-depleted (Mackie, 2016), weakly mineralized and highly altered suggesting they emplaced prior to gold deposition.

Thus, the main components of the Madsen Gold Project mineralization model include:

- Significant gold deposition occurred prior to the main, belt-scale deformation event (D2) within largely planar structures that have been nearly completely recrystallized by overprinting deformation and metamorphism; and
- Geometrically, gold deposits were folded by small-scale, localized folds, structurally dismembered by transposition and gold was remobilized into secondary (metamorphic) mineral phases. Effective exploration drill targeting requires anticipation of these shapes and expectation of a heterogeneous gold system.

At a small-scale (10s of metres), characteristics of mineralized rock were heavily influenced by host rock. These controls include both rheological (quartz porphyry dykes) and chemical controls (Balmer mafic versus ultramafic and Balmer mafic versus felsic Confederation rock).

8.3 Concepts Underpinning Exploration at Madsen

Exploration for gold at Madsen 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 is a highly flawed approach. 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 criterion. Structural complexities such as jogs or interaction with lithological contacts are also targeted.

9 Exploration

A summary of the historical exploration work completed between 1928 and 2013 is discussed in Section 6.0.

Since acquiring the Madsen Gold Project in 2014, in support of extensive diamond drill testing Pure Gold has completed several focused surface exploration campaigns (Table 9-1) comprised of geological mapping and rock and soil sampling. 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 and Dev Northwest targets in 2016 and 2017 and the Wedge deposit in 2018. 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. These surface programs have been 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. The sampling programs have 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 have been developed and significant high-grade gold-bearing drill intercepts have resulted at Starratt, Fork, and Russet South.

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.

Table 9-1: Madsen Gold Project Non-drilling Exploration 2014 – 2018

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 & Russet South 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 South 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 South, Dev, Russet North	202 rock, 72 channel, 3,234 MMI soil	Baker and Swanton (2015)
Petrography	2015, 2016	Russet South, Madsen	67 thin polished sections	Ross (2015), Leitch (2016)
Mechanical stripping, rock sampling	2015	Russet South	78 rock	Pure Gold database
Mechanical stripping, geological mapping, rock sampling	2016, 2017	Dev, Dev Northwest, Roberts, Roberts South	296 Rocks	Jones (2016), Pure Gold database
Soil sampling	2016	Property-wide	2481 Soils	Pure Gold database
Geological mapping, rock sampling	2017	Property-wide	143 Rocks	Pure Gold database
Soil sampling	2017	Derlak	686 Soils	Pure Gold database
Geological mapping, rock sampling, mechanical stripping	2018	Wedge	125 Rocks	Pure Gold database
Historical core re-logging	2017, 2018	Property-wide	595 Holes 271,429 m	Nuttall (2017), Bultitude (2018)

Source: D. Baker (2018)

9.1 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 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.

9.2 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 Madsen mine grid surface survey controls and created a new transformation conversion to and from the latest federal datum NAD83(CSRS, 2010) UTM 15 CGVD 2013 (Dorland, 2017). Following this work, Dorland conducted an underground control survey from the established surface controls down the Madsen ramp to 2 level, adjusted the surface and underground control surveys and placed new wall control points.

9.3 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.

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 outcrops across the property.

The resultant property-wide geological map with lithological, alteration and structural interpretation is summarized in Figure 7-11. Mapping has defined structural features (foliations, folds) relating to different deformation events and constrained the timing of gold mineralization relative to these events.

9.4 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 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 South 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 numerous samples with >1 g/t Au.

9.5 Rock Geochemistry

Pure Gold has collected and analyzed approximately 1,874 surface rock samples using whole rock lithogeochemical, fire assay gold plus multi-element or portable XRF analysis. The lithogeochemical samples were collected to determine the composition of the main lithologic units during mapping and to determine gold content and alteration. 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.

9.6 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.

In all, Pure Gold has 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 7.4). Given the highly sensitive nature of the MMI technique, some areas of anomalous gold near historical mine sites are likely due to wind or water transported tailings contamination.

9.7 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 current 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.

9.8 Petrography

Pure Gold has undertaken 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 geologic mapping scheme.

9.9 Underground Bulk Sample

Pure Gold collected an underground bulk sample of the McVeigh Zone in 2018 to gauge lateral and vertical continuity of gold mineralization, validate the resource model, provide additional geotechnical information, and 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. Additionally, 13 raise faces were also blasted from two separate raises. Each face was geologically mapped, photographed and sampled. Typically, two parallel sampling lines were 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 exceeds 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 holes (excluding QAQC samples, which were systematically inserted into each sample batch) were collected.

The test mining was advanced using a single-boom hydraulic jumbo drill, typically with one round blasted per day. Blasted rock was mucked using scooptrams 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 ~10–20 m horizontally with a Bazooka air drill which returns EW size (25.4 mm or 1 inch) core samples which 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.

9.10 Exploration Targets

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

9.11 Sampling Methods and Quality

The rock and soil sampling by Pure Gold has been systematic and completed according to a clear set of 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.

9.12 Interpretation

The Madsen Gold Project surface (non-drilling) exploration dataset comprises systematic, property-wide, multifaceted information carefully collected by modern techniques. Combining surface geophysical (magnetic), geochemical and geological information with historical data and drilling data has allowed for a property-wide geologic map which 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.

10 Drilling

10.1 Historical Drilling

Information about historical drilling on the Madsen Property is described in Sections 6.0 and 14.0.

10.2 Pure Gold Drilling (2014–2018)

From project acquisition through 2018, Pure Gold has drilled 751 NQ-sized core holes for 208,669 m. 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 the resource domains with a focus on resource growth. Additionally, several target areas across the property were tested including Starratt, Fork, Russet South, and Wedge and initial drill holes tested several regional targets.

During the 2018 underground bulk sample program, drift walls were tested with a Bazooka air drill which returns EW size (25.4 mm or 1 inch) core samples. A total of 1,976 m of Bazooka core drilling was completed in 153 horizontal holes.

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 is consistent with interpretations that the gold mineralization at Madsen developed early in the tectonic history of the belt and has been deformed and folded.

Drilling was completed by Major Drilling in 2014 and early 2015 and Hy-Tech drilling from early 2015 to present. Most holes (except the underground Bazooka holes) were drilled with NQ-size 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 using a Trimble ProXRT differential GPS receiver with Omnistar real-time correction, which achieves sub-metre precision. 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 for drill alignments and initial gyro orientations. All drilling sites were cleared of any cut timber and debris, re-contoured and re-seeded with a native seed mix post-drilling. All drill holes were logged, photographed, and sampled at the Madsen Mine following the procedures described in Section 10.4. All data collected during core processing was stored in the Reflex Hub (formerly ioHub) cloud database until early 2018 when the database was moved to the Datashed™ system.

10.2.1 Core Processing

Upon initiation of drilling on a hole, the project geologist assigns a logging geologist and 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 project geologist to validate all drill hole data and ensure transfer to the cloud database on completion of the logging and sampling.

10.2.2 Geological Quick Logging

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.

10.2.3 Geotechnical Procedures

All drill core is prepared by a technician prior to geological logging. 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, measuring offset angles of bottom of hole marks, recording loss of orientation lines, and placing down-hole metre marks as well as measuring recovery, rock quality designation and magnetic susceptibility. All downhole measurements are collected to the nearest centimetre.

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.

10.2.4 Geological Logging

All drill core at Madsen was geologically logged directly into a laptop or tablet computer using Reflex Logger software until late 2017 and LogChief™ in 2018. 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. Mineralized veins 20 cm wide or greater, are logged separately. The focus of the geologic logging is to highlight the mineralized zones and to also capture lithology, alteration, vein and structural data.

10.2.5 Structural Data

All drilling by Pure Gold uses Reflex ACTIII core orientation tools. All core intervals in the Balmer Assemblage target units are oriented with a bottom of core mark and additional intervals of interest such as quartz veins captured in other less prospective rock units. 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 are 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.

10.2.6 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.

10.2.7 Core Storage

After logging, photographing and sampling, the core is 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.

10.3 Summary

Diamond (core) drilling is the most appropriate method for the Madsen Gold Project and this technique has been applied by all operators since early exploration and mining. Historical drilling is tightly-spaced (nominally drilled at 25-foot centres) within mined-out areas but other areas show evidence of alteration and elevated gold but have been drilled at much broader spacing. Based on core recovery measurement of >40,000 individual drill runs completed by Pure Gold, recovery averages 99.6% which is very good and indicative that core recovery is not a factor in the accuracy or reliability of results. Pure Gold’s methodology and procedures meet or exceed typical industry standards and historical operators generally operated at standards in line with the times. As such, the drilling conducted at the Madsen Gold Project has produced a reliable geological and geochemical database.

11 Sample Preparation, Analyses and Security

Sampling procedures and methods have evolved significantly over the long history of exploration and mining at the Madsen Gold Project and specific procedures also varied among operators. As such, descriptions of sample preparation, analyses and security are described separately below according to time period and/or operator.

The author 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 by Pure Gold meet or exceed modern best practices. The historical procedures and those undertaken by Pure Gold are adequate for modern targeting, modelling and resource estimation.

11.1 Sampling

11.1.1 Historical Sampling (1936–1982)

Drill core, chip and muck sample preparation, analysis and security procedures for historical samples taken during the operation of the Madsen Mine are not documented and therefore difficult to review. Samples were assayed for gold at the mine laboratory but no information exists regarding lab 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.

Sample preparation, analysis and security procedures for 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 lab certifications but as indicated in the preceding paragraph, such early work predates applicable ISO standards. The preparation and assaying techniques are not documented.

11.1.2 Placer Dome (2001–2006)

Placer Dome used two primary laboratories for assaying drill core samples collected from the Madsen Gold Project. All samples from 2001 to 2006 were assayed by XRAL Laboratory in Toronto, Ontario or ALS Chemex Laboratory in Vancouver, British Columbia.

11.1.3 Wolfden and Sabina (2003–2012)

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 using inductively coupled mass spectroscopy (ICP-MS) 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 (2006–2013)

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 (2014–2018)

During 2014, 2015, 2016 and 2018 Pure Gold submitted all drill core and surface rock samples to ALS Minerals (ALS) Laboratory in Thunder Bay and Vancouver for sample preparation and analysis, respectively. Pure Gold submitted pulp duplicate samples to SGS Laboratory in Burnaby, British Columbia for check assay testing. 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 Madsen samples (accredited lab 579). The SGS laboratory is CAN-P-159, CAN-P-1578, and CAN-P-4E (ISO/IEC 17025:2005) certified by the SCC for the analytical methods used on the Madsen 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).

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 was re-assayed by metallic screen fire assay (method Au-SCR24). 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.

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.

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 analyses after being delivered to the Red Lake laboratory.

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 Madsen 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 Madsen 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 Madsen samples (accredited lab 184).

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, by a 30 g fire assay with an atomic absorption spectroscopy finish (method GE_FAA313). In cases where the assay value returned >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 <1 kg and >1 kg respectively). 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).

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. 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 PUL45).

Analysis for gold was conducted at the SGS laboratory in Red Lake, by a 30 g fire assay with an atomic absorption spectroscopy finish (method GO_FAA303). In cases where the assay value returned >100 ppm Au, a follow up gravimetric analysis was conducted (30 g fire assay with a gravimetric finish, method GO_FAG303).

11.2 Sample Security

11.2.1 Historical Sampling (1936–1982)

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

11.2.2 Claude (2006–2013)

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 (2014–2018)

Currently (and during the 2014–2018 drilling programs), Madsen Gold Project personnel employ 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 locked and fenced logging facility by drill crew members at twice daily shift changes via truck or snowmobile;
iii. Core shack personnel open and sort core boxes for logging;
iv. Core awaiting sawing (sampling) is stored in wooden racks in the core shack;
v. Core is sawn and bagged into pre-labelled sample bags by samplers under the supervision of core logging geologists and the project geologist;
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) 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, pallets are collected from the Madsen Mine site directly by SGS personnel and driven to their Red Lake facility;

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

x. Hardcopy chain of custody forms and sample analytical instructions are included with each shipment with copies sent by email, ALS and SGS reported all shipments were received intact.

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.

11.3.1 Historical Period (1936–2000)

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 (2001–2006)

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 (2003–2012)

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

11.3.4 Claude (2006–2013)

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, 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 (2014 – 2018)

For all Pure Gold drilling programs, Madsen Gold Project personnel implement a Quality Assurance and Quality Control (QAQC) program comprising of 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 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 lab. 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 20th sample, with two additional blanks inserted following samples containing visible gold.

Standards used by Pure Gold between 2014 and 2018 range 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 11-1. 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 11-1: CRMs used by Pure Gold (2014–2017)

Supplier	Standard ID	Year(s) in use	Use Case	Gold Assays (ppm Au)	
				Certified Value	SD
Ore Research	OREAS6pc	2015	Low Grade	1.52	0.07
CDN Labs	CDN-GS-1M	2016–2017	Low Grade	1.07	0.05
Rocklabs	SG56	2014–2016	Low Grade	1.027	0.01
Rocklabs	SH55	2016	Low Grade	1.375	0.05
CDN Labs	CDN-GS-1T	2017-2018	Low Grade	1.08	0.05
CDN Labs	CDN-GS-1V	2018	Low Grade	1.02	0.10
Ore Research	OREAS 214	2016–2018	Medium Grade	3.03	0.08
Ore Research	OREAS 17c	2014–2015	Medium Grade	3.04	0.08
CDN Labs	CDN-GS-5F	2014–2015	High Grade	5.27	0.17
Rocklabs	SL61	2015–2016	High Grade	5.931	0.06
CDN Labs	CDN-GS-6E	2016–2018	High Grade	6.06	0.15
Rocklabs	SQ 36	2014–2016	High Grade following VG	30.04	0.02
CDN Labs	CDN-GS-22	2016–2018	High Grade following VG	22.94	0.56
Rocklabs	SQ87	2016	High Grade following VG	30.87	0.21

Source: Pure Gold (2018)

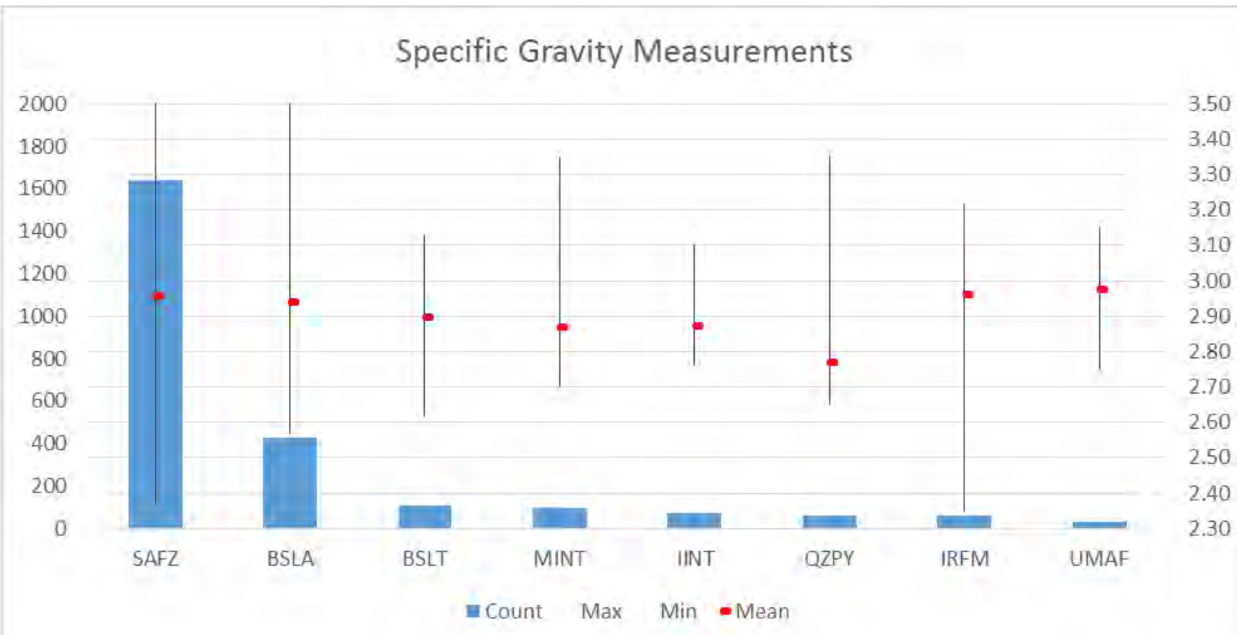
11.4 Specific Gravity Data

Historically, the Madsen Mine used a tonnage factor of 11.25 cubic feet per ton to convert volumes into tonnages. This factor was determined from a bulk sample of Austin ore in 1938 and applied throughout the historical mine life. This tonnage factor is equivalent to a specific gravity of 2.84.

Claude collected specific gravity data from core using the water displacement method until approximately mid-2012 (Cole et al., 2010). A total of 3,010 specific gravity determinations were completed on core from across the Madsen deposit as well as from the Fork and Russet South deposits. The average specific gravity determined from this study was 2.91.

Pure Gold collected specific gravity determinations on selected intervals of core using the ALS water displacement method (OA-GRA08) at the analytical preparation stage in 2016 from the Madsen, Fork and Russet South deposits. A total of 3,013 samples were selected from a full range of lithologies and measured at the ALS prep lab. Results for the most common rocks types within the vicinity of mineralization are summarized in Figure 11-1. Significantly, these values do not vary much between rock types or between auriferous and barren rock. The overall average specific gravity for these samples is 2.94.

Figure 11-1: Summary of Specific Gravity Data by Rock Type from Pure Gold Study



Source: Pure Gold database (2017)

As part of the FS testwork completed on the 12 composites created from the 2000 kg of metallurgical samples, specific gravity determinations w using the water displacement method returned an average value of 2.96. Tailings studies completed for the FS testwork by Knight-Piésold returned slightly higher average values of 3.00, as would be expected from ground tailings (75 micron). Based on the consistency and volume of samples with modern testwork an SG of 2.94 was selected for the Mineral Resource Estimates for all zones.

The selected value compares favorably to other deposits in the district that share the same lithologies and alteration with the Red Lake Mine using an average of 2.91, Cochenour Mine an average of 2.91, and 2.98 for the High Grade Zone at Red Lake Mine (Goldcorp, 2019).

12 Data Verification

Owing to the long history of exploration and production at Madsen, 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 2002–2006 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 has been the foundation for drill-targeting and geological interpretation by Pure Gold and has been substantially added to and verified since 2014.

The qualified person has consulted to the Madsen Gold Project since 2014 and has completed data verification throughout this work including:

- Confirmation of hundreds of historical assay, lithological and collar location database records with hard copy drill logs stored in the Madsen site archive;
- In-field location confirmation of numerous drill hole casings during geological mapping across the Madsen Property;
- Independent drill hole database validation using Micromine™ software drill database validation tools;
- Confirmation of consistent use of historical lithological logging codes by comparison with modern drilling data;
- Validation of geological sectional interpretations with level plan geological interpretations including reference with original hard copy mine level plans; and
- Review of consultant QAQC reports and verification of their conclusions on CRM and blank sample analyses.

12.1 Database Validation based on Logged Lithological Intervals

Using the historical drill hole database, data from modern (Pure Gold) drilling and with reference to the original mine level plans and cross sections, author Baker conducted systematic geological interpretation of the Madsen Mine along 100 foot-spaced section lines corresponding to the original imperial mine grid. In total, 140 sectional and 32 level plan interpretations have been completed extending from the southwestern end of Starratt, across Russet South and northeast to Coin Lake. Initially, only sectional interpretations for the top 16 levels of the Madsen Mine were created but later this interpretation was extended beyond the lowermost depths of the historical workings. Sectional interpretations were reconciled using level plans constructed at the elevation of the historical workings and ultimately the level plans were used to build a 3D wireframe geological model in Leapfrog Geo™ which is regularly updated with new drill data.

The drill hole database includes nearly 237,000 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 nearly 70 years of exploration, the coding schemes varied with time, so some interpretation of these codes was required prior to geological interpretation. Initially, upon acquisition of the project, Pure Gold built a code equivalence table and recoded the database (preserving the original code) to the modern coding scheme. Baker used this scheme and verified many intervals by referring to recent adjacent Pure Gold drill holes. The initial recoded scheme was deemed useful and consistent enough 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 original lithological codes have been made to refine the interpreted codes.

12.2 Drill Hole Location and Survey Data

As qualified person, author Baker has personally collected GPS location data confirming numerous surface drill holes from various eras including Placer Dome, Claude and Pure Gold collar locations. Invariably they match the drill hole database which comprises converted historical location data and modern survey location data collected during a focused survey program in 2014 (described in Section 9.2). Earlier surface and underground drill hole locations cannot 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 original coordinate system (Imperial Mine Grid).

Conversely at the Russet South 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 historical intercepts for Russet South have not been used in the calculation of the resource, which has relied entirely on holes drilled by Pure Gold.

Downhole survey methods for the earliest (mostly underground) drilling were rudimentary compared with today's gyroscopic survey methods. Original drill logs indicate that a variety of methods were employed or in some cases, no downhole surveying was completed. Magnetic survey methods are problematic at Madsen 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 (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 which are well-surveyed by modern gyroscopic tools. Pure Gold has also re-surveyed (downhole) many important holes drilled by previous operators which only had magnetic survey data. The underground test mining in 2018 exposed many exploration and resource definition drill holes and their known locations were generally in agreement with locations calculated from downhole surveys.

In conclusion, the collar locations for the historical drill hole database are well-compiled, have been translated accurately from mine grid coordinates and are adequate for the purpose of this report. 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. All Pure Gold completed drill holes (except underground Bazooka holes) have been located with sub-meter differential GPS and surveyed downhole with modern gyroscopic survey tools.

12.3 Analytical Data Verification

Analytical quality control measures were generally not used prior to 2009 as detailed in Section 11.3. Several operators make mention of the existence of quality control programs but provide few details. Implementation of rigorous analytical quality control measures for Madsen geochemical samples began in 2009 by Claude. Original records of these programs are sparse but SRK reports on the procedures and results of Claude's program and their authors had direct access to the work program (Cole et al., 2010).

Upon acquisition of the project, Pure Gold implemented a robust quality control program as detailed in Section 11.3.5. Data generated to the end of 2016 drilling campaigns from analyses of certified standard reference material, blank, field, coarse reject (preparation), pulp and umpire duplicate samples were plotted and interpreted independently by Gary Lustig and Dennis Arne (Arne, 2014; Lustig, 2015; Arne, 2016). Quality control samples for the 2017 drilling campaign have been reviewed by Chris Lee (Jutras et al., 2017) and Dennis Arne recently reviewed all 2017 and 2018 quality control data (Arne, 2018). Overall compliance rates for these samples are acceptable but given the large number of quality control samples submitted, numerous issues and areas for improvement have been high-lighted. These have been 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 has been largely mitigated.

Coarse blank samples exhibit acceptable rates of compliance in 2017 (99%) and in 2018 (99.3%) in line with expected commercial laboratory performance (Arne, 2018).

Analytical data from certified reference material samples systematically inserted into core sample streams show low average biases indicating these data to be accurate (Arne, 2018). Certified reference material sample failure rates are at the upper end of acceptability for 2017 data, but failures in 2018 were significantly lower, which is likely attributed to switching to a different analytical laboratory.

Duplicate sample analytical data was reviewed in detail to assess the uncertainties associated with the various stages of sampling, crushing and pulverization and to advise on sampling and analytical protocols (Arne, 2016; Arne, 2018). Analytical precision implied by these duplicate pairs is generally within the acceptable range for gold deposits characterized by coarse gold (Abzalov, 2008; Arne, 2016; Arne, 2018).

During the 2018 test mining program, blank, standard reference material and preparation duplicate samples were analyzed within the sample streams of chip and muck samples. Coarse blank samples, however, showed a 94–95% rate of compliance which is lower than core samples. Standard reference material samples performed well save for seven samples (of 233) that fell outside the 3SD control limit. . Preparation duplicate data shows good reproducibility above 0.05 g/t Au. Given the production pace of sample collection, submittal and analysis (the local lab returned gold values within 18 hours of receiving samples) this performance is considered acceptable and fit for purpose.

All Pure Gold drilling data has 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 Pure Gold upon receipt of analytical certificates.

12.4 Other Data Verification

Pure Gold routinely commissions 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. Recently, Pure Gold commissioned a third independent database review on the database supporting this report (Murphy, 2019). Murphy identified some discrepancies between the current 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 concludes that the database and structure are acceptable and a reasonable rendition of the historical and modern data.

The current author shares the opinions of Mackie and Murphy that the Madsen Gold Project database is of high quality, reliable and considers it appropriate for geological targeting and resource estimation.

12.5 Summary

The large Madsen Gold Project drilling database is built from historical and modern work that spans nearly 70 years. Records of quality control or data handling procedures are largely non-existent prior to 2009. The early work was, of course, in hard-copy only and it took nearly 11 years of sporadic effort to translate this into a 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 has been conducted with clear data handling protocols and an industry-standard quality control program making verification of these data simpler. The database has been independently validated (Mackie, 2015; Mackie, 2017; Murphy, 2018).

Given the long history at Madsen, the geographic, geological and analytical data housed in the Madsen Project database is highly robust, well-organized, easy to use and effective for interpretation work and decision-making and is appropriate for the purposes used in this technical report.

12.6 Metallurgy Data Summary

Pure Gold provided location and details of drill holes and the locations of the holes were plotted against the planned areas to be mined. The drill holes were found to be spatially representative and provided sufficient variability in gold head grade. Intervals were chosen from the core and sent to BaseMet Laboratories in Kamloops, BC. Drill core was used to create Variability composites from each of the five zones in the deposit well as composites representing the Years -1, 1, Years 2, 3 and Years 4 to 7, based on areas to be mined from the PEA mine schedule. It is the QP's opinion that there is sufficient data and test work to estimate metallurgical recoveries and define the flowsheet at a Feasibility Study Technical Report as required by NI 43-101 guidelines.

12.7 Mining Data Summary

Mining data was verified during the site visits, through review of previous studies and historical data. Any studies referred to were thoroughly reviewed, updated and revised as required to align with the FS mine design and mine plan. All mining data was verified and is adequate for the FS Technical Report as required by NI 43-101 guidelines.

12.8 Waste and Water Management Data

Design parameters for the tailings material were verified by completing an extensive tailings laboratory testwork program. Foundation conditions at the existing TMF were investigated by performing a drilling and test pitting program at critical locations within the TMF. Material characterization was carried out on samples collected during this investigation to determine design parameters. Existing studies and data collected during historic site investigations was reviewed and revised as required. Design parameters used in the design are verified and are adequate for the FS Technical Report as required by NI 43-101 guidelines.

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 90's 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 90's continued to be achieved. The present mill was purchased and relocated in the 1990's 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 this Feasibility Study, 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 following report:

- Base Metallurgical Laboratories (2018). "Metallurgical Testing for the Madsen Gold Project" BL0288" (Issued: 21 September 2018).

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

Based on the results from Base Met (2018), gold doré with no significant levels of deleterious elements can be produced with a primary grind size of 80% passing (P_{80}) 75 µm followed by gravity concentration, 2 hour pre-oxidation, 24 hour cyanide leach, 6 hour carbon-in-pulp (CIP) adsorption, desorption and refining process. The blended average of the samples tested based on the mine plan using this method is estimated to achieve a recovery of 96% Au.

13.2 Base Met (2018) Test Program Results

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 this Feasibility Study.

The test program was done in four phases: Variability Scoping Composites, Year Composites, Tailings Generation 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. Phase 2 included test work on three year composites that represented the years -1 to 1, 2 to 3 and 4 to 7 of the PEA 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

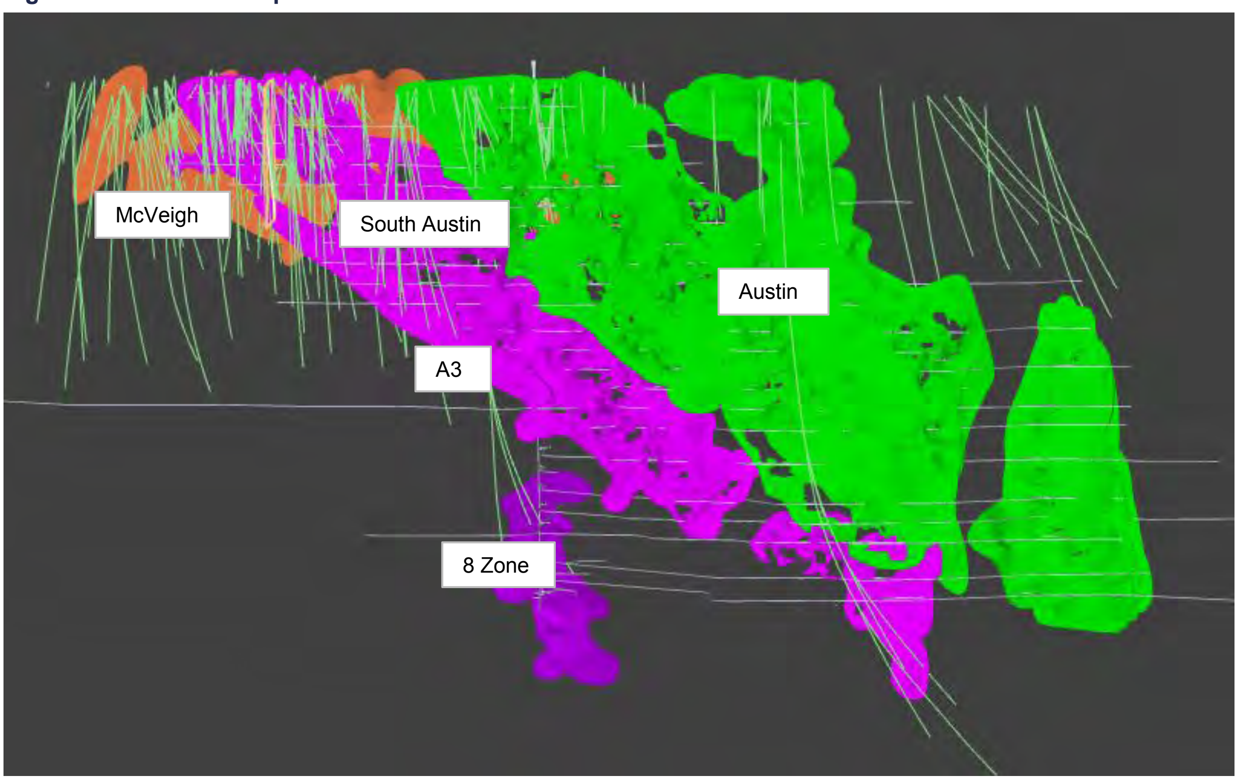
Samples were received in February 2018 by BaseMet in two forms. Approximately 870 kg arrived as cut core (1/4 and 1/2 core) and about 14.9 kg arrived as coarse rock samples. In total, 286 individual core interval samples and 20 coarse rock samples were received. The drill core was collected from McVeigh,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

Twelve scoping variability samples were created using the individual drill core from the five zones. Three composites were constructed to represent years -1 to 1, 2 to 3 and 4 to 7 based on the PEA mine schedule. Forty-eight variability composites were created from the five zones but there was only significant mass for 30 composites to be tested. The coarse rock samples were used for the Crusher Work Index (CWi) determinations.

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 riffled 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: Madsen Deposit



Source: Pure Gold / JDS (2018)

13.2.2 Scoping Variability Composites

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 (P_{80}) 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.

13.2.2.1 Head Assays

Twelve global 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 samples and head assays are shown below in Table 13-1.

Table 13-1: Head Assays of Composite Samples

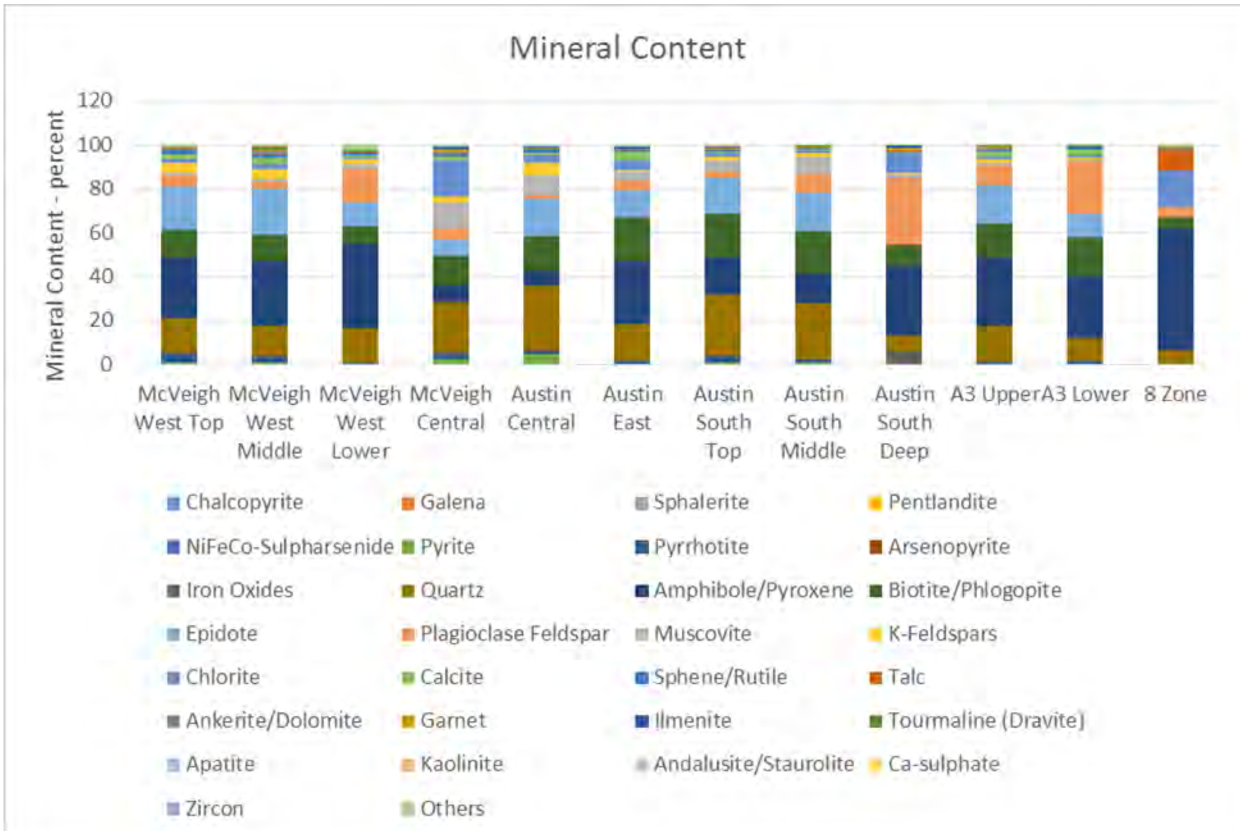
Composite ID	Assay – percent or grams/tonne					
	Au (g/t)	Ag (g/t)	Fe (%)	S (%)	C (%)	As (g/t)
McVeigh West Top	5.42	2	5.4	2.11	0.72	128
McVeigh West Middle	3.75	1	4.7	1.24	0.94	65
McVeigh West Lower	9.42	1	1.6	0.35	0.28	41
McVeigh Central	4.62	2	7.5	2.02	0.47	334
Austin Central	5.72	1	6.0	2.78	0.44	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)

13.2.2.2 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: Base Met (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 observed.

13.2.2.3 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).indicating that the material is moderately hard. The results ranged between 9.7 kWh/t to 17.1 kWh/t and averaged 14.4 kWh/t.

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 13-2.

Table 13-2: Comminution Results – Scoping Variability Composites

Sample ID	Size Fraction Tested (mm)	DWi	DWi	Mia	Mih	Mic	A	b	sg	ta	SMC	F_{80} µm	P_{80} µm	Gpr	BWi
		kWh/m³	%	kWh/t	kWh/t	kWh/t									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: Base Met (2018)

13.2.2.4 Flowsheet Development

The baseline leach parameters included a primary P_{80} 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 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 87% to 99% and an average cyanide consumption of 0.7 kg/t and lime 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.

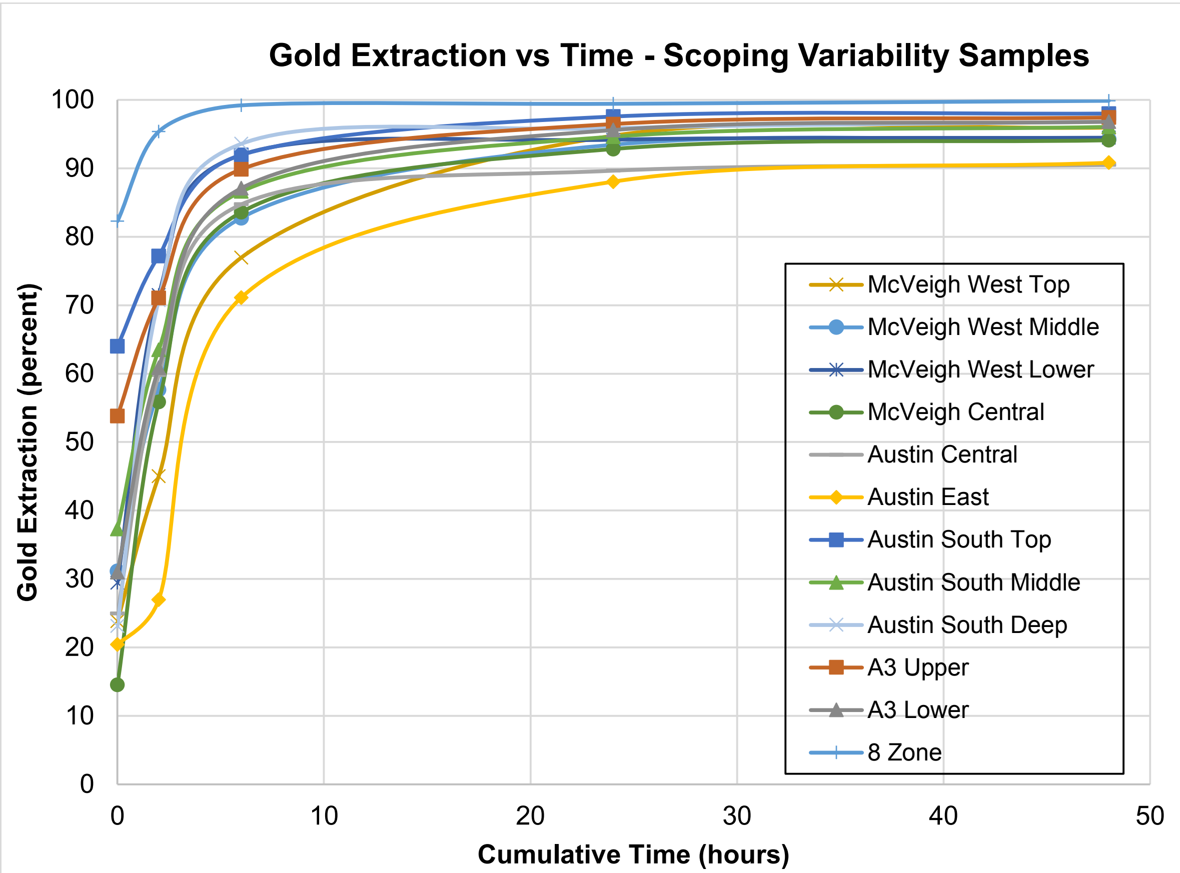
The results are summarized in Table 13-3 and gold extraction versus time using the baseline conditions for the 12 composites are illustrated in Figure 13-3.

Table 13-3: Bottle Roll Leach Results for Base Met (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: Base Met (2018)

Figure 13-3: Effect of Grind Size on Gold Extraction



Source: Base Met (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 Year -1 to 1, Year 2 to 3 and Year 4 to 7. The PEA mine schedule was used as a guide for target grade and ratio of material from each of the 5 zones to be mined. Test parameters from Phase 1 were used as the baseline conditions 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 Composite 1 and 2 due to the limited sample size of Year Composite 3.

13.2.3.1 Head Assays

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

Table 13-4: Head Assays for Base Met (2018)

Composite ID	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: Base Met (2018)

13.2.3.2 Mineralogy

The three year composites were sent out for Un-sized Bulk Mineral Analysis (BMA) using QEMSCAN. Table 13-5 is a summary of the mineral content for each of the composites.

Table 13-5: Mineral Content – Year Composites

Minerals	Year 1 to 1	Year 2 to 3	Year 4 to 7
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

Note: 1) Iron Oxides include Magnetite, Hematite and Geothite.
2) Others includes trace amount of Andalusite, Zircon and unresolved mineral species.
Source: Base Met (2018)

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

13.2.3.3 Comminution Results

BWi tests were conducted on all three year composites at a closing screen sizing of 106 microns (µm) which 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 Comp 1, 2 and 3, respectively.

13.2.3.4 Gravity Recoverable Gold Test Work (GRG)

The three year comps were 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 P_{80} 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 including two concentrators to handle the total feed to the mill in the existing grinding circuit followed by intensive leaching of the concentrate is recommended. The anticipated recovery from the two units ranges between 62.3% and 79.2% of the total gold in the mill feed.

13.2.3.5 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.

13.2.3.5.1 Primary Grind

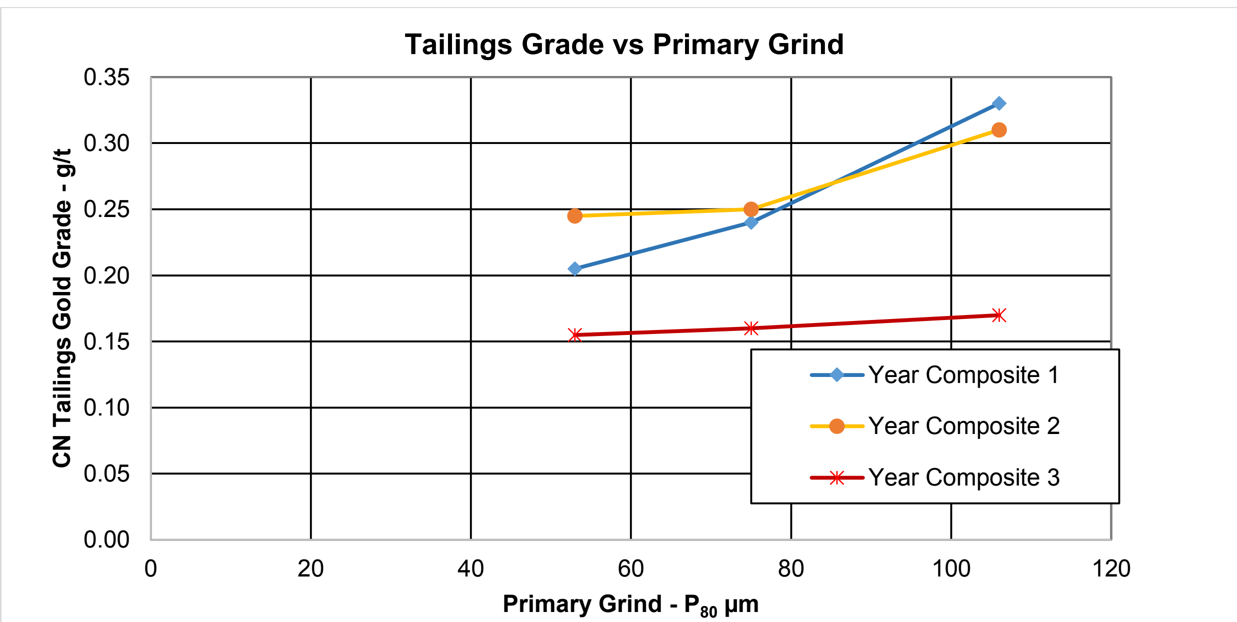
At a grind size above 75 µm an increase in the tailings grade was observed. There was no correlation observed between grind size and gold extraction although tailings grade appeared to increase for Year Comps 1 and 2 at grind sizes 80 µm and above. The results are summarized in Table 13-6 and illustrated in Figure 13-4 below. Based on these results a primary grind size of 75 µm was used in subsequent tests.

Table 13-6: 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				Cu	Fe	As
Year Composite 1	21	106	0.39	0.13	95.3	0.33	6.9	16.9	8.9	1.0
	22	75	0.46	0.13	96.8	0.24	7.4	20.0	13.7	0.9
	23	53	0.56	0.11	96.1	0.21	5.3	34.0	15.5	1.6
Year Composite 2	24	106	0.48	0.18	98.3	0.31	18.6	15.6	17.9	5.6
	25	75	0.55	0.18	96.5	0.25	7.1	16.2	19.0	5.2
	26	53	0.67	0.22	98.7	0.25	18.4	26.8	21.9	5.7
Year Composite 3	27	106	0.54	0.27	98.2	0.17	9.3	11.6	12.0	9.0
	28	75	0.56	0.25	98.5	0.16	10.4	10.6	13.4	10.0
	29	53	0.64	0.27	98.2	0.16	8.6	12.2	17.0	9.4

Source: Base Met (2018)

Figure 13-4: Effect of Grind Size on Gold Extraction



Source: Base Met (2018)

13.2.3.5.2 Optimization Test Work

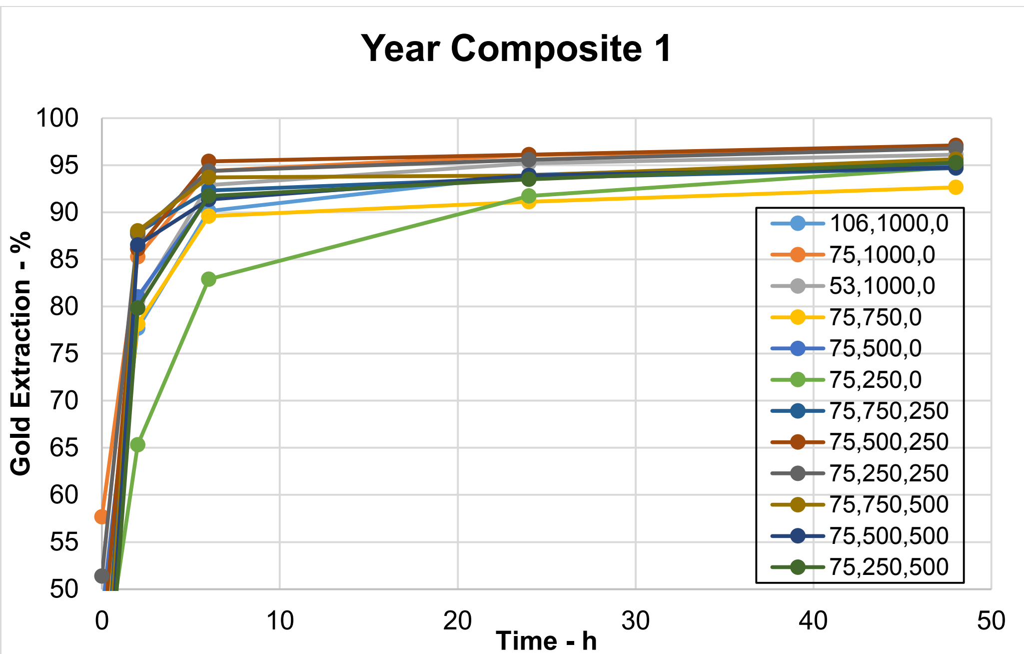
Tests were conducted at varying cyanide concentrations ranging from 250 ppm to 1000 ppm and lead nitrate addition between 0 ppm to 500 ppm. The higher cyanide and lead nitrate additions do not significantly improve gold extraction. The test conducted without lead nitrate did not perform as well. The 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 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.

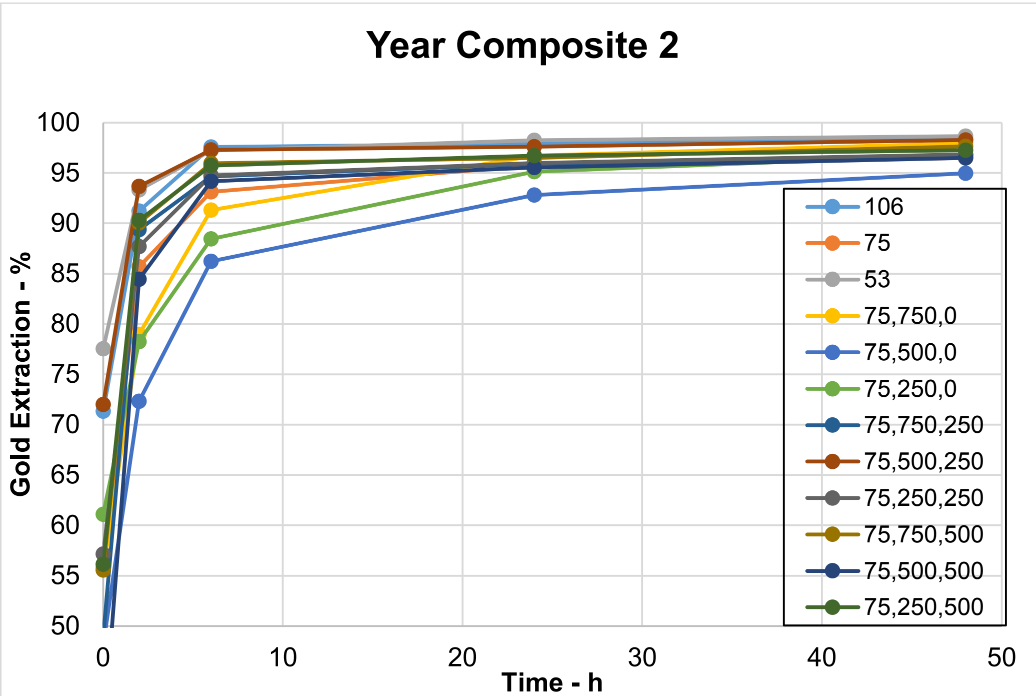
The results are illustrated in Figure 13-5, 13-6 and 13-7.

Figure 13-5: Year Comp 1 Gold Extraction vs Time



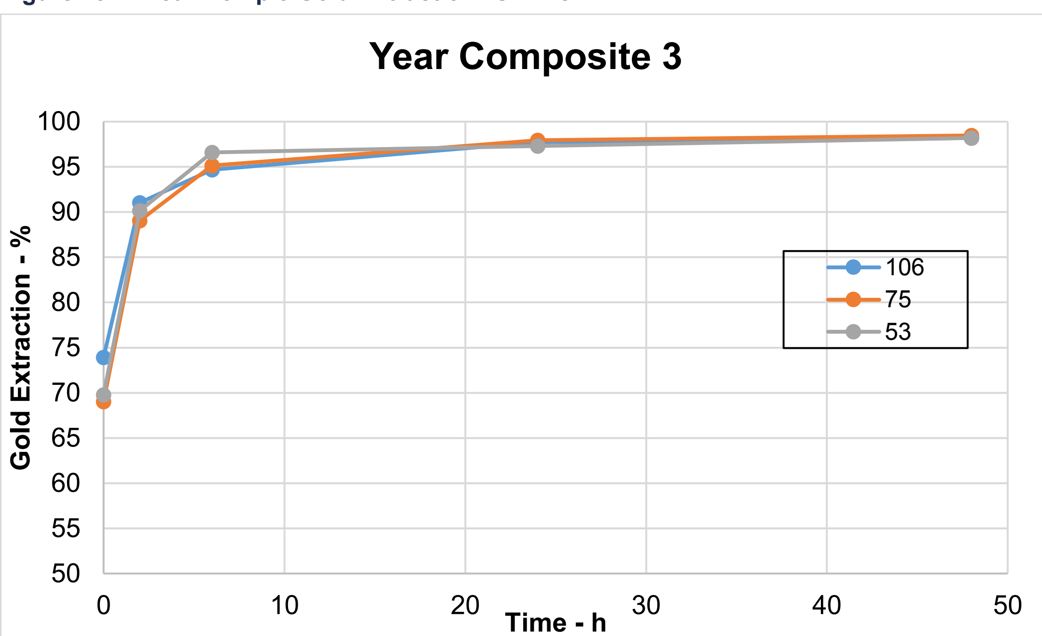
Source: Base Met (2018)

Figure 13-6: Year Comp 2 Gold Extraction vs Time



Source: Base Met (2018)

Figure 13-7: Year Comp 3 Gold Extraction vs Time

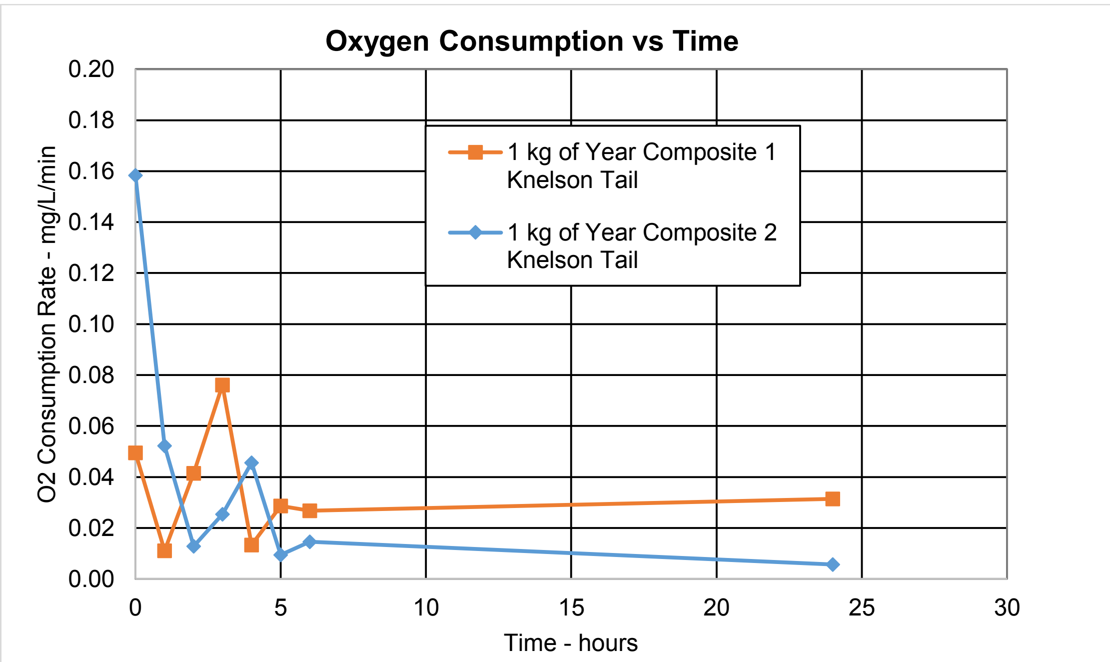


Source: Base Met (2018)

13.2.4 Oxygen Uptake Test Work

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

Figure 13-8: Oxygen Uptake Rate Results



Source: Base Met (2018)

13.2.5 Carbon Adsorption Tests

To determine gold loading rates and constants for carbon type equilibrium carbon loading and sequential triple carbon tests were conducted using pregnant solution from leach tests 74 (Year Comp1) and 77 (Year Comp 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.

13.2.6 Cyanide Destruction Test Results

Feed for the cyanide destruction test work was created from Year 1 and Year 2 composites using cyanide detoxification of the leach tailings slurry using SO_2/Air method. The produced pulp from Year Comp 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 produce the optimized conditions and then tested on Year Comp 2.

Continuous cyanide destruction test work was completed to produce a treated product using the SO_2/Air process, targeting 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 Comp 1.

The cyanide pulp produced during the test program responded well to the SO_2/Air cyanide destruction process, producing a treated pulp with <5 mg/L CN_{WAD}. The results are shown in Table 13-7. The conditions used in test C5 were incorporated into the process design for the cyanide destruction circuit. The results include a SO_2:CN_{WAD} ratio of 5:1 and 30 mg/L copper sulphate for a total of 60 minutes.

Table 13-7: Cyanide Destruction Results

Sample ID	Test No.	Test Parameters				Final Solution Assays				
		PH	Retention Time (min)	SO2 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	0.4	-	-	-	-
	C2	8.5	90.8	6	15	2.6	2.60	1.43	0.10	<0.01
	C3	8.5	88.2	6	30	2.3	7.53	0.63	0.13	<0.01
	C4	8.6	89.1	5	30	3.4	8.00	1.67	0.23	<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: Base Met (2018)

13.2.7 Variability Testing

A total of 30 variability samples (VC) were constructed from McVeigh, Austin, South Austin (including A3 domain) and 8 Zones. The composites were tested using the optimized test conditions to confirm metallurgical response.

13.2.7.1 Head Assays

The samples and head assays are shown below in Table 13-8.

Table 13-8: Variability Composite Head Assays

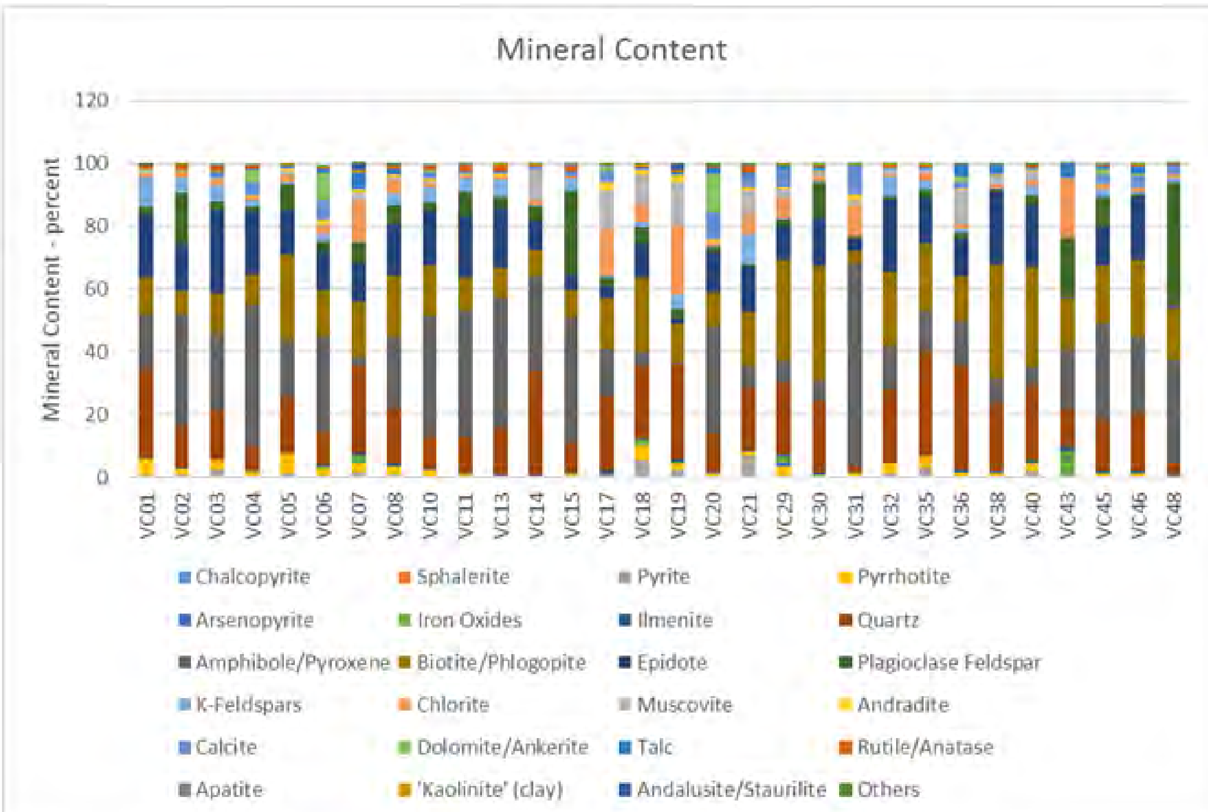
Sample	Assays – percent of g/tonne				
	Au*	Ag*	Fe	S	C
VC 1	14.42	5	5.70	2.66	0.08
VC 2	2.61	1	3.00	1.29	0.12
VC 3	3.77	2	5.55	2.56	0.58
VC 4	2.98	2	3.01	0.74	1.38
VC 5	1.62	2	9.40	2.89	0.08
VC 6	4.74	2	4.55	1.20	4.83
VC 7	12.03	2	42.25	2.12	0.04
VC 8	3.55	1	6.15	1.40	0.10
VC 10	3.55	<1	4.50	0.81	0.59
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
VC31	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

*grams/tonne
Source: Base Met (2018)

13.2.7.2 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.

Figure 13-9: Mineral Content –Variability Composites



Source: Base Met (2018)

13.2.7.3 Comminution Results

BWi tests were conducted at a closing screen sizing of 106 µm indicating that the material is moderately hard. The results ranged between 9.5 kWh/t to 17.1 kWh/t and averaged 13.8 kWh/t. 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 13-9.

Table 13-9: Comminution Results – Variability Composites

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: Base Met (2018)

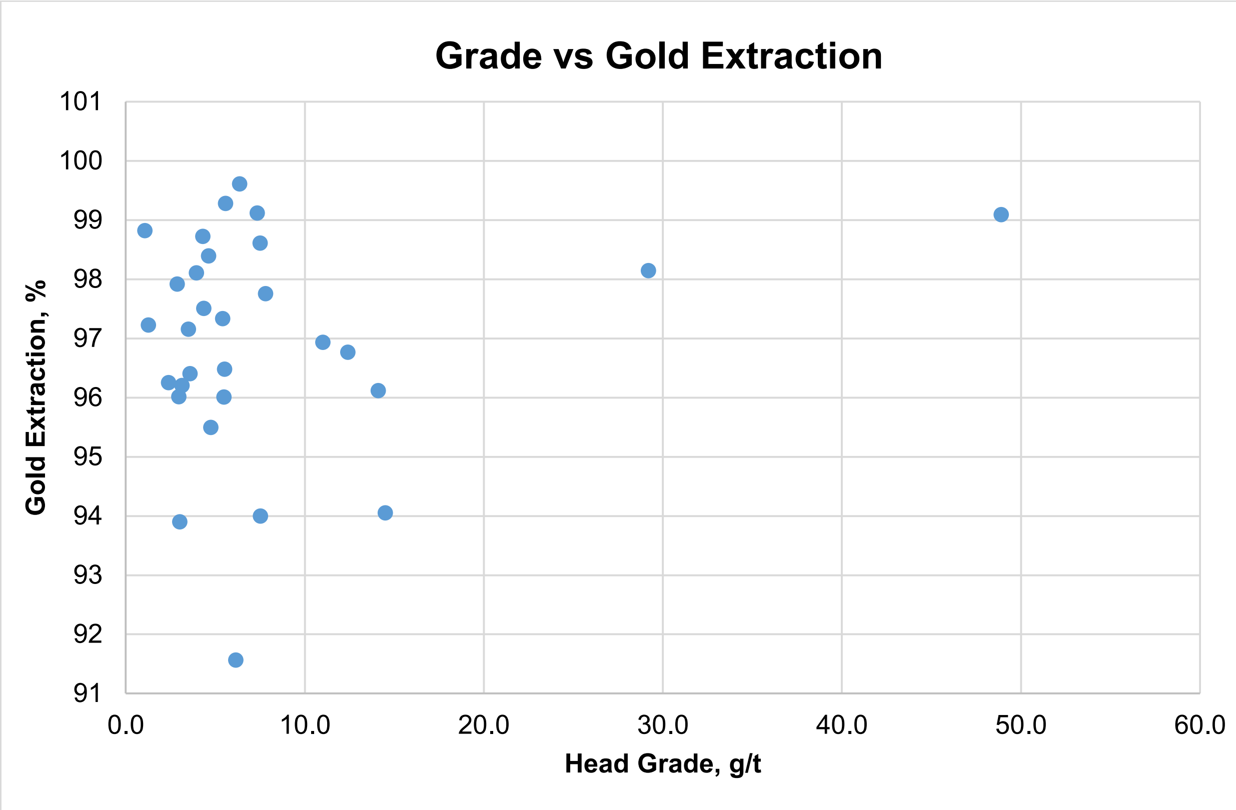
13.2.7.4 Variability Gravity Leach Tests

The variability samples were prepared to a primary P_{80} 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, 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 96.6% and the average recovery to the gravity concentrate was 45.7%. The NaCN and lime consumptions averaged 0.33 g/t and 0.42 g/t, respectively. No correlation was noted between head grade and gold extraction as shown in Figure 13-10.

The results are summarized in Table 13-10 and gold extraction versus time curves are illustrated in Figure 13-11. Gravity gold recovery averaged 45.7% and overall extraction averaged 96.6% for all variability composites.

Figure 13-10: Head Grade vs Gold Extraction – Variability Composites



Source: Base Met (2018)

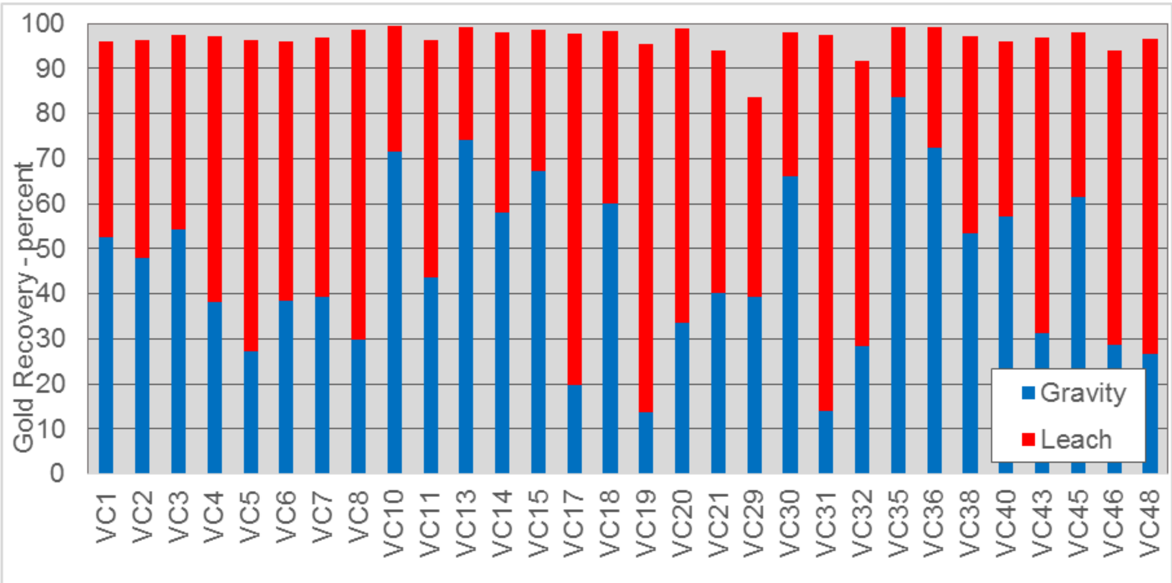
Table 13-10: Gravity Leach Results – Variability Composites

Zone	Variability Composite	ID	Test	Gravity Rec'y	Gold Extraction	Silver Extraction	Consumption (kg/t)		Recalc. Head Grade (g/t)	
				%	48 h	48hr	NaCN	Lime	Au	Ag
McVeigh West Top	PGU-0004	VC1	82	52.5	96.1	54.0	0.31	0.35	14.1	5.10
	PGU-0018	VC2	83	48.0	96.4	22.9	0.17	0.30	3.58	2.10
	PGU-0024	VC3	84	54.4	97.5	58.5	0.28	0.36	4.35	1.90
	PGU-0033	VC4	85	38.2	97.2	18.6	0.28	0.28	3.50	1.50
	PG16-154	VC5	86	27.1	96.3	56.5	0.50	0.44	2.39	1.10
McVeigh West Middle	PG16-098	VC6	87	38.4	96.0	47.7	0.36	0.54	2.97	0.90
	PG16-112	VC7	88	39.4	96.8	76.5	0.29	0.53	12.4	1.70
	PG16-117	VC8	89	29.7	98.7	54.2	0.44	0.43	4.30	1.10
	PG16-138	VC10	90	71.5	99.6	81.7	0.28	0.37	6.36	0.40
	PG16-153	VC11	91	43.5	96.2	14.8	0.24	0.26	3.14	2.30
	PG16-185	VC13	92	74.2	99.1	25.3	0.26	0.27	7.35	0.70
McVeigh West Lower	PG16-253	VC14	93	58.0	98.1	3.1	0.17	0.24	3.94	4.90
	PGU-0044	VC15	94	67.2	98.6	5.8	0.43	0.27	7.51	2.50
McVeigh Central	PG16-056	VC17	95	19.7	97.8	20.3	0.36	0.49	7.80	5.00
	PG16-058-A	VC18	96	59.9	98.4	39.8	0.80	0.69	4.63	2.60
	PG16-058-B	VC19	97	13.8	95.5	28.6	0.80	0.63	4.75	4.50
	PG16-060	VC20	98	33.5	98.8	31.5	0.37	0.52	1.06	0.70
Austin Central	PG16-055	VC21	99	40.1	94.0	63.8	0.41	0.59	14.5	5.50
Austin East	PG16-114	VC29	101	39.3	83.6	68.3	0.32	0.63	7.53	1.60
	PG16-115	VC30	102	66.0	97.9	28.1	0.21	0.34	2.87	0.70
	PG16-227	VC31	103	13.9	97.3	29.8	0.18	0.44	5.42	0.70

Zone	Variability Composite	ID	Test	Gravity Rec'y	Gold Extraction	Silver Extraction	Consumption (kg/t)		Recalc. Head Grade (g/t)	
				%	48 h	48hr	NaCN	Lime	Au	Ag
Austin South Top	PGU-0035	VC32	104	28.3	91.6	88.5	0.34	0.31	6.15	4.40
	PG16-148	VC35	105	83.6	99.1	38.3	0.43	0.47	48.9	3.90
	PG16-150	VC36	106	72.4	99.3	27.6	0.17	0.39	5.57	2.20
	PG16-195	VC38	107	53.3	97.2	10.8	0.29	0.33	1.26	1.80
Austin South Middle	PG16-066	VC40	108	57.2	96.0	42.6	0.24	0.42	5.49	2.10
Austin South Deep	PG17-456	VC43	109	31.2	96.9	34.9	0.24	0.37	11.0	2.40
A3 Upper	PG17-304	VC45	110	61.5	98.1	42.8	0.32	0.49	29.2	6.30
A3 Lower	PG16-282	VC46	111	28.7	93.9	38.6	0.18	0.34	3.02	1.30
	PG17-320	VC48	112	26.8	96.5	57.2	0.14	0.37	5.51	1.90
Average				**45.7**	**96.6**	**40.4**	**0.33**	**0.42**	**8.02**	**2.5**

Source: Base Met (2018)

Figure 13-11: Variability Composite Bottle Roll Leach Results



Source: Base Met (2018)

13.3 Relevant Results

Based on the results from the BaseMet (2018) test program and the existing plant, the process flowsheet will include a single stage of crushing followed by a SAG and ball mill to achieve a P_{80} of 75 µm. Two gravity concentrator will be installed prior to the ball mill circuit to collect any gravity recoverable gold.

Cyclone overflow from the secondary grinding circuit will then be subjected to pre-leach thickening to produce a solids content of 50% prior to leaching. A pre-oxidation stage will be used to oxidize the slurry for 2 hours before 48 hours of cyanide leaching. The leached slurry will then flow through a carbon in pulp circuit to adsorb the gold and silver cyanide complexes onto the pores of activated carbon. The loaded carbon will be processed through an adsorption, desorption and refining circuit. The leached slurry will be pumped to the cyanide destruction circuit to reduce the CN_{WAD} content to below <5 mg/L. The test conditions from Base Met (2018) BL0288 were used to determine the optimized flowsheet, reagent regime and design parameters.

Preliminary estimates of gold recovery are summarized in Table 13-11, providing the basis for the economic analysis presented in Section 22. These projections are based on the results from BaseMet (2018) and the mine schedule (2018).

Table 13-11: Preliminary Recovery Projections

Description	Average Extraction from Test Work Results, Au%	LOM Estimated Weighted Average Recovery, %
Austin	94.7	
South Austin	96.3	
A3	95.5	
McVeigh	96.9	
8 Zone	99.4	
LOM Average (Mine Schedule)	**96.3**	**95.8**

Source: JDS (2018)

14 Mineral Resource Estimate

The current mineral resource estimate of the Madsen Project comprises the Madsen, Fork, Russet South and Wedge deposits. The Madsen mineral resource estimate represents an update of the previous mineral resource disclosed on August 2, 2017 by Pure Gold Mining, which formed the basis for the PEA announced on September 14, 2017. The mineral resources of the Fork and Russet South deposits, represent updates of the mineral resources for these two deposits, first disclosed on December 14, 2017. Finally, the mineral resources from the Wedge deposit, is for the first time disclosed in this study. Details of the Madsen Mineral Resource Estimate are described in Section 14.1, details of the Fork and Russet South deposits' Mineral Resources are described in Section 14-2, and details of the Wedge deposit's Mineral Resource are described in Section 14-3. The current Mineral Resource for the Madsen Gold Project follows a drilling program of 194 holes from surface and underground undertaken by Pure Gold Mining since the PEA study of September 2017.

The zones within the Madsen Mineral Resources include the Austin, South Austin, McVeigh, and the 8 Zone. A separate block model was built for each of these mineral zones, with the A3 domain being part of the South Austin zone block model. All measurements are metric with coordinates in the local metric mine grid.

The estimation of the mineral resources of the Fork and Russet South deposits were also updated in this study. The Fork deposit is located approximately 1.5 km southwest of the Madsen deposit, while the Russet South deposit is located approximately 1.75 km northwest of the Madsen deposit. Separate block models were produced for each deposit. All measurements are metric with coordinates in the UTM system.

This study also presents a first estimation of the mineral resource of the Wedge deposit, located approximately 1.4 km southwest of the Madsen deposit. A separate block model was produced for the Wedge deposit. All measurements are metric with coordinates in the UTM system.

The geologic interpretations were carried out by Pure Gold's personnel while the estimation of gold grades into a mineral resource was carried out by Mr. Marc Jutras, Principal, Mineral Resources at Ginto Consulting Inc. Mr. Jutras is an independent qualified person as defined under National Instrument 43-101.

These mineral resource estimations were primarily undertaken with Maptek™ Vulcan™ software and utilities internally developed in GSLIB-type format. The mineral resource domains were generated in Leapfrog Geo™ software. The following sections outline the procedures undertaken to calculate the mineral resource, first for the Madsen deposit, followed by the Fork and Russet South deposits, and the Wedge deposit.

14.1 Madsen Deposit (Austin, South Austin, McVeigh, 8 Zones)

This section describes the mineral resource estimate for the Madsen gold deposit.

14.1.1 Drill Hole Data

The drill hole database supporting the Madsen mineral resource estimate was provided by Pure Gold with a cut-off date of January 16, 2018. It is comprised of 14,822 holes located within the Madsen Property area with 1,220,041.5 m of drilling. There are 549 holes drilled by Pure Gold from 2014 to 2018 with a total of

180,827.9 m of drilling. Of the 549 holes drilled by Pure Gold, 491 holes were drilled from surface with 176,648.3 m of drilling, and 58 holes were drilled from underground with 12,179.6 m of drilling. All holes are diamond drill holes.

A few changes were made to the original drill hole database:

- 205 historical holes without any Au assays were removed (no logs found, abandoned holes, geotechnical holes);
- Assays with 0.000 g/t Au values were changed to 0.002 g/t Au;
- Assays with -1.000 g/t Au values (missing assays) were changed to 0.005 g/t Au;
- Assays with -0.050 g/t Au values (below detection limit) were changed to 0.025 g/t Au;
- Assays with -0.005 g/t Au values (below detection limit) were changed to 0.003 g/t Au;
- 121 historical holes with logs but no Au assays were considered barren and replaced with values of 0.005 g/t Au;
- 29 historical duplicate holes had their X coordinate increased by 0.05 m. These holes had the same x, y, z, azimuth and dip values but with different names, depths, and Au assays;
- 63 historic drill hole names with more than 12 characters were brought back to 12 characters. The first 9 characters were kept and a “Z” with a 2-digit counter was added;
- Hole 05NM30: a down-hole survey at 0.0 m was added with an azimuth value of -59.31° and dip of -41.0°;
- Hole M1 was removed as no down-hole surveys are available; and
- Holes TB0703, TB0704, TB0705, and TB0175 had their down-hole dips changed from 0.0° to -90°.

14.1.1.1 Drill Hole Data Statistics

The Madsen drill hole database as of January 16, 2018 is comprised of 14,822 drill holes with 711,349 gold assays in grams per tonne. Multi-element analyses are also available for the more recent drilling as well as other geologic information including alteration, lithology, veining, mineralization, structure, magnetic susceptibility, specific gravity, ICP geochemistry, and geotechnical data.

Statistics on the drill hole database are presented in Table 14-1 and in Figure 14-1. As seen in Figure 14-1, the average drill hole depth is 82.3 m, with depths varying from 1.2 m to 2,543.0 m. Most of the underground holes are of short lengths while the surface holes are of much longer lengths. Sample lengths are observed to be 1.63 m on average, with samples lengths varying from 0.01 m to 1,249.5 m, and with the most common sampling length being 1.52 m.

Gold grade statistics on the original samples are presented in Table 14-2 at various cut-off grades. It can be seen from this table that the metres and accumulation (grade x thickness) of gold have similar and consistent decreasing patterns with increasing grade cut-offs. It is also noted that the average gold grades of samples at elevated cut-offs is more than twice the cut-off grade, indicating the presence of a large proportion of higher grade samples.

Table 14-1: Drill Hole Summary

Operator	Years	Number of Holes	Metres	Number of Assays
Russet Red Lake Gold Mines	1944 to 1947	105	8,449.4	2,070
Aiken-Russet Red Lake Mines	1968, 1969, 1974, 1977	46	4,640.9	1,044
Madsen R.L. Gold Mines	1936 to 1976	12,572	675,033.7	456,438
Noranda Inc.	1981, 1982	27	4,994.8	2,539
United Reef Petroleum	1987	24	5,466.6	2,348
Red Lake Buffalo Resources	1988, 1990	18	2,511.2	442
Madsen Gold Corp.	1992 to 1998	556	30,312.2	24,947
Placer Dome Inc.	2001 to 2005	114	60,832.7	32,493
Claude Resources Inc.	1998 to 2012	631	204,837.8	93,403
Pure Gold Mining Inc.	2014 to 2018	549	180,827.9	75,869
Others		180	42,134.3	19,756
Total		14,822	1,220,041.5	711,349

Source: Ginto Consulting Inc. (2019)

Figure 14-1: Statistics on the Madsen Drill Hole Database – January 16, 2018

Collar Data	Number of Data	Mean	Standard Deviation	Coefficient of Variation	Minimum	Lower Quartile	Median	Upper Quartile	Maximum	Number of 0.0 values	Number of < 0.0 values
Easting (X)	14822	4938.19	852.765	0.173	1122.02	4517.11	4838.6	5364.66	11020.3	—	—
Northing (Y)	14822	2420.53	411.998	0.17	78.57	2213.59	2391.91	2516.54	6616.56	—	—
Elevation (Z)	14822	1000.66	353.317	0.353	222.1	746.2	1034.0	1308.13	1541.97	—	—
Hole Depth	14822	82.313	135.3	1.644	1.22	27.95	46.02	76.81	2543.0	—	—
Azimuth	14822	177.987	118.671	0.667	0.01	106.94	180.39	214.58	359.99	—	—
Dip	14822	-6.395	33.503	-5.239	-90.0	-29.27	0.0	0.0	90.0	—	—
Overburden	14822	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	—	—
Survey Data											
Azimuth	59113	171.59	145.387	0.847	0.0	19.58	178.65	336.51	360.0	—	—
Dip	59113	-54.623	13.044	-0.239	-90.0	-62.53	-54.19	-46.68	86.0	—	—
Assay Data											
Interval Length (from-to)	711349	1.627	5.445	3.348	0.01	0.88	1.52	1.53	1249.46	0	0
AU_GPT	711349	1.238	19.249	15.555	0.0	0.003	0.086	0.34	6661.03	887	627

Source: Ginto Consulting Inc. (2019)

Table 14-2: Statistics on Gold Grades of Original Samples

Statistics of Gold Assays Above Cut-Off								
Cut-Off g/t	Total Metres	Increm. Percent	Avg. Au g/t	grd-thk g/t-m	Increm. Percent	Std. Dev.	Coef. of Var.	# of Samples
0.0	1,220,041.5	100.0	1.28	1,561,653.1	100.0	19.273	15.534	711,349
1.0	80,792.9	6.6	7.66	618,873.6	39.6	54.944	5.889	85,414
2.0	49,639.8	4.1	11.65	578,303.7	37.0	67.502	4.978	55,953
3.0	32,299.2	2.6	16.65	537,781.7	34.4	80.877	4.339	38,489
4.0	26,482.4	2.2	19.55	517,730.9	33.2	88.374	4.068	32,005
5.0	20,980.8	1.7	23.52	493,468.4	31.6	98.365	3.774	25,585
6.0	18,175.7	1.5	26.31	478,202.7	30.6	105.138	3.606	22,247
7.0	15,334.2	1.3	29.98	459,719.3	29.4	113.803	3.420	18,828
8.0	13,873.3	1.1	32.34	448,662.5	28.7	119.307	3.316	17,039
9.0	12,136.7	1.0	35.75	433,887.0	27.8	127.274	3.181	14,856
10.0	11,266.1	0.9	37.77	425,520.6	27.2	131.958	3.109	13,757

Source: Ginto Consulting Inc. (2019)

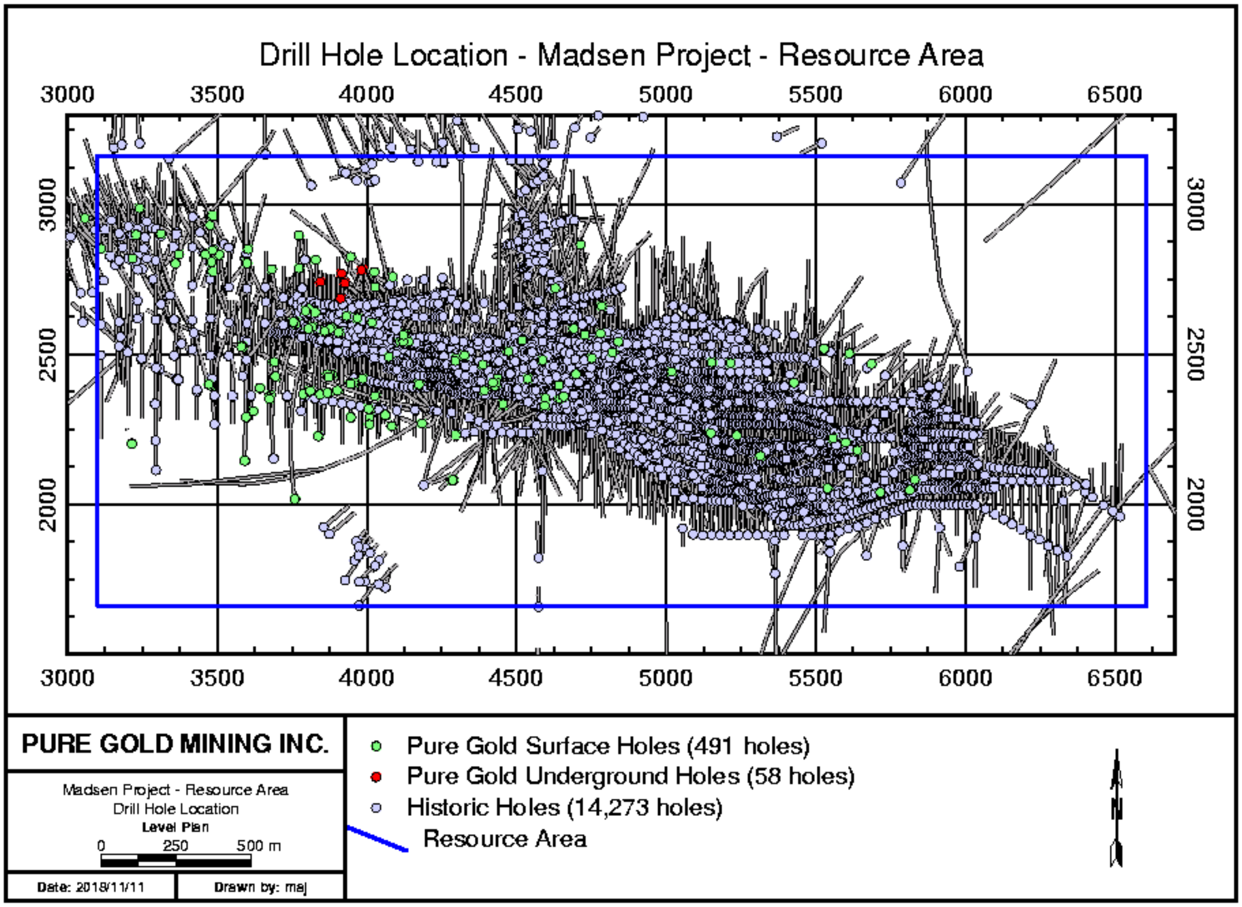
14.1.1.2 Location, Orientation, and Spacing of Drill Holes

The location of the drill holes in the resource area is presented in Figure 14-2 for the Madsen Project area. As seen in this figure, although a large proportion of the drill holes are located within the area of interest, drill holes in surrounding areas are also observed. The latter are however not part of the current study.

Statistics on drill hole spacing are presented in Table 14-3 for each zone within the area of interest and within the high-grade and low-grade units. The overall average drill spacing is 6.4 m in the high-grade zones and 9.5 m in the low-grade zones, while the overall median drill spacing is 6.0 m in the high-grade zones and 6.8 m in the low-grade zones. These results indicate a very tight drill spacing in the area of interest.

With regard to the orientation of the drill holes, although a multitude of orientations is observed, two main orientations of drilling are noted at azimuths of 0° and 180°. Along those orientations dips vary from +90° to -90° (Figure 14-3), which represents the bottom half of a sphere, displays the various azimuth and dip angles of the downward drill holes for the Madsen Project.

Figure 14-2: Drill Hole Location Map – Resource Area



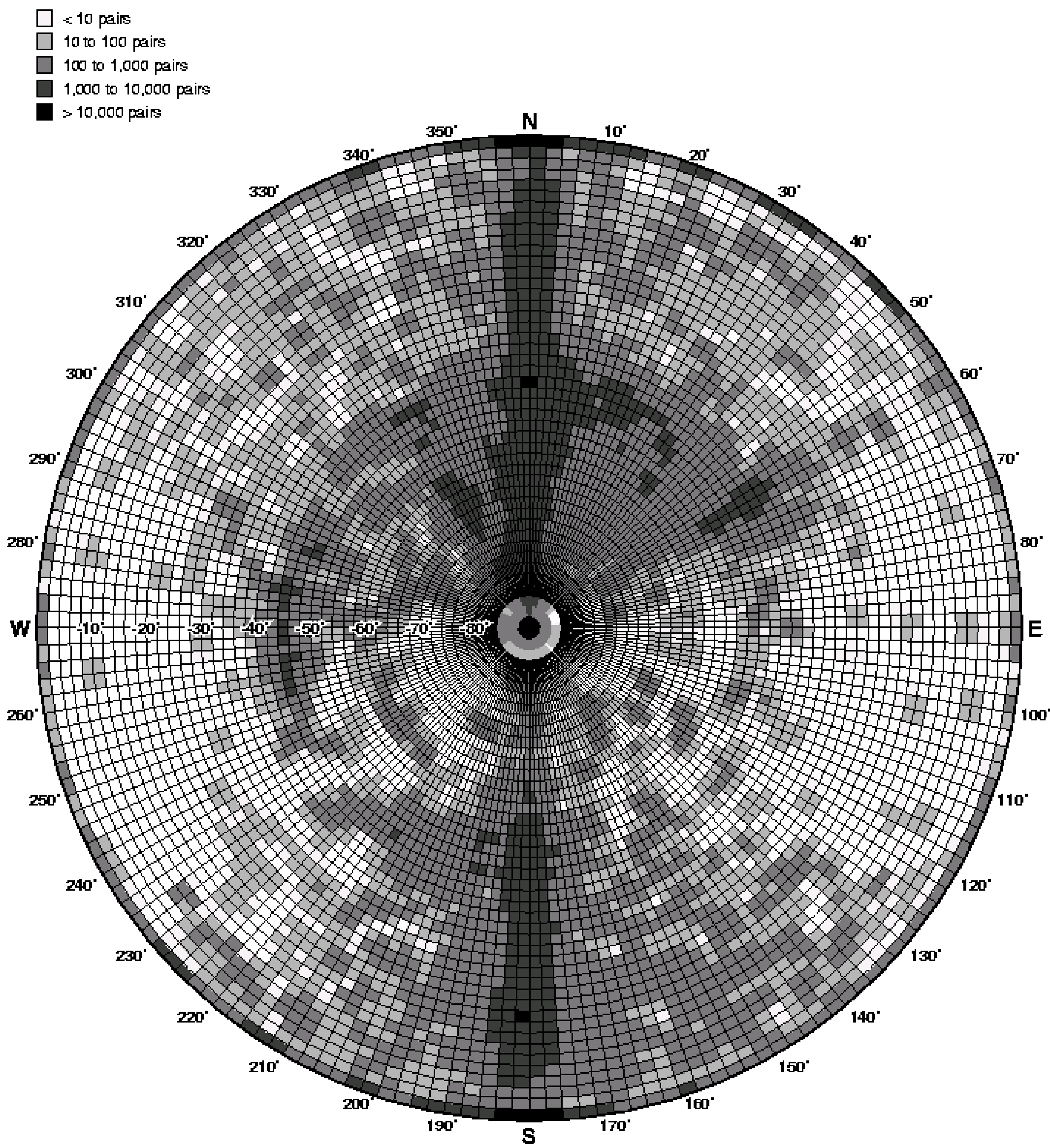
Source: Ginto Consulting Inc. (2019)

Table 14-3: Drill Hole Spacing Statistics

	Mean (m)		Median (m)	
	High-Grade Zone	Low-Grade Zone	High-Grade Zone	Low-Grade Zone
Austin	6.3	7.5	6.1	6.6
South Austin	6.3	7.2	6.0	6.5
A3	8.0	9.8	6.2	7.5
McVeigh	6.9	17.9	5.6	9.6
8 Zone	7.4	-	4.8	-
All	6.4	9.6	6.0	6.8

Source: Ginto Consulting Inc. (2019)

Figure 14-3: Stereonet of Drill Hole Orientations at Madsen – Orientations of Consecutive Pairs in Same Hole



Source: Ginto Consulting Inc. (2019)

14.1.2 Geologic Modelling

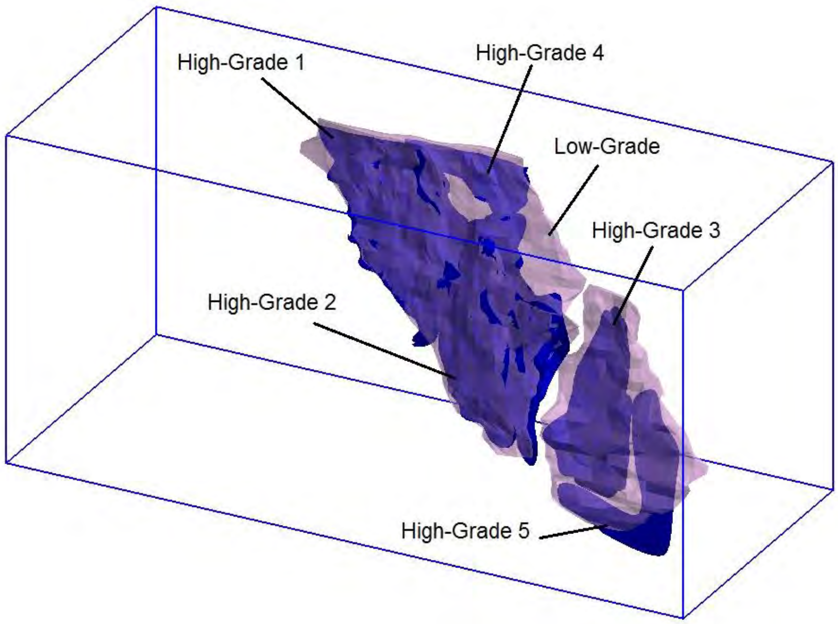
The modelling of the geologic domains at Madsen represent an update from the December 2017 PEA study. A similar approach was utilized for the interpretation process with mineralized zones updated with the latest drill hole information.

The domains modelled include the Austin, the South Austin (including the A3 domain), the McVeigh, and the 8 Zone. Geological domain models were developed by Pure Gold's personnel for each specific zone, including both “high grade domains” and “low grade domains”. A low-grade domain was not modelled for the 8 Zone due to the discrete nature of the quartz vein hosted mineralization in that domain. The geologic domains were built using the recently developed understanding that the mineralization: (i) was emplaced within an early cryptic structure that transects the Balmer stratigraphy, and that this structure has been, (ii) transposed, metamorphosed and annealed during D2 deformation. The continuity of mineralization is therefore restricted to relatively narrow corridors within the ‘SAFZ’ unit (see Section 7) and may be strongly undulating and/or fold repeated. High grade domains have been wireframed to capture this continuity which is defined by a rapid change in gold grades, corresponding in general to a grade of approximately 3 g/t Au. Low grade domains represent the broader alteration halo (SAFZ unit), in which high grade intercepts exist but do not exhibit the same high degree of continuity.

Three-dimensional modelling of the mineral resource domains was performed using Leapfrog Geo™ software; specifically, the Vein Modelling Tool. The high grade domains were defined using drill hole composites of 3 g/t Au and greater for all holes in the database, which served as snapping points for the interpolation of 3-D surfaces to form the hanging wall and footwall contacts of each domain. The outer boundaries of each high grade domain were then manually clipped to an approximate distance of 50 m away from the nearest drill hole to restrict undue extrapolation of any grade estimates. A minimum width of 2 m was applied to each high grade domain where there were no data to show otherwise. Modelling of the low grade domains was performed by the personnel at Equity Exploration Consultants Ltd., who used manually digitized wireframes to primarily capture the logged SAFZ unit, but also rare mineralization within other adjacent units.

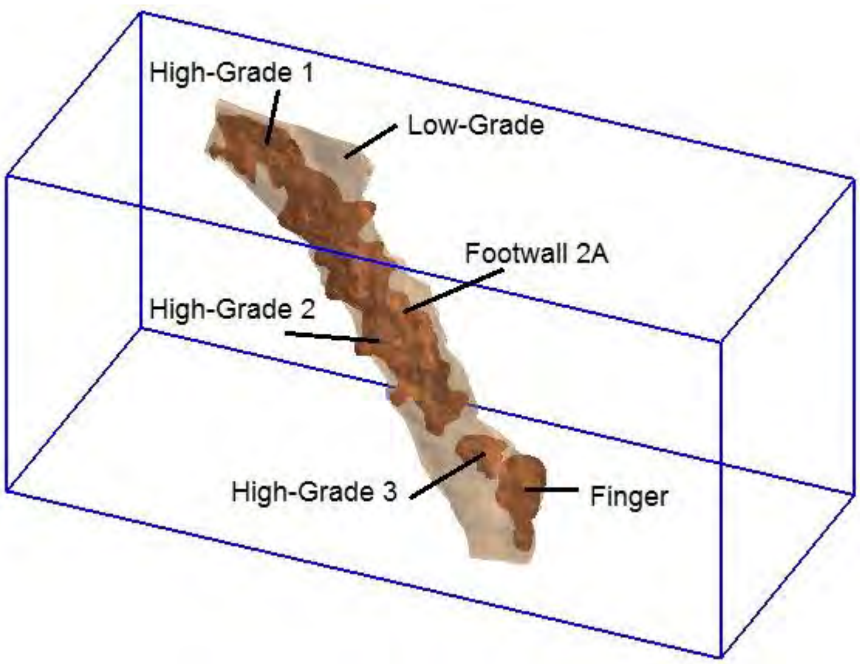
The Austin and South Austin high grade domains are mainly oriented east-west at an azimuth of 095° (mine grid), dipping to the south at -65°, and plunging to the east at approximately -35°. The McVeigh high grade domains are broadly sub-parallel to the Austin domains, but plunge in the opposite direction at about 75°.The high grade 8 Zone is slightly different, with a shape more elongated along a shallower dip of -40°. The low grade domains are similarly oriented along the east-west direction, dipping to the south at -65°, and plunging to the east at -40°. The low grade domains enclose the high grade domains. Examples of the high and low grade domains for each zone are presented in Figures 14-4 to 14-9.

Figure 14-4: Geologic Model of the Austin Domain – Viewed to the Northwest



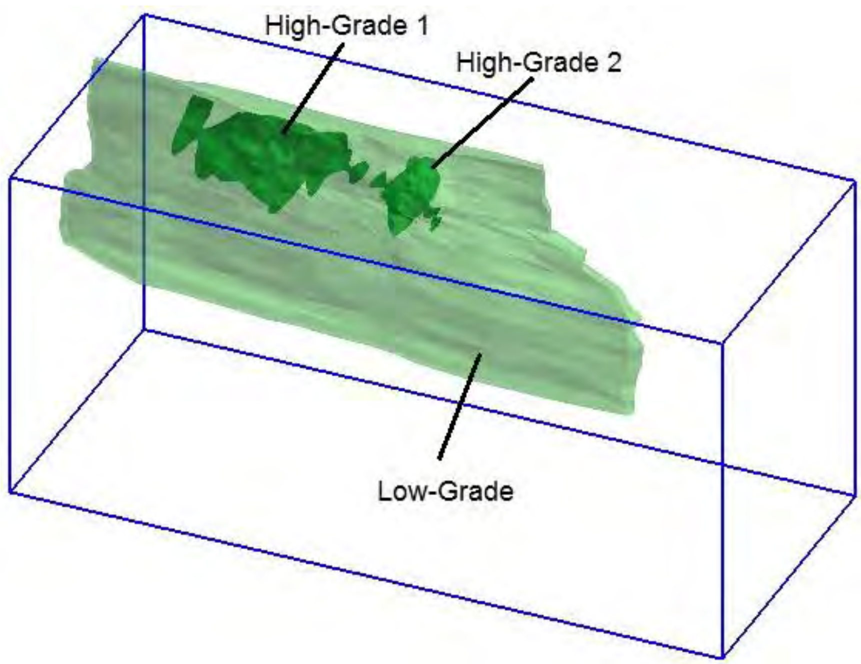
Source: Ginto Consulting Inc. (2019)

Figure 14-5: Geologic Model of the South Austin Domain – Viewed to the Northwest



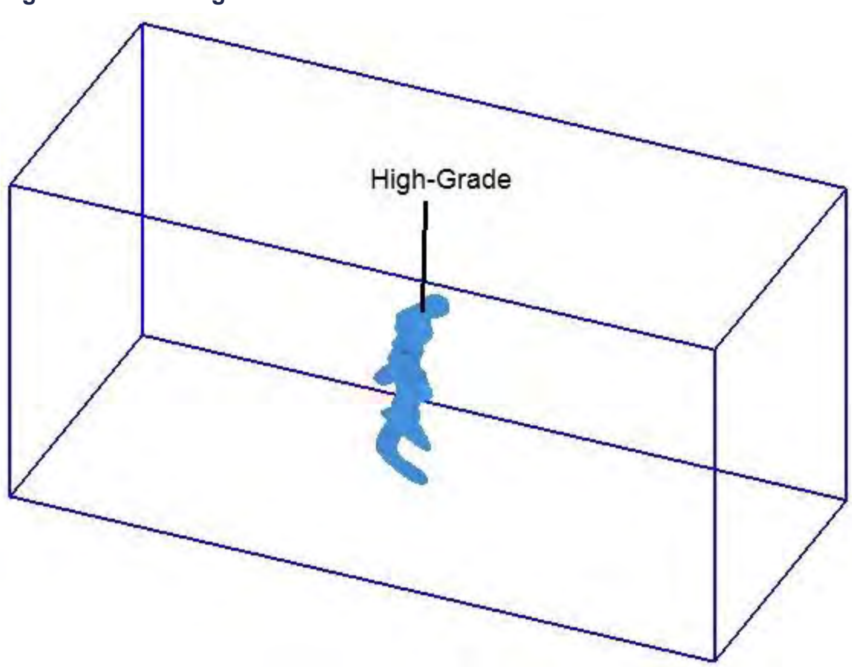
Source: Ginto Consulting Inc. (2019)

Figure 14-6: Geologic Model of the McVeigh Domain – Viewed to the Northwest



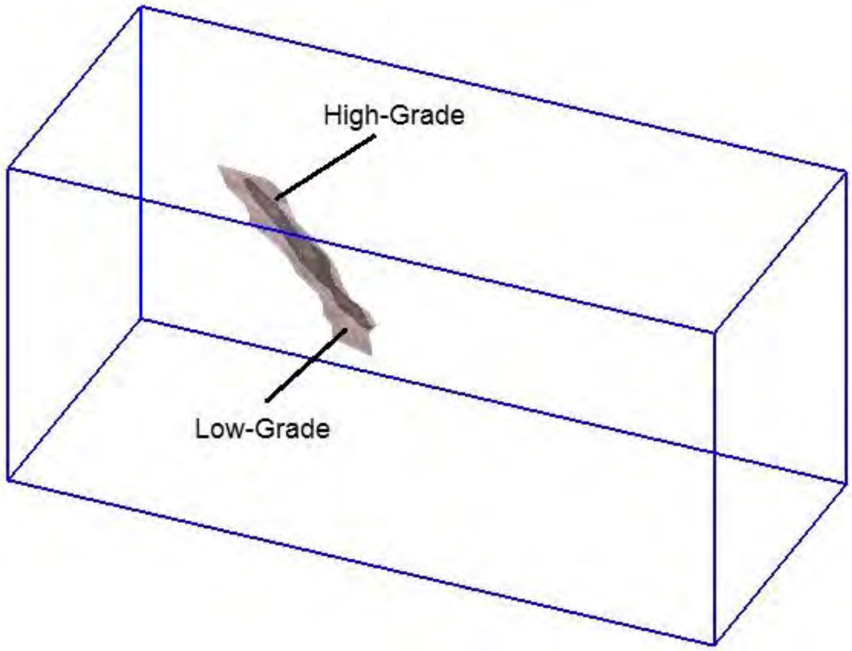
Source: Ginto Consulting Inc. (2019)

Figure 14-7: Geologic Model of the 8 Zone Domain – Viewed to the Northwest



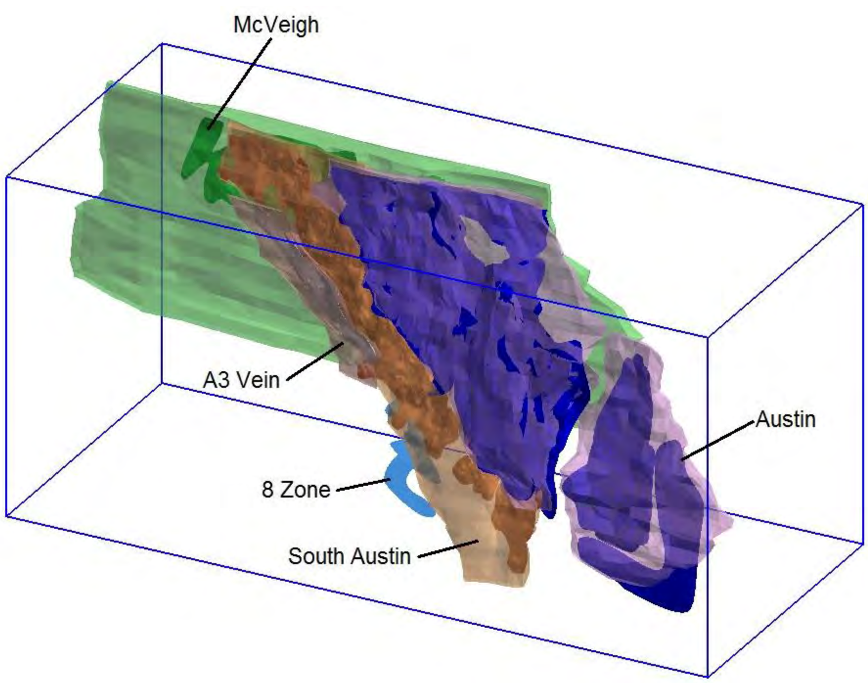
Source: Ginto Consulting Inc. (2019)

Figure 14-8: Geologic Model of the A3 Domain – Viewed to the Northwest



Source: Ginto Consulting Inc. (2019)

Figure 14-9: Geologic Model (All Domains) – Viewed to the Northwest



Source: Ginto Consulting Inc. (2019)

These domains differ significantly from the domain models used in the 2009 SRK mineral resource estimate in that the high grade domains are much narrower. Where the previous model had one high grade domain, the current model has in some cases as many as five discrete high grade domains within the same volume. Average thicknesses of the 14 current high grade wireframes are between 5 m and 10 m, up to a maximum of 30 m; whereas the 2009 model domains were closer to 50 m, on average. A comparison of the different generations of domain models to the mined stopes from historic production shows that the distribution and dimensions of the current domains are a closer approximation to what was mined. This correlation provides a high level of confidence that the current domain model is a reasonable representation of the high grade mineralization at Madsen and is acceptable as a basis for the mineral resource estimate.

14.1.2.1 Geologic Domain Codes

From the modeling of the geologic controls on mineralization, a set of rock codes were defined for each of the zones. Table 14-4 is a list of codes for the zones of interest of the Madsen Project. It should be noted that a new high-grade domain was added to the Austin zone for this resource update: high-grade 5.

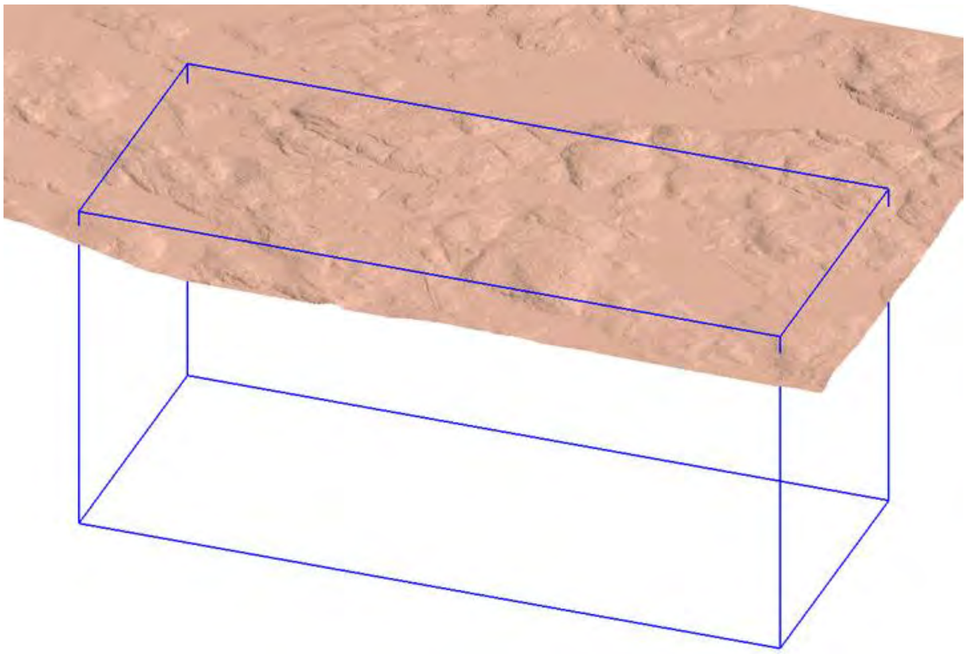
Table 14-4: Geologic Domain Codes for the Madsen Project

Zone	Domain Codes	Description	Volume m^3
Austin	1	high-grade 1	4,099,453.2
	2	high-grade 2	3,476,075.6
	3	high-grade 3	2,006,630.3
	4	high-grade 4	641,459.4
	5	high-grade 5	485,944.6
	6	low-grade	107,382,982.6
South Austin	1	high-grade 1	390,902.6
	2	high-grade 2	2,645,642.2
	3	high-grade 3	171,861.6
	4	finger zone	210,332.9
	5	footwall 2a zone	97,516.8
	6	low-grade	41,517,194.1
McVeigh	1	high-grade 1	1,999,557.4
	2	high-grade 2	426,364.0
	3	low-grade	289,058,656.5
8 Zone	1	high-grade	805,039.8
A3	1	high-grade	181,241.1
	2	low-grade	4,401,567.8

Source: Ginto Consulting Inc. (2019)

The topographic surface at Madsen was obtained by a sub-meter Lidar survey down-sampled to a 5 m resolution and utilized as built for the estimation of the mineral resources. An example of this surface is presented in Figure 14-10.

Figure 14-10: Topographic Surface at Madsen – Viewed to the Northwest



Source: Ginto Consulting Inc. (2019)

14.1.2.2 Dykes

A series of barren post-mineral dykes are observed within the resource area at Madsen. Due to their geometric complexity, it is quite difficult to correlate them from one hole to the next and consequently to model them with wireframes. As an alternative approach, an indicator technique was selected. In this procedure, the dykes from the lithology database were first regrouped into intermediate intrusives (IINT) and mafic intrusives (MINT) by Pure Gold's personnel. An indicator code of 1.0 was assigned for each dyke interval and 0.0 for all others. A histogram of dyke lengths was computed and showed that the most common dyke length was 1.0 m with 40% of the data. The indicator data was then composited to 1.0 m regular intervals. A variographic study was performed on the IINT and MINT composited indicator dyke data with results presented in Table 14-5.

Table 14-5: Variography Results for Indicator Dykes at Madsen

Parameters	01 – intermediate intrusives (IINT)			02 – mafic intrusives (MINT)		
	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	95°	185°	185°	105°	195°	195°
Dip**	0°	-65°	25°	0°	-45°	45°
Nugget Effect C_0	0.249			0.123		
1st Structure C_1	0.126			0.064		
2nd Structure C_2	0.202			0.274		
1st Range A_1	13.8 m	12.4 m	5.2 m	48.5 m	57.1 m	88.9 m
2nd Range A_2	125.0 m	90.3 m	35.5 m	141.0 m	111.0 m	116.0 m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

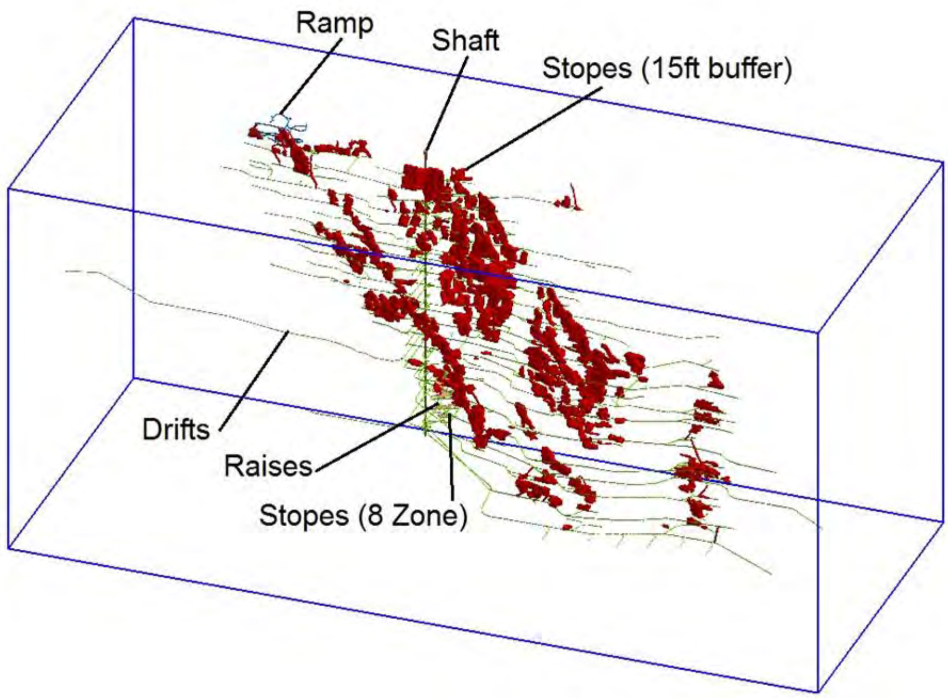
The indicator dyke composites were then estimated within the mineralized zones of interest of the resource area. An ordinary kriging interpolation method was utilized to estimate dyke proportions into a block model corresponding to that of the gold grade estimates (see Section 14.1.6).

A minimum of two samples and maximum of 12 samples were utilized to calculate an estimate. The search ellipsoid was dimensioned and oriented according to the variogram parameters for each type of dyke. The resulting estimate represents a proportion of dyke within each block with values varying from 0.0 (no dyke) to 1.0 (all dyke). These estimates of dyke proportions were kept to later edit the block model of gold grade estimates.

14.1.2.3 Underground Mined Voids

The original wireframes of the underground mined voids from the 2009 resource estimate by Claude Resources were utilized in the current study, as no new underground development was carried out since then. The voids were grouped into 5 separate units as follows: stopes, drifts, shafts, raises, and ramps. The stope wireframes were expanded 5 m in all directions to provide a geotechnical buffer in order to address the more degraded condition of the underground stopes. No geotechnical buffer was developed for the 8 Zone due to the more discrete nature of the quartz vein-hosted mineralization. For each set of wireframes, a fraction value representing the proportion of the block inside the wireframe was calculated and stored for each block of the block model. These values were kept to later edit the block model of gold grade estimates. The underground voids are displayed in Figure 14-11.

Figure 14-11: Underground Mined Voids at Madsen – Viewed to the Northwest



Source: Ginto Consulting Inc. (2019)

14.1.3 Compositing

Statistics were computed on the original sample lengths and it was noted that the most common sample length for the Austin, South Austin (including A3 domain), and McVeigh zones is 1.52 m (5ft), with 45% of the data. For the 8 Zone, statistics on the sample length show that the most common sampling length is 0.3 m, with 23% of the data.

For the Austin, South Austin (including A3 domain), and McVeigh zones, the compositing length was set at 1.52 m to reflect the most common sampling length, as well as providing a satisfactory ratio of sample length to block height (1:2). For the 8 Zone the most common sampling length of 0.3 m represents a low ratio of sample length to block height of 1:10 and for such a compositing length of 0.60 m, representing a multiple of 0.3 m and a ratio of 1:5, was selected.

The compositing process consisted in starting the compositing at the collar of each hole with continuous composite intervals. At the contact with a different unit from the geology model, a last interval was composited, while a new set of regular composite lengths is generated within the other unit. Within the Austin, South Austin, McVeigh, and 8 Zones, a total of 418,432 composites were generated from 13,258 holes. A summary of statistics on the composites at Madsen is presented in Table 14-6.

Table 14-6: Drill Hole Composites Summary at Madsen

Company	# of Holes	%	# of Composites	%	# of Metres	%	Average Au Grade g/t
Austin	8,253	62.2	229,169	54.8	316,868.5	54.5	1.16
South Austin	4,098	30.9	102,513	24.5	142,824.8	24.6	1.16
A3	366	2.8	5,344	1.3	7,440.5	1.3	0.48
McVeigh	1,454	11.0	77,723	18.5	111,787.5	19.2	0.40
8 Zone	223	1.7	3,683	0.9	2,079.2	0.4	8.40
All	13,258	100.0	418,432	100.0	581,000.5	100.0	1.07

Source: Ginto Consulting Inc. (2019)

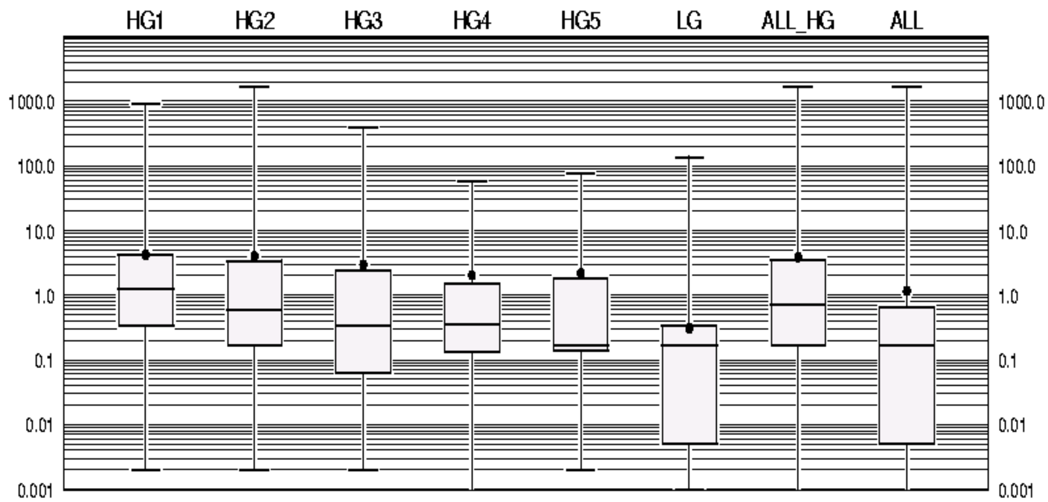
14.1.4 Exploratory Data Analysis (EDA)

A set of various statistical applications was utilized to provide a better understanding of the gold grade populations within the various mineralized zones.

14.1.4.1 Univariate Statistics

Basic statistics were performed on the gold grades of the Austin, South Austin (including A3 domain), McVeigh, and 8 Zone composites. Histograms and probability plots indicated that the gold grade distributions resemble positively skewed lognormal populations. Basic statistics results are presented as boxplots per unit for each zone in Figure 14-12 to Figure 14-16. As seen in these figures, the gold grade populations are more heterogeneous with coefficients of variation (CV) greater than 3.0 in many of the units. This is most likely attributable to high gold grade values found in these unit.

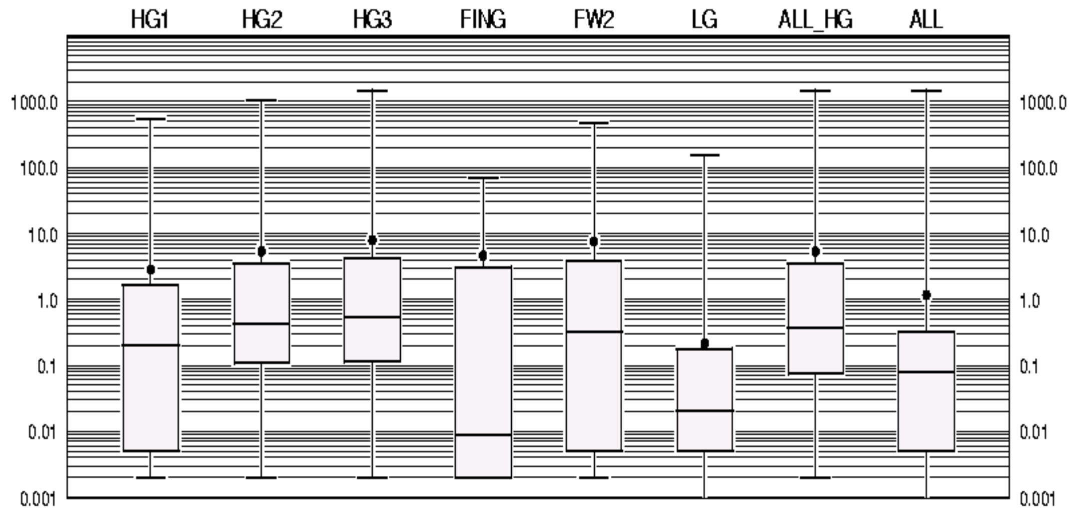
Figure 14-12: Basic Statistics of Gold – Austin Zone 1.52m Composites in g/t HG and LG Domains



	HG1	HG2	HG3	HG4	HG5	LG	ALL_HG	ALL	
Number of data	26710	19232	10608	1066	279	171274	57895	229169	Number of data
Mean	4.204	4.017	2.959	2.032	2.17	0.306	3.876	1.155	Mean
Std. Dev.	14.363	17.906	9.42	4.788	5.509	0.788	14.823	7.422	Std. Dev.
Coef. of Var.	3.416	4.457	3.183	2.356	2.539	2.58	3.824	6.424	Coef. of Var.
Maximum	911.559	1648.33	391.64	57.881	77.061	136.46	1648.33	1648.33	Maximum
Upper quartile	4.11	3.375	2.379	1.516	1.801	0.34	3.518	0.642	Upper quartile
Median	1.243	0.591	0.34	0.361	0.17	0.17	0.716	0.17	Median
Lower quartile	0.34	0.17	0.064	0.131	0.14	0.005	0.17	0.005	Lower quartile
Minimum	0.002	0.002	0.002	0.0	0.002	0.001	0.0	0.0	Minimum
Number of holes	3214	2655	1343	242	102	8081	7459	8253	Number of holes

Source: Ginto Consulting Inc. (2019)

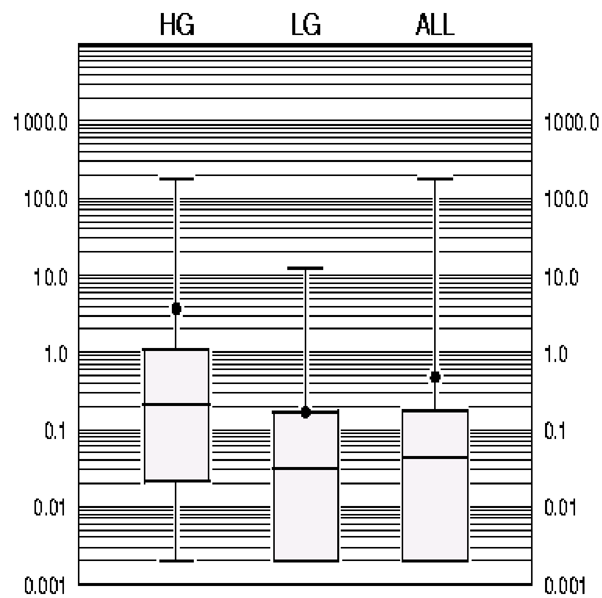
Figure 14-13: Basic Statistics of Gold – South Austin Zone 1.52m Composites in g/t HG and LG Domains



	HG1	HG2	HG3	FING	FW2	LG	ALL_HG	ALL	
Number of data	2346	15622	1289	334	966	81956	20557	102513	Number of data
Mean	2.84	5.354	7.843	4.549	7.558	0.213	5.326	1.164	Mean
Std. Dev.	12.487	21.001	51.835	10.94	26.171	0.929	23.658	10.428	Std. Dev.
Coef. of Var.	4.397	3.922	6.609	2.405	3.463	4.359	4.442	8.962	Coef. of Var.
Maximum	543.77	1038.76	1470.65	70.175	461.939	152.279	1470.65	1470.65	Maximum
Upper quartile	1.649	3.464	4.205	3.09	3.82	0.179	3.43	0.324	Upper quartile
Median	0.198	0.422	0.549	0.009	0.32	0.02	0.364	0.078	Median
Lower quartile	0.005	0.108	0.113	0.002	0.005	0.005	0.074	0.005	Lower quartile
Minimum	0.002	0.002	0.002	0.002	0.002	0.0	0.002	0.0	Minimum
Number of holes	604	2335	205	106	287	4054	3295	4098	Number of holes

Source: Ginto Consulting Inc. (2019)

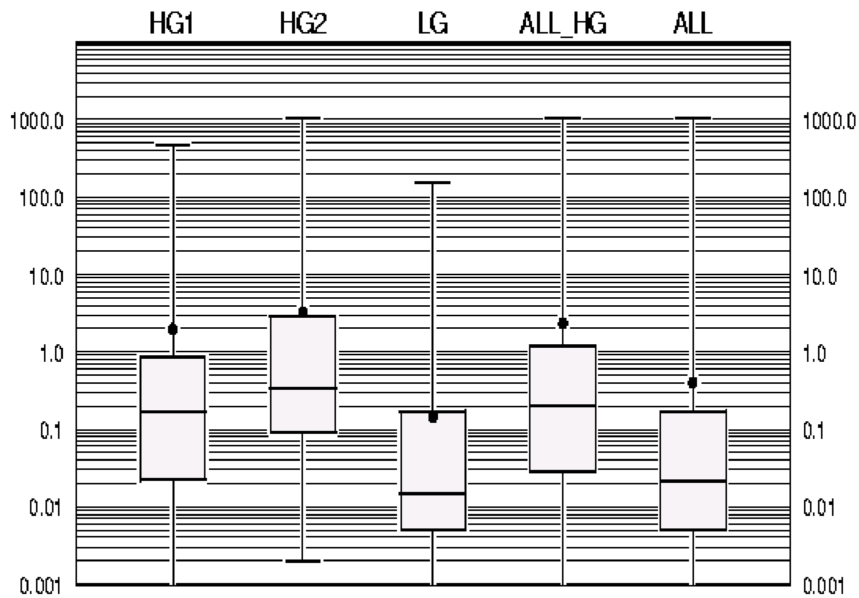
Figure 14-14: Basic Statistics of Gold – A3 Domain 1.52m Composites in g/t HG and LG domains



	HG	LG	ALL	
Number of data	546	4798	5344	Number of data
Mean	3.657	0.168	0.476	Mean
Std. Dev.	14.813	0.366	4.52	Std. Dev.
Coef. of Var.	4.05	2.173	9.504	Coef. of Var.
Maximum	178.29	12.187	178.29	Maximum
Upper quartile	1.088	0.17	0.173	Upper quartile
Median	0.214	0.031	0.043	Median
Lower quartile	0.021	0.002	0.002	Lower quartile
Minimum	0.002	0.002	0.002	Minimum
Number of holes	201	366	366	Number of holes

Source: Ginto Consulting Inc. (2019)

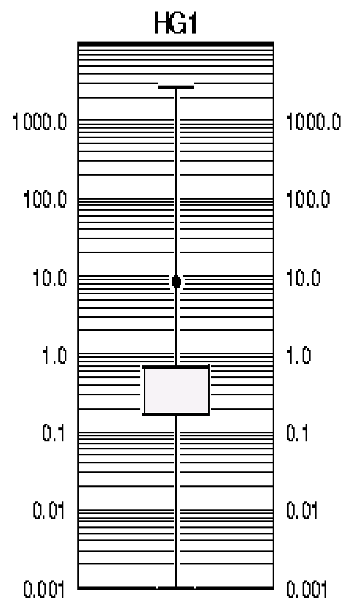
Figure 14-15: Basic Statistics of Gold – McVeigh Zone 1.52m Composites in g/t HG and LG Domains



	HG1	HG2	LG	ALL_HG	ALL	
Number of data	7124	2858	67741	9982	77723	Number of data
Mean	1.961	3.264	0.144	2.329	0.399	Mean
Std. Dev.	10.878	22.751	0.753	15.213	5.288	Std. Dev.
Coef. of Var.	5.546	6.969	5.225	6.531	13.257	Coef. of Var.
Maximum	477.287	1049.9	150.86	1049.9	1049.9	Maximum
Upper quartile	0.856	2.925	0.17	1.194	0.17	Upper quartile
Median	0.17	0.34	0.015	0.204	0.021	Median
Lower quartile	0.022	0.092	0.005	0.028	0.005	Lower quartile
Minimum	0.0	0.002	0.0	0.0	0.0	Minimum
Number of holes	669	382	1450	1051	1454	Number of holes

Source: Ginto Consulting Inc. (2019)

Figure 14-16: Basic Statistics of Gold – 8 Zone 0.6 m Composites in g/t HG and LG Domains



Number of data	3683	Number of data
Mean	8.395	Mean
Std. Dev.	75.428	Std. Dev.
Coef. of Var.	8.985	Coef. of Var.
Maximum	2680.41	Maximum
Upper quartile	0.69	Upper quartile
Median	0.17	Median
Lower quartile	0.17	Lower quartile
Minimum	0.001	Minimum
Number of holes	223	Number of holes

Source: Ginto Consulting Inc. (2019)

14.1.4.2 Capping of High-Grade Outliers

It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results from specific statistical utilities. This procedure is performed on high-grade values that are considered outliers and that cannot be related to any geologic feature. In the case at Madsen, the higher gold grades were examined with three different tools: the probability plot, decile analysis, and cutting statistics. The usage of various investigating methods allows for a selection of the capping threshold in a more objective and justified manner. For the probability plot method, the capping value is chosen at the location where higher grades depart from the main distribution. For the decile analysis, the capping value is chosen as the maximum grade of the decile containing less than an average of 10% of metal. For the cutting statistics, the selection of the capping value is identified at the cut-off grade where there is no correlation between the grades above this cut-off. The resulting compilation of the capping thresholds is listed in Table 14-7. One of the objectives of the capping strategy is to have less than 10% of the metal affected by the capping process. This was achieved in most of the cases, however in some instances it was noted that the capping had a greater effect on the metal content, indicating that few higher-grade outliers were quite different than the population in general by carrying a good proportion of the metal content.

Table 14-7: List of Capping Thresholds of Higher Gold Grade Outliers

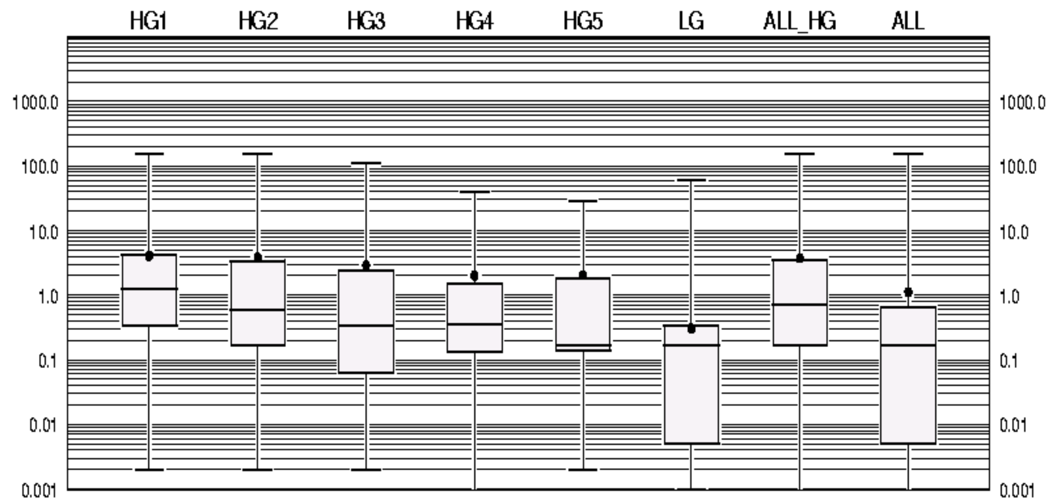
Domain	Units	Capping Threshold g/t	% Metal Affected	Number of Comps Capped
Austin	high-grade 1	150.0	4.0	24
	high-grade 2	150.0	4.0	26
	high-grade 3	110.0	3.0	16
	high-grade 4	40.0	1.0	5
	high-grade 5	28.0	9.0	2
	low-grade	60.0	1.0	5
South Austin	high-grade 1	80.0	9.0	6
	high-grade 2	250.0	3.0	16
	high-grade 3	180.0	21.0	4
	finger zone	40.0	6.0	9
	footwall 2a zone	180.0	5.0	3
	low-grade	60.0	2.0	4
A3	high-grade	80.0	9.0	6
	low-grade	60.0	0 .0	0
McVeigh	high-grade 1	100.0	8.0	9
	high-grade 2	90.0	12.0	5
	low-grade	15.0	5 .0	8
8 Zone	high-grade	450.0	22.0	14

Source: Ginto Consulting Inc. (2019)

Basic statistics were re-computed with the gold grades capped to the thresholds listed in Table 14-7. Boxplots of Figure 14-17 to Figure 14-21 display the basic statistics resulting from the capping of the higher gold grade outliers. It can be observed from those figures that the coefficients of variation are in general below or close to 3.0 for the different gold grade populations. However, a few units display a coefficient greater than 3.0, as seen for the LG unit at South Austin, the HG unit at A3, the HG1 unit at McVeigh, and the HG unit at 8 Zone. The effect of the capping of higher gold grade outliers has slightly reduced the overall mean gold grade of the high-grade domains by 4.0% at Austin, by 6.1% at South Austin, by 13.8% at A3, by 10.9% at McVeigh, and by 24.2% at 8 Zone. The greater reduction observed at 8 Zone is due to a few high-grade outliers carrying a large portion of the metal and thus having a greater influence on the population's average gold grade.

Because of the generally low coefficients of variation observed for the gold grade populations of the major units at Madsen, it was concluded that there is no need to treat the higher-grade composites differently than the lower grade composites during the estimation process. Ordinary kriging on capped composites was thus selected as a well-suited estimation technique in this case.

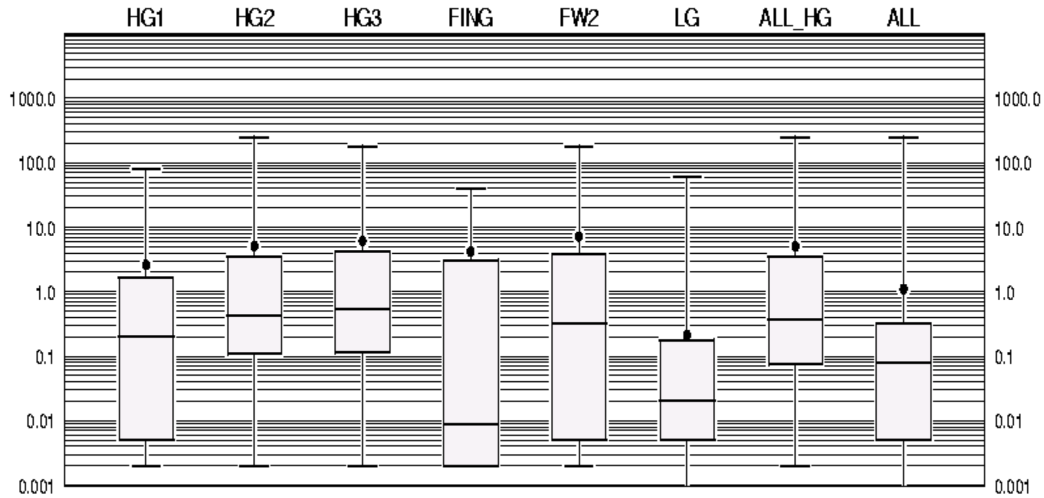
Figure 14-17: Basic Statistics of Capped Gold Grades – Austin 1.52 m Composites in g/t – HG and LG Zones



	HG1	HG2	HG3	HG4	HG5	LG	ALL_HG	ALL	
Number of data	26710	19232	10608	1066	279	171274	57895	229169	Number of data
Mean	4.046	3.821	2.875	2.013	2.041	0.305	3.721	1.118	Mean
Std. Dev.	9.537	10.55	7.779	4.619	4.294	0.778	9.54	4.921	Std. Dev.
Coef. of Var.	2.357	2.761	2.705	2.294	2.104	2.546	2.564	4.401	Coef. of Var.
Maximum	150.0	150.0	110.0	40.0	28.0	60.0	150.0	150.0	Maximum
Upper quartile	4.11	3.375	2.379	1.516	1.801	0.34	3.518	0.642	Upper quartile
Median	1.243	0.591	0.34	0.361	0.17	0.17	0.716	0.17	Median
Lower quartile	0.34	0.17	0.064	0.131	0.14	0.005	0.17	0.005	Lower quartile
Minimum	0.002	0.002	0.002	0.0	0.002	0.001	0.0	0.0	Minimum
Number of holes	3214	2655	1343	242	102	8081	7459	8253	Number of holes

Source: Ginto Consulting Inc. (2019)

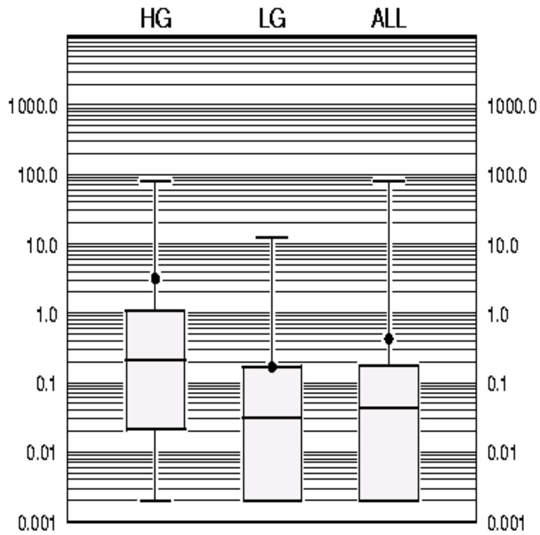
Figure 14-18: Basic Statistics of Capped Gold – South Austin - 1.52 m Composites in g/t – HG and LG Zones



	HG1	HG2	HG3	FING	FW2	LG	ALL_HG	ALL	
Number of data	2346	15622	1289	334	966	81956	20557	102513	Number of data
Mean	2.612	5.14	6.067	4.151	7.135	0.212	4.999	1.101	Mean
Std. Dev.	7.518	15.6	16.801	9.119	20.921	0.814	15.256	6.872	Std. Dev.
Coef. of Var.	2.879	3.035	2.769	2.197	2.932	3.84	3.052	6.241	Coef. of Var.
Maximum	80.0	250.0	180.0	40.0	180.0	60.0	250.0	250.0	Maximum
Upper quartile	1.649	3.464	4.205	3.09	3.82	0.179	3.43	0.324	Upper quartile
Median	0.198	0.422	0.549	0.009	0.32	0.02	0.364	0.078	Median
Lower quartile	0.005	0.108	0.113	0.002	0.005	0.005	0.074	0.005	Lower quartile
Minimum	0.002	0.002	0.002	0.002	0.002	0.0	0.002	0.0	Minimum
Number of holes	604	2335	205	106	287	4054	3295	4098	Number of holes

Source: Ginto Consulting Inc. (2019)

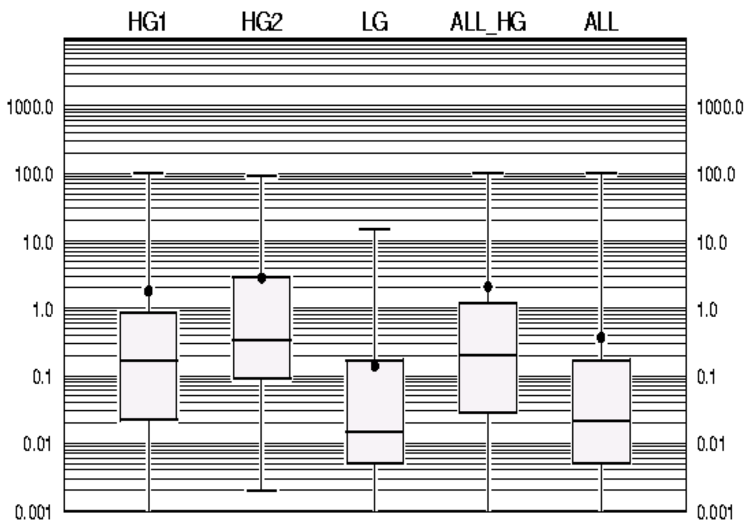
Figure 14-19: Basic Statistics of Capped Gold – A3 1.52 m Composites in g/t – HG and LG Zones



	HG	LG	ALL	
Number of data	546	4798	5344	Number of data
Mean	3.151	0.168	0.431	Mean
Std. Dev.	10.406	0.366	3.221	Std. Dev.
Coef. of Var.	3.303	2.173	7.474	Coef. of Var.
Maximum	80.0	12.187	80.0	Maximum
Upper quartile	1.088	0.17	0.173	Upper quartile
Median	0.214	0.031	0.043	Median
Lower quartile	0.021	0.002	0.002	Lower quartile
Minimum	0.002	0.002	0.002	Minimum
Number of holes	201	366	567	Number of holes

Source: Ginto Consulting Inc. (2019)

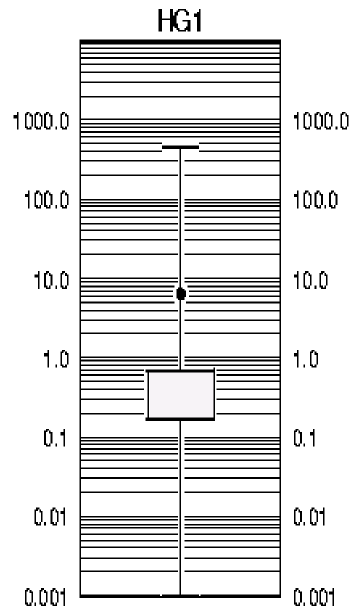
Figure 14-20: Basic Statistics of Capped Gold – McVeigh 1.52 m Composites in g/t – HG and LG Zones



	HG1	HG2	LG	ALL_HG	ALL	
Number of data	7124	2858	67741	9982	77723	Number of data
Mean	1.792	2.792	0.14	2.074	0.366	Mean
Std. Dev.	6.858	6.946	0.373	6.897	2.461	Std. Dev.
Coef. of Var.	3.828	2.488	2.654	3.325	6.727	Coef. of Var.
Maximum	100.0	90.0	15.0	100.0	100.0	Maximum
Upper quartile	0.856	2.925	0.17	1.194	0.17	Upper quartile
Median	0.17	0.34	0.015	0.204	0.021	Median
Lower quartile	0.022	0.092	0.005	0.028	0.005	Lower quartile
Minimum	0.0	0.002	0.0	0.0	0.0	Minimum
Number of holes	669	382	1450	1051	1454	Number of holes

Source: Ginto Consulting Inc. (2019)

Figure 14-21: Basic Statistics of Capped Gold – 8 Zone 0.6 m Composites in g/t – HG Zone



Number of data	3683	Number of data
Mean	6.36	Mean
Std. Dev.	37.168	Std. Dev.
Coef. of Var.	5.844	Coef. of Var.
Maximum	450.0	Maximum
Upper quartile	0.69	Upper quartile
Median	0.17	Median
Lower quartile	0.17	Lower quartile
Minimum	0.001	Minimum
Number of holes	223	Number of holes

Source: Ginto Consulting Inc. (2019)

14.1.4.3 Declustering

In general, there is a tendency to drill more holes in higher grade areas than in lower grade areas when delimiting a potential ore body. As a result, the higher grade portion of a deposit will be overly represented and would translate into a bias towards the higher grades when calculating statistical parameters of the population. Thus, a declustering method is utilized to generate a more representative set of statistical results within the zone of interest. In this case, a polygonal declustering technique was applied to the composites of the high-grade zones of Austin, South Austin (including A3 domain), McVeigh, and 8 Zone. This approach consists of assigning the volume of a polygon, defined by the halfway distance between a sample and its surrounding neighbours, as a weight for each sample within the high-grade mineralized zone. Therefore, a sample that is isolated will have a larger weight than a sample located in a densely sampled area.

Comparisons of average gold capped and declustered grades with the capped and un-declustered gold averages show little clustering overall with only slight decreases or increases in average declustered grades. The regular pattern of the tight underground definition drilling is most likely responsible for the limited clustering observed. A reduction of 9.6% of the mean gold grade of the high-grade domains was observed at South Austin and A3, while increases of 1.3%, and 8.5% were noted at McVeigh, and 8 Zone, respectively. No changes were observed at Austin.

The average grade from the declustered statistics provides an excellent comparison with the average grade of the interpolated blocks, as a way to assess any overall bias of the estimates.

14.1.5 Variography

A variographic analysis was carried out on the gold grade composites within the different geologic domain units at Austin, South Austin, McVeigh, 8 Zone, and A3. The objective of this analysis was to spatially establish the preferred directions of gold grade continuity. In turn, the variograms modeled along those directions would be later utilized to select and weigh the composites during the block grade interpolation process. For this exercise, all experimental variograms were of the type relative lag pairwise, which is considered robust for the assessment of gold grade continuity.

Variogram maps were first calculated to examine general gold grade continuities in the XY, XZ, and YZ planes. The next step undertaken was to compute omni-directional variograms and down-hole variograms. The omni-directional variograms are calculated without any directional restrictions and provide a good assessment of the sill of the variogram. As for the down-hole variogram, it is calculated with the composites of each hole along the trace of the hole. The objective of these calculations is to provide information about the short scale structure of the variogram, as the composites are more closely spaced down the hole. Thus, the modeling of the nugget effect is usually better derived from the down-hole variograms.

Directional variograms were then computed to identify more specifically the three main directions of continuity. A first set of variograms were produced in the horizontal plane at increments of 10 degrees. In the same way a second set of variograms were computed at 10° increments in the vertical plane of the horizontal direction of continuity (plunge direction). A final set of variograms at 10° increments were calculated in the vertical plane perpendicular to the horizontal direction of continuity (dip direction). The final variograms were then modeled with a 2-structure spherical variogram, and resulting parameters presented in Table 14-8 to Table 14-12 for gold populations of the different zones.

The directions of gold grade continuity are in general agreement with the orientation of the mineralized zone, with best directions of continuity trending east-west and down-dip at approximately -65°. The ranges of gold grade continuity along the principal direction (strike) vary from 28 m to 47 m in the high-grade units and from 36 m to 55 m in the low-grade units. Along the minor direction (dip), the ranges of continuity vary from 26 m to 41 m in the high-grade units and from 40 m to 50 m in the low-grade units. Finally, along the vertical direction (across strike and dip), the ranges of continuity vary from 11 m to 21 m in the high-grade units and from 29 m to 31 m in the low-grade units. The modeled variograms have relatively low nugget effects with values varying from 8% to 27% of the sill for the high-grade units and from 13% to 27% of the sill for the low-grade units.

The experimental variograms are considered of good quality throughout the Madsen deposit, most likely due to the tighter spaced drilling in the mineralized zones.

Table 14-8: Modeled Variogram Parameters for Gold Composites at Austin

Parameters	1 – high-grade 1			2 – high-grade 2			3 – high-grade 3		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	90°	180°	180°	95°	185°	185°	100°	190°	190°
Dip**	0°	-65°	25°	0°	-65°	25°	0°	-65°	25°
Nugget Effect C_0	0.288			0.598			0.301		
1st Structure C_1	1.113			1.202			1.735		
2nd Structure C_2	0.448			0.409			0.393		
1st Range A_1	5.7m	7.8m	4.6m	5.2m	5.7m	4.6m	6.2m	7.3m	6.2m
2nd Range A_2	40.6m	35.8m	20.7m	43.9m	31.5m	14.3m	34.7m	38.4m	18.0 m
Parameters	4 – high-grade 4			5 – high-grade 5			6 – low-grade		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	90°	180°	180°	90°	180°	180°	95°	185°	185°
Dip**	0°	-60°	30°	0°	-60°	30°	0°	-65°	25°
Nugget Effect C_0	0.366			0.366			0.451		
1st Structure C_1	0.872			0.872			1.107		
2nd Structure C_2	0.562			0.562			0.345		
1st Range A_1	6.2m	7.3m	7.3m	6.2m	7.3m	7.3m	3.0 m	4.1m	2.5m
2nd Range A_2	40.6m	28.2m	15.9m	40.6m	28.2m	15.9m	48.2m	40.1m	29.3m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

Table 14-9: Modeled Variogram Parameters for Gold Composites at South Austin

Parameters	1 – high-grade 1			2 – high-grade 2			3 – high-grade 3		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	95°	185°	185°	100°	190°	190°	95°	185°	185°
Dip**	-5°	-70°	20°	-25°	-70°	20°	0°	-60°	30°
Nugget Effect C_0	0.569			0.447			0.682		
1st Structure C_1	1.338			1.645			1.140		
2nd Structure C_2	0.661			0.389			0.733		
1st Range A_1	1.9 m	4.1 m	1.9 m	3.5 m	3.5 m	3.5 m	7.8 m	7.8 m	5.2 m
2nd Range A_2	42.2 m	40.0 m	10.5 m	46.5 m	41.1 m	14.8 m	39.0 m	26.1 m	14.8 m
Parameters	**4 – finger**			**5 – footwall 2a**			**6- low-grade**		
	Principal	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**
Azimuth*	100°	190°	190°	95°	185°	185°	95°	185°	185°
Dip**	10°	-65°	25°	-15°	-80°	10°	0°	-65°	25°
Nugget Effect C_0	0.225			0.785			0.583		
1st Structure C_1	1.422			1.480			1.124		
2nd Structure C_2	1.054			0.504			0.433		
1st Range A_1	5.7 m	13.7 m	10.0 m	2.5 m	4.1 m	2.5 m	2.5 m	2.5 m	2.5 m
2nd Range A_2	33.0 m	41.0 m	13.7 m	39.5 m	35.7 m	10.5 m	55.1 m	49.2 m	30.9 m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

Table 14-10: Modeled Variogram Parameters for Gold Composites at McVeigh

Parameters	1 – high-grade 1			2 – high-grade 2			3 – low-grade		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	95°	185°	185°	80°	170°	170°	95°	185°	185°
Dip**	0°	-80°	10°	0°	-70°	20°	0°	-65°	25°
Nugget Effect C_0	0.256			0.425			0.264		
1st Structure C_1	1.188			1.552			1.114		
2nd Structure C_2	0.758			0.444			0.575		
1st Range A_1	4.1 m	4.1 m	5.7 m	1.9 m	4.6 m	4.1 m	3.0 m	10.5 m	7.3 m
2nd Range A_2	34.7 m	38.4 m	13.2 m	28.2 m	30.9 m	14.3 m	36.3 m	50.2 m	30.4 m

Source: Ginto Consulting Inc. (2019)

Table 14-11: Modeled Variogram Parameters for Gold Composites at 8 Zone

Parameters	1 – high-grade		
	Principal	Minor	Vertical
Azimuth*	15°	105°	15°
Dip**	40°	0°	-50°
Nugget Effect C_0	0.170		
1st Structure C_1	1.038		
2nd Structure C_2	0.605		
1st Range A_1	6.2m	7.3m	4.1m
2nd Range A_2	35.2m	27.1m	11.6m

Source: Ginto Consulting Inc. (2019)

Table 14-12: Modeled Variogram Parameters for Gold Composites at A3

Parameters	1 – high-grade			2 – low-grade		
	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	100°	190°	190°	95°	185°	185°
Dip**	0°	-80°	10°	0°	-65°	25°
Nugget Effect C_0	0.602			0.583		
1st Structure C_1	1.446			1.124		
2nd Structure C_2	0.568			0.433		
1st Range A_1	7.3 m	6.8 m	9.5 m	2.5 m	2.5 m	2.5 m
2nd Range A_2	38.0 m	28.8 m	12.7 m	55.1 m	49.2 m	30.9 m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

14.1.6 Gold Grade Estimation

The estimation of gold grades into a block model was carried out with the ordinary kriging technique. The estimation strategy and parameters were tailored to account for the various geometrical, geological, and geostatistical characteristics previously identified. A separate block model of gold grade estimates was assigned to each of the zones with a total of 4 block models: Austin, South Austin and A3, McVeigh, 8 Zone. The estimate of the A3 domain was included as a part of the South Austin block model due to its spatial proximity. Each block model has the same grid definition, as presented in Table 14-13. It should be noted that the origin of the block model corresponds to the lower left corner, the point of origin being the exterior edges of the first block. A block size of 3 m (easting) x 3 m (northing) x 3 m (elevation) was selected to better reflect the geometrical configuration and anticipated underground production rate. The block model is orthogonal with no rotation applied to it.

Table 14-13: Block Grid Definition – South Austin, McVeigh, 8 Zone, A3

Coordinates	Origin (m)	Rotation (azimuth)	Distance (m)	Block Size (m)	Number of Blocks
Easting (X)	3,100.0	0°	3,504.0	3.0	1,168
Northing (Y)	1,660.0		1,500.0	3.0	500
Elevation(Z)	-100.0		1,680.0	3.0	560
Number of Blocks		327,040,000			

Source: Ginto Consulting Inc. (2019)

The database of 1.52 m capped gold grade composites was utilized as input for the grade interpolation process at Austin, South Austin, McVeigh, and A3, while for the grade estimation of the 8 Zone, the database of 0.6 m capped gold composites was utilized.

The size and orientation of the search ellipsoid for the estimation process was based on the variogram parameters modeled for gold. A minimum of two samples and maximum of 12 samples were selected for the block grade calculations. Hard boundaries were assigned in the estimation of each unit. No other restrictions, such as a minimum number of informed octants, a minimum number of holes, a maximum number of samples per hole, etc., were applied to the estimation process. A summary of the estimation parameters is presented in Table 14-14.

Table 14-14: Estimation Parameters for Gold

Estimation Parameters – Gold Grade								
Rock Code	minimum # of samples	maximum # of samples	search ellipsoid – long axis - azimuth/dip	search ellipsoid – long axis - size	search ellipsoid – short axis - azimuth/dip	search ellipsoid – short axis - size	search ellipsoid – vertical axis - azimuth/dip	search ellipsoid – vertical axis - size
Austin								
1	2	12	90°/0°	41.0m	180°/-65°	36.0m	180°/25°	21.0m
2	2	12	95°/0°	44.0m	185°/-65°	32.0m	185°/25°	14.0m
3	2	12	100°/0°	35.0m	190°/-65°	38.0m	190°/25°	18.0m
4	2	12	90°/0°	41.0m	180°/-60°	28.0m	180°/30°	16.0m
5	2	12	90°/0°	41.0m	180°/-60°	28.0m	180°/30°	16.0m
6	2	12	95°/0°	48.0m	185°/-65°	40.0m	185°/25°	29.0m
South Austin								
1	2	12	95°/-5°	42.0m	185°/-70°	40.0m	185°/20°	11.0m
2	2	12	100°/-25°	47.0m	190°/-70°	41.0m	190°/20°	15.0m
3	2	12	95°/0°	39.0m	185°/-60°	26.0m	185°/30°	15.0m
4	2	12	100°/10°	33.0m	190°/-65°	41.0m	190°/25°	14.0m
5	2	12	95°/-15°	40.0m	185°/-80°	36.0m	185°/10°	11.0m
6	2	12	95°/0°	55.0m	185°/-65°	49.0m	185°/25°	31.0m
McVeigh								
1	2	12	95°/0°	35.0m	185°/-80°	38.0m	185°/10°	13.0m
2	2	12	80°/0°	28.0m	170°/-70°	31.0m	170°/20°	14.0m
3	2	12	95°/0°	36.0m	185°/-65°	50.0m	185°/25°	30.0m
8 Zone								
1	2	12	15°/40°	35.0m	105°/0°	27.0m	15°/-50°	12.0m
A3								
1	2	12	100°/0°	38.0m	190°/-80°	29.0m	190°/10°	13.0m
2	2	12	95°/0°	55.0m	185°/-65°	49.0m	185°/25°	31.0m

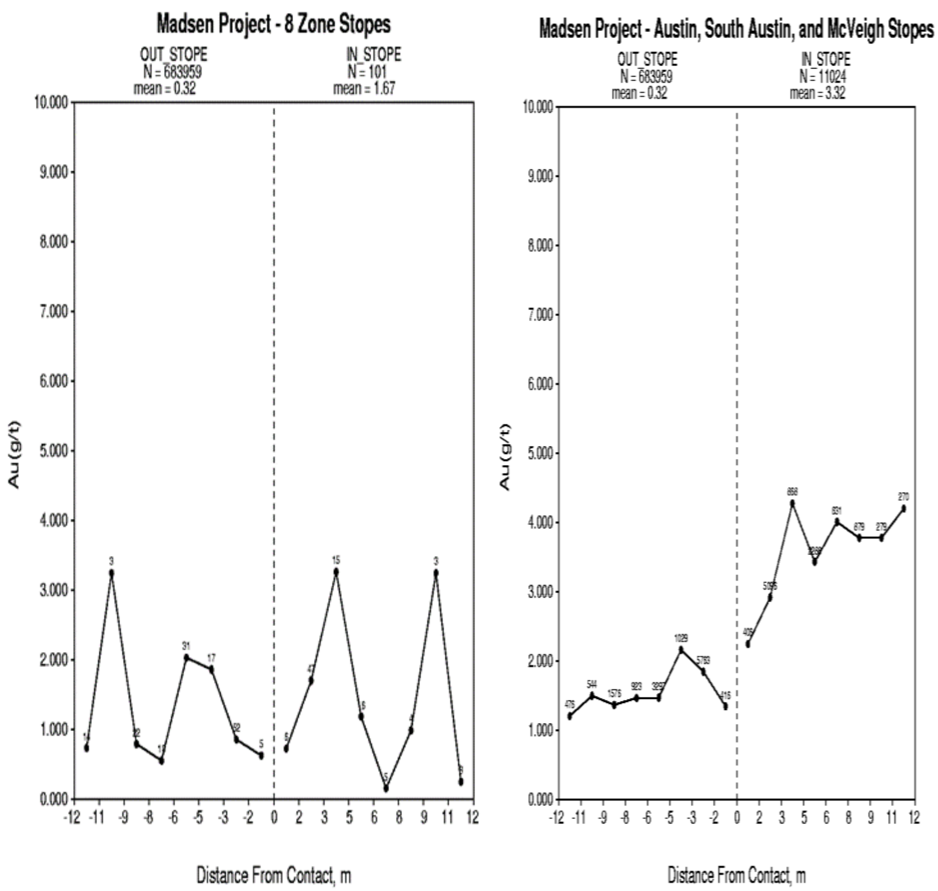
Source: Ginto Consulting Inc. (2019)

The grade estimation process consisted of a three-pass approach with the parameters of the first pass as presented in Table 14-14. The estimation parameters of the second and third passes are the same with the exception of an enlarged search ellipsoid by 1.5 times and 3 times the dimensions from the first pass, respectively. In this case, priority was given to estimates from the first pass, followed by estimates from the second pass for un-estimated blocks from the first pass, and finally the estimates of the third pass for un-estimated blocks from the first and second passes. Only blocks within the high-grade and low-grade zones were estimated.

In the planning of the grade estimation strategy in an environment where previous extensive underground mining occurred, two scenarios where investigated: estimation with grades outside mined stopes only and estimation with grades inside and outside mined stopes. To better understand the behaviour of drill hole gold grades in the vicinity of stope boundaries, contact plots were performed. In these plots, the average gold grades are compared on both side of the stope contacts in increments of distance away from the contacts. Contact plots for the Austin, South Austin, McVeigh, and 8 Zone areas are displayed in Figure 14-22.

From the plots in Figure 14-22, it can be seen that no abrupt changes in gold grades are observed on both sides of stope contacts. For Austin, South Austin, and McVeigh mineralized zones, the changes in gold grades are more transitional, while they are more similar at 8 Zone. For these reasons it was decided to proceed with a grade estimation procedure where all drill hole gold grades inside and outside stopes were utilized. Mined underground voids were then extracted from the block model.

Figure 14-22: Contact Plots of Gold Grades in the Vicinity of Mined Stopes at Madsen



Source: Ginto Consulting Inc. (2019)

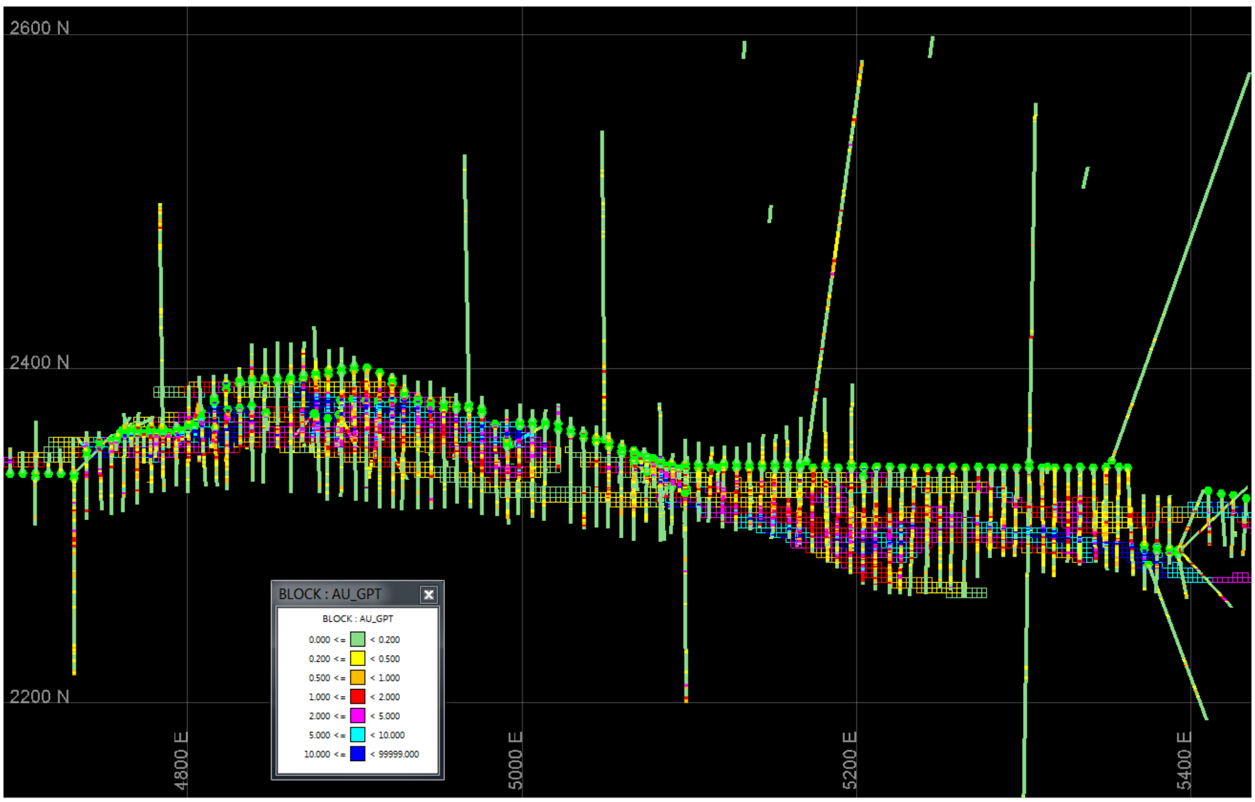
14.1.7 Validation of Grade Estimates

Validation tests were carried out on the estimates to examine the possible presence of a bias and to quantify the level of smoothing / variability.

14.1.7.1 Visual Inspection

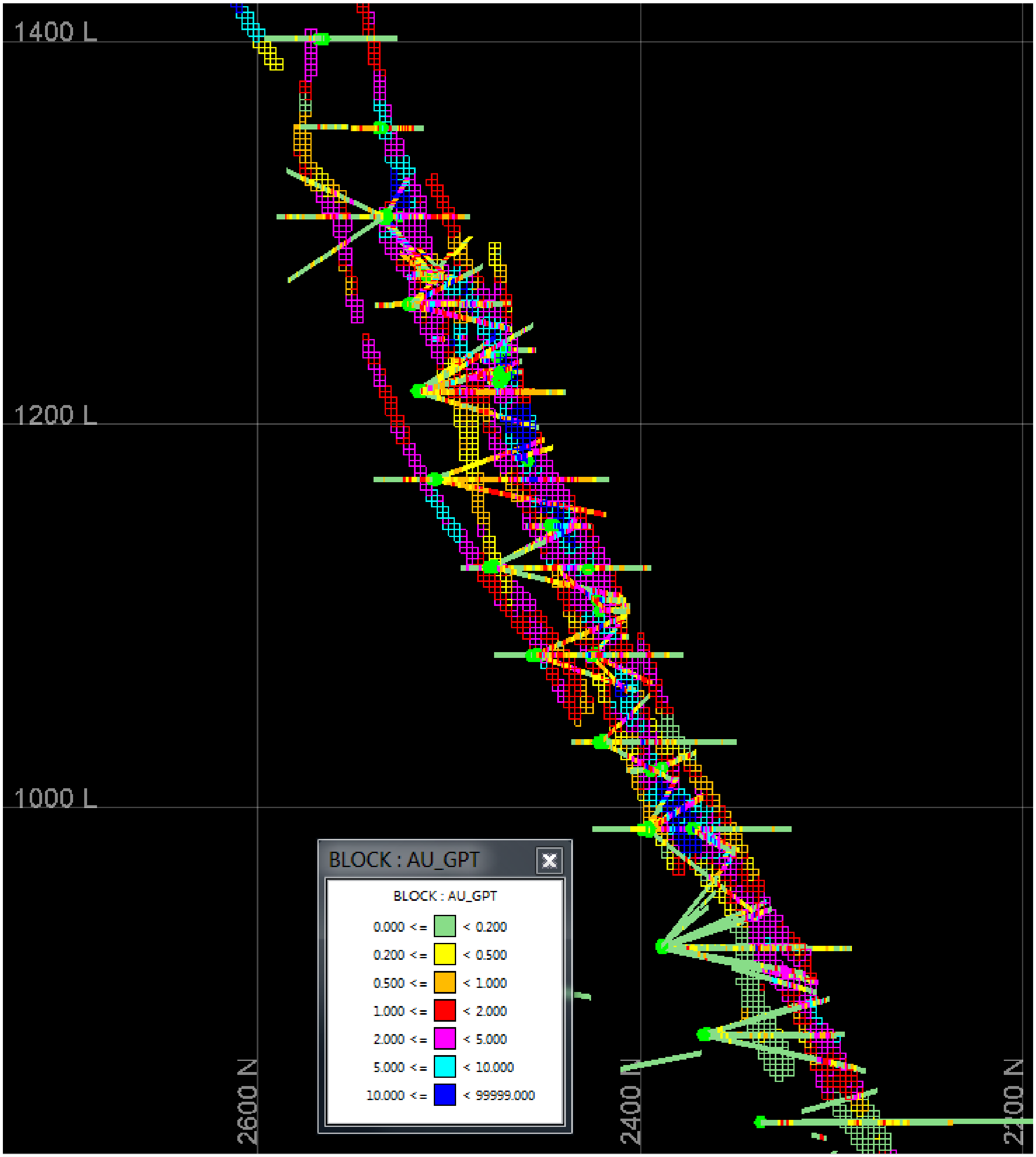
A visual inspection of the block estimates with the drill hole grades on plans, east-west and north-south cross-sections was performed as a first check of the estimates. Observations from stepping through the estimates along the different planes indicated that there was good agreement between the drill hole grades and the estimates. The orientations of the estimated grades were also as expected according to the projection angles defined by the search ellipsoid. Examples of cross-sections and level plans for gold grade estimates of the different mineralized domains are presented in Figure 14-23 to Figure 14-32.

Figure 14-23: Gold Block Grade Estimates and Drill Hole Grades at Austin – High-Grade 1 & 2 – Level 985 El



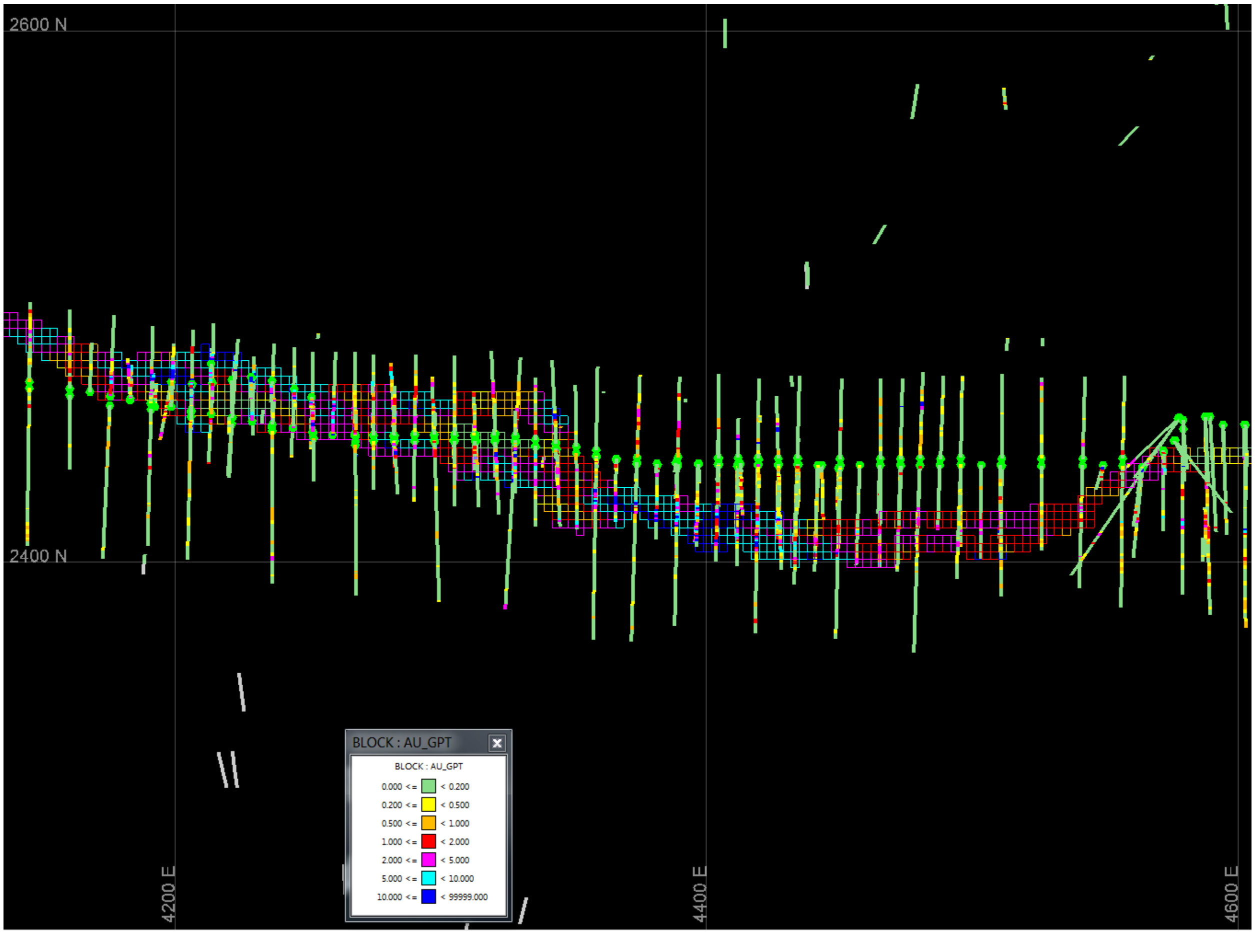
Source: Ginto Consulting Inc. (2019)

Figure 14-24: Gold Block Grade Estimates and Drill Hole Grades at Austin – High-Grade 1 – North-South Section 4880E



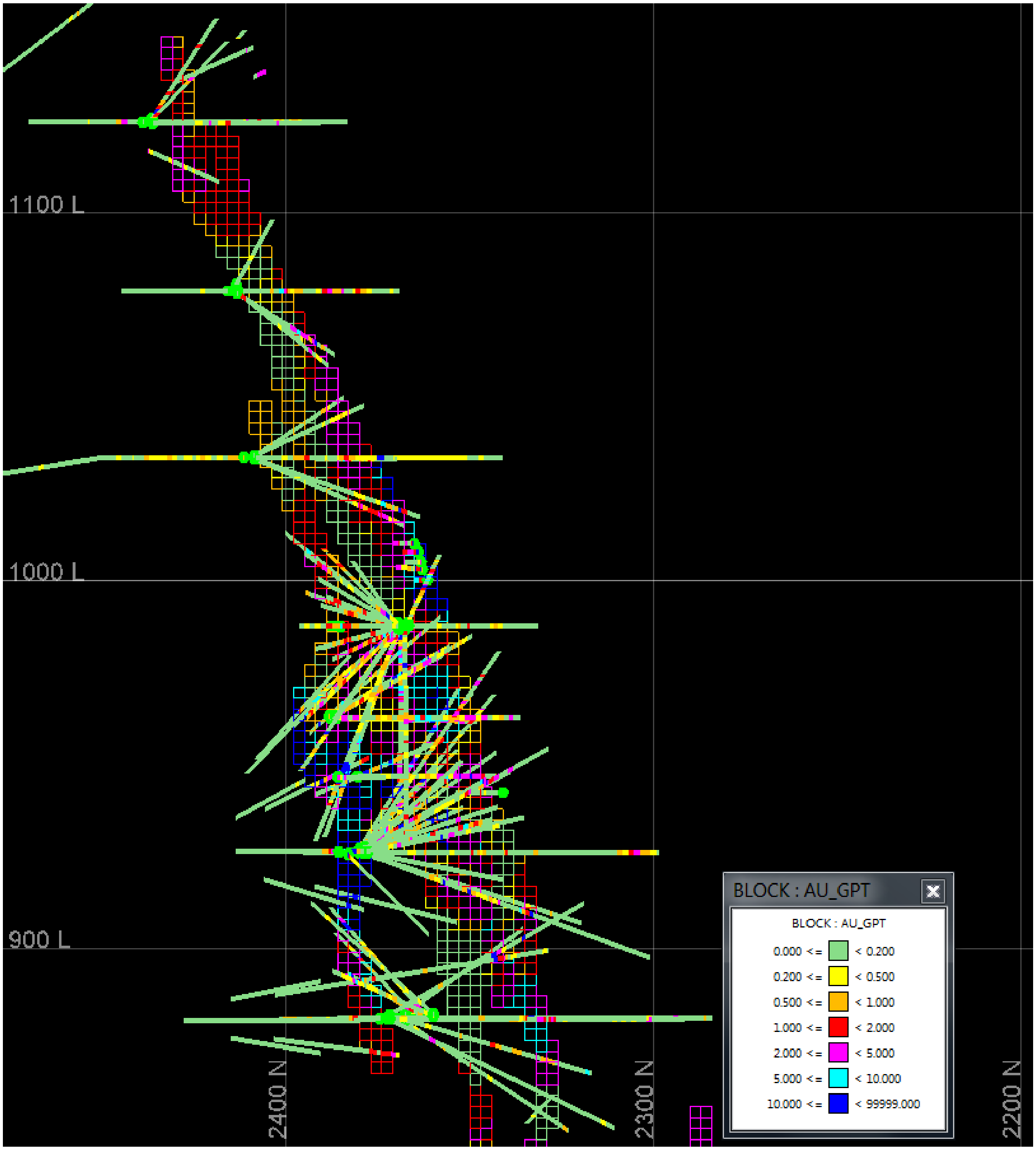
Source: Ginto Consulting Inc. (2019)

Figure 14-25: Gold Block Grade Estimates and Drill Hole Grades at South Austin – High-Grade 2 – Level 1125 El



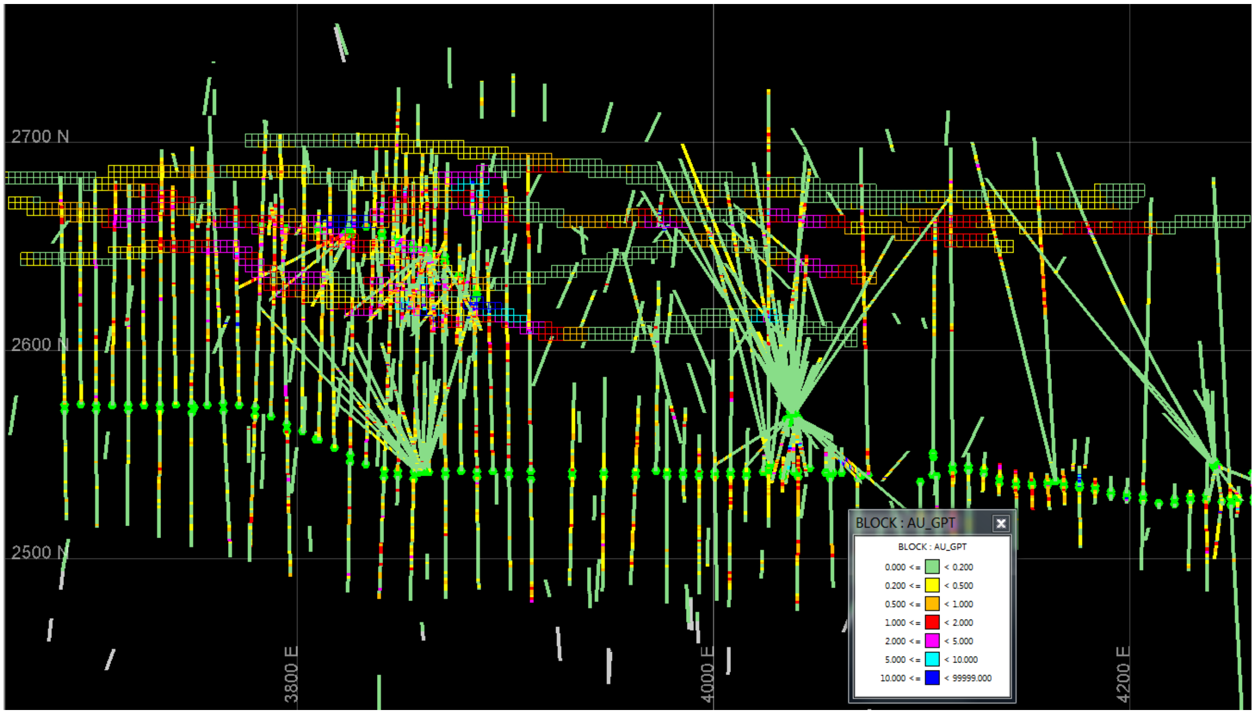
Source: Ginto Consulting Inc. (2019)

Figure 14-26: Gold Block Grade Estimates and Drill Hole Grades at South Austin – High-Grade 2 and FW2 - North-South Section 4550E



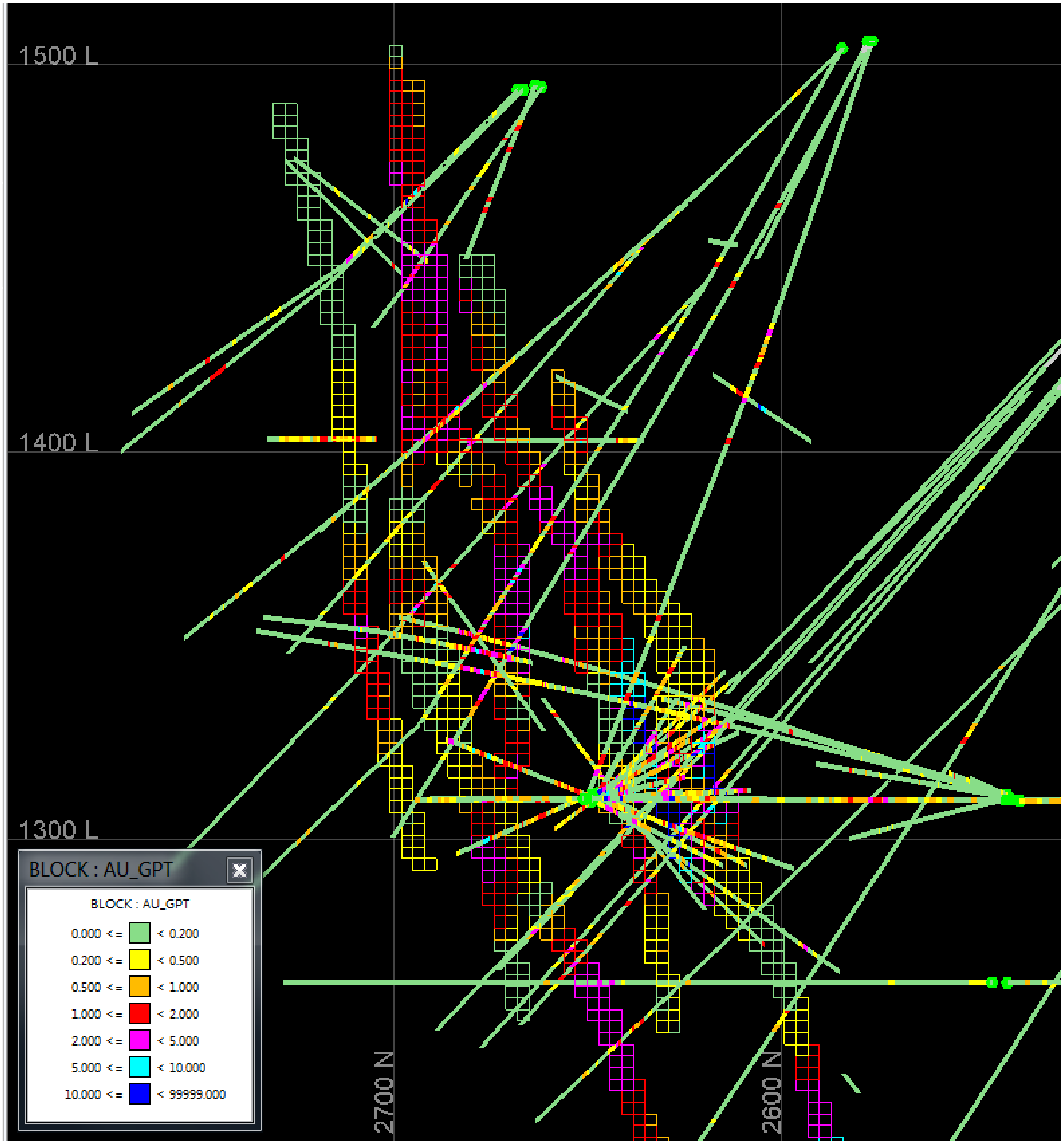
Source: Ginto Consulting Inc. (2019)

Figure 14-27: Gold Block Grade Estimates and Drill Hole Grades at McVeigh – High-Grade 1 – Level 1315 El



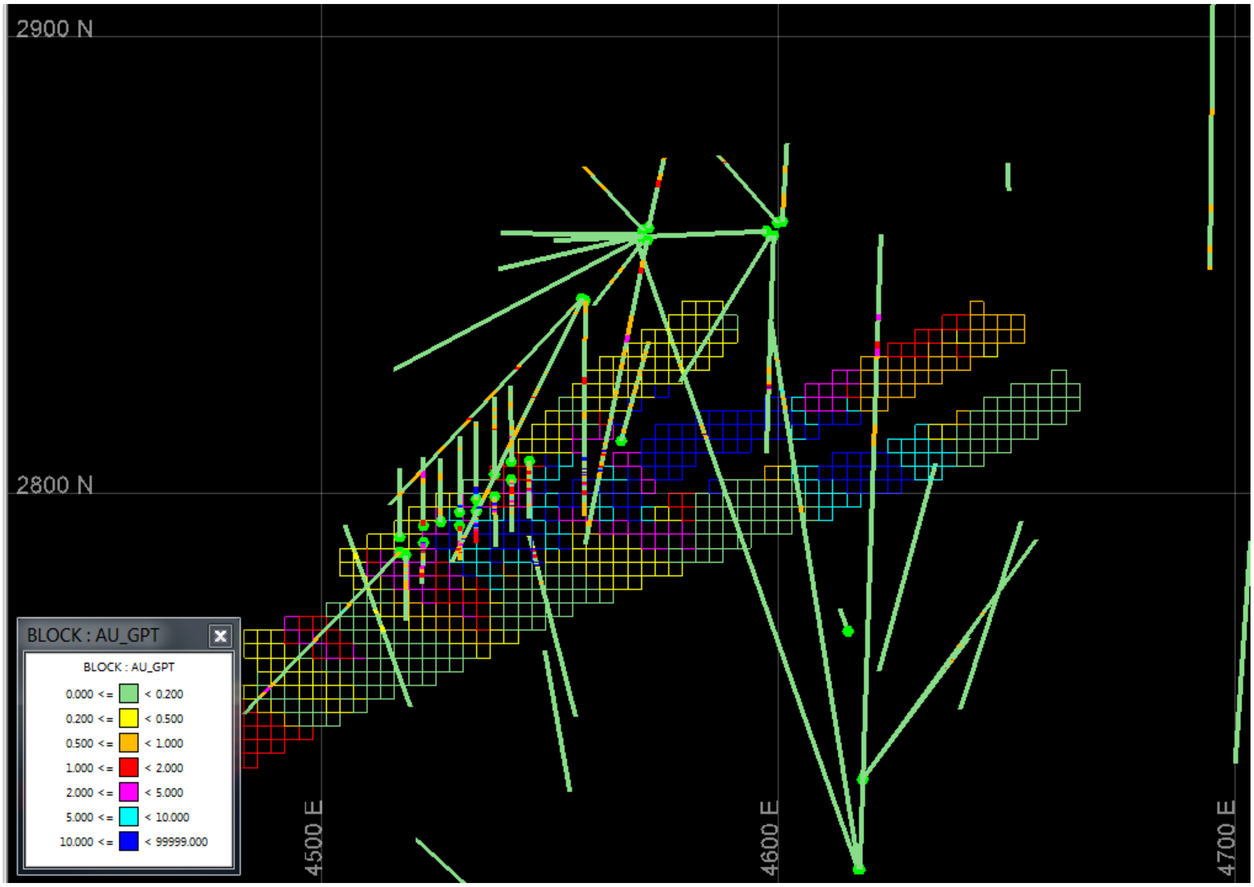
Source: Ginto Consulting Inc. (2019)

Figure 14-28: Gold Block Grade Estimates and Drill Hole Grades at McVeigh – High-Grade 1 – North-South Section 3865E



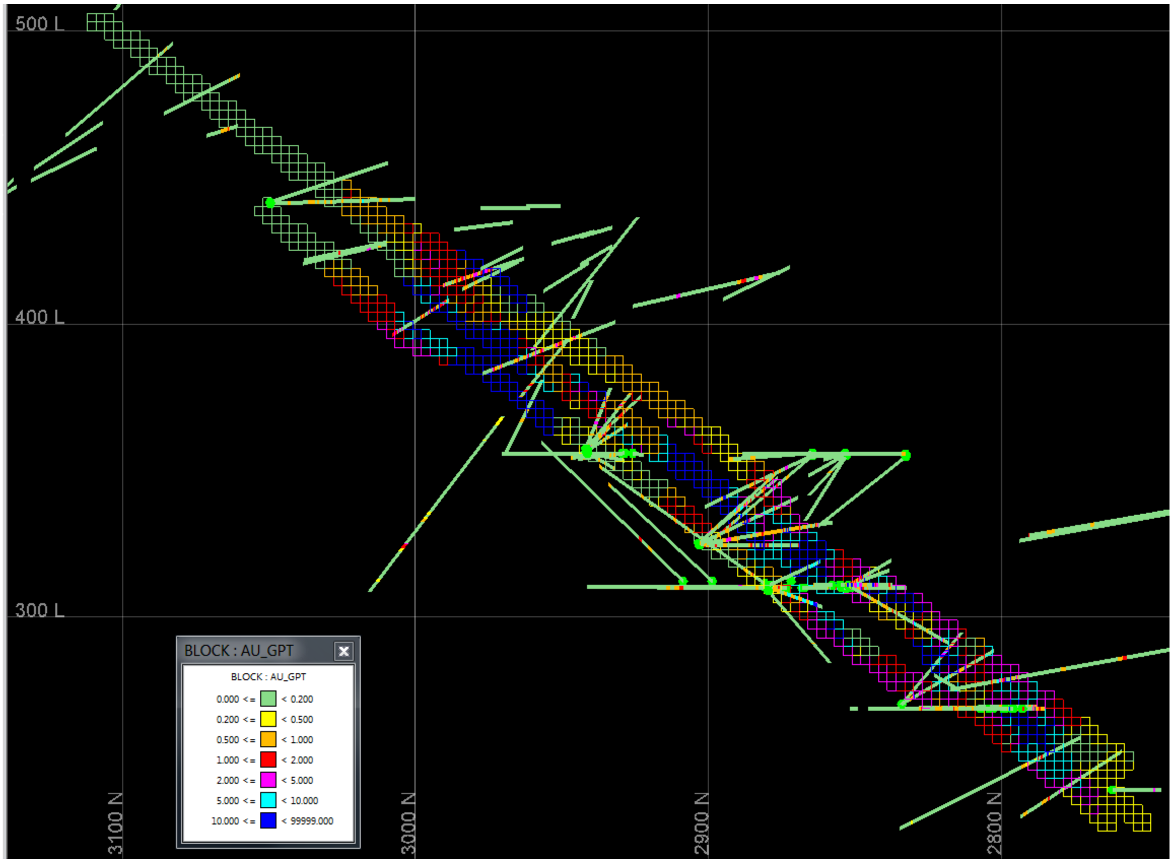
Source: Ginto Consulting Inc. (2019)

Figure 14-29: Gold Block Grade Estimates and Drill Hole Grades at 8 Zone – High-Grade – Level 260 El



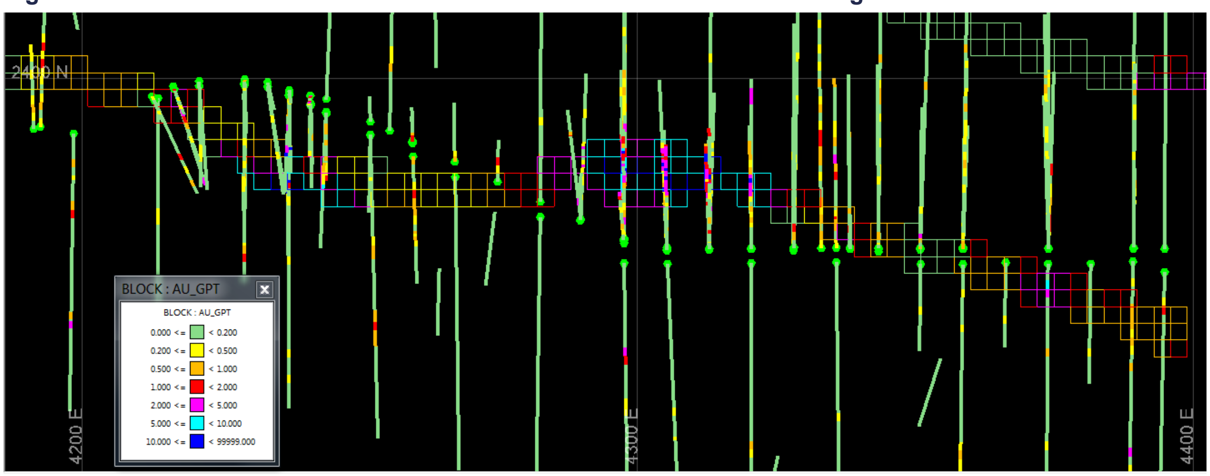
Source: Ginto Consulting Inc. (2019)

Figure 14-30: Gold Block Grade Estimates and Drill Hole Grades at 8 Zone – High-Grade – North-South Section 4540E



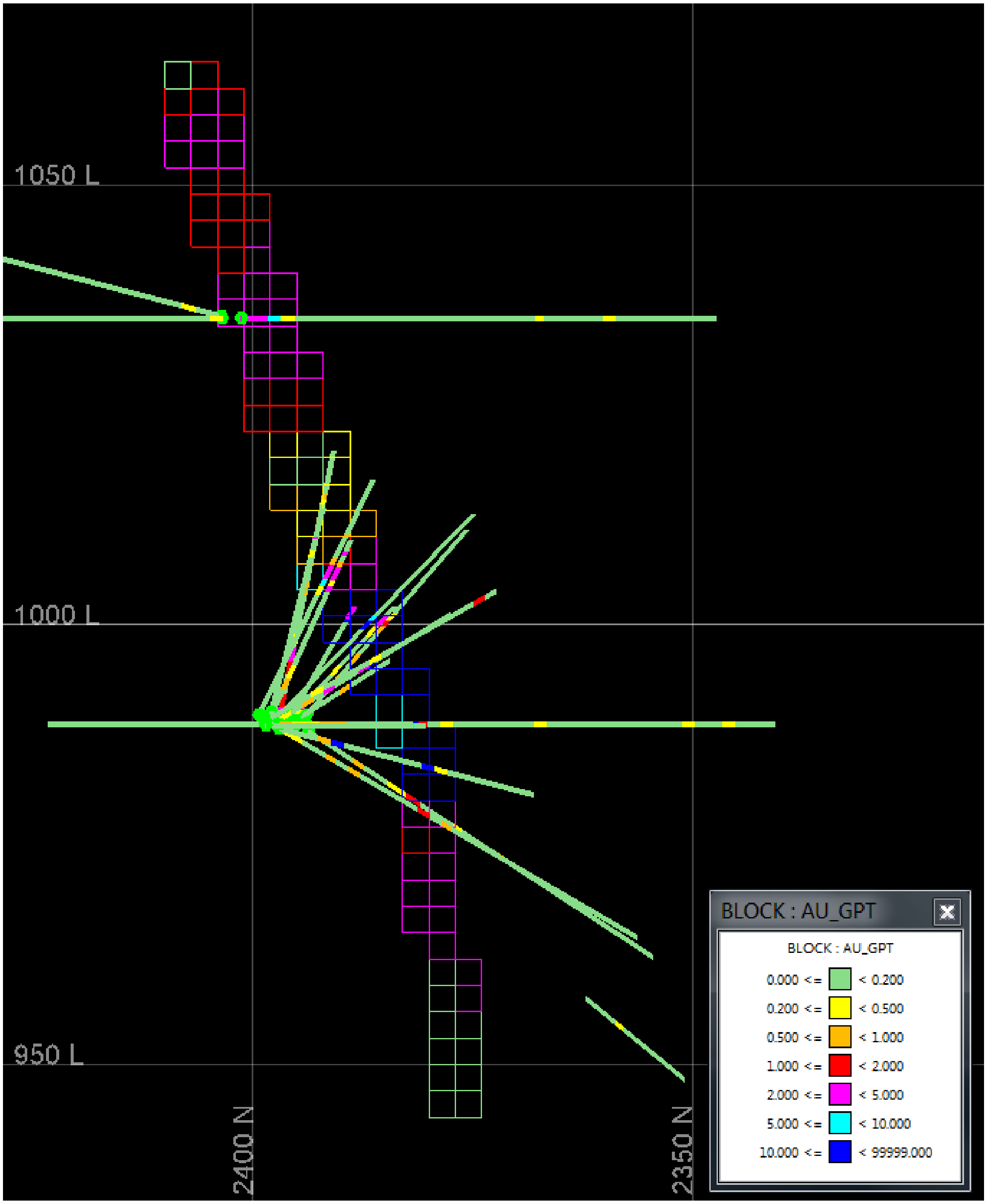
Source: Ginto Consulting Inc. (2019)

Figure 14-31: Gold Block Grade Estimates and Drill Hole Grades at A3 – High-Grade – Level 990 El



Source: Ginto Consulting Inc. (2019)

Figure 14-32: Gold Block Grade Estimates and Drill Hole Grades at A3 – High-Grade – North-South Section 4240E



Source: Ginto Consulting Inc. (2019)

14.1.7.2 Global Bias Test

The comparison of the average gold grades from the declustered composites and the estimated block grades examines the possibility of a global bias of the estimates. As a guideline, a difference between the average gold grades of more than ± 10% would indicate a significant over- or under-estimation of the block grades and the possible presence of a bias. It would be a sign of difficulties encountered in the estimation process and would require further investigation.

Results of this average gold grade comparison are presented in Table 14-15 for the different high-grade mineralized zones at Madsen.

Table 14-15: Average Gold Grade Comparison – Polygonal-declustered Composites with Block Estimates – High-grade Zones

Statistics	Declustered Composites	Block Estimates
Austin		
Average Gold Grade g/t	3.72	3.39
Difference	-8.7%	
South Austin and A3		
Average Gold Grade g/t	4.52	4.04
Difference	-10.6%	
McVeigh		
Average Gold Grade g/t	2.10	2.01
Difference	-4.3%	
8 Zone		
Average Gold Grade g/t	6.90	6.76
Difference	-2.0%	

Source: Ginto Consulting Inc. (2019)

As seen in Table 14-15, the average gold grades between the declustered composites and the block estimates are within or very close to the acceptable limits of tolerance. Therefore, it can be concluded that no significant global bias is present in the gold grade estimates.

14.1.7.3 Local Bias Test

A comparison of the grade from composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scatterplot gives a statistical valuation of the estimates. The estimated block grades should be similar to the composited grades within the block, without being exactly the same value. Thus, a high correlation coefficient will indicate satisfactory results in the interpolation process, while a medium to low correlation coefficient will be indicative of larger differences in the estimates and would require a further review of the interpolation process. Results from the pairing of composited and estimated grades within blocks pierced by a drill hole are presented in Table 14-16 for the high-grade gold zones at Madsen.

As seen in Table 14-16 for gold, the block grade estimates are very similar to the composite grades within blocks pierced by a drill hole, with high correlation coefficients, indicating satisfactory results from the estimation process.

Table 14-16: Gold Grade Comparison for Blocks Pierced by a Drill Hole – Paired Composite Grades with Block Grade Estimates – High-grade Zones

Data	Average Gold Grade g/t	Correlation Coefficient
Austin		
Composites	3.89	0.861
Block Estimates	3.66	
South Austin and A3		
Composites	5.19	0.817
Block Estimates	4.58	
McVeigh		
Composites	2.17	0.823
Block Estimates	2.10	
8 Zone		
Composites	7.21	0.815
Block Estimates	7.83	

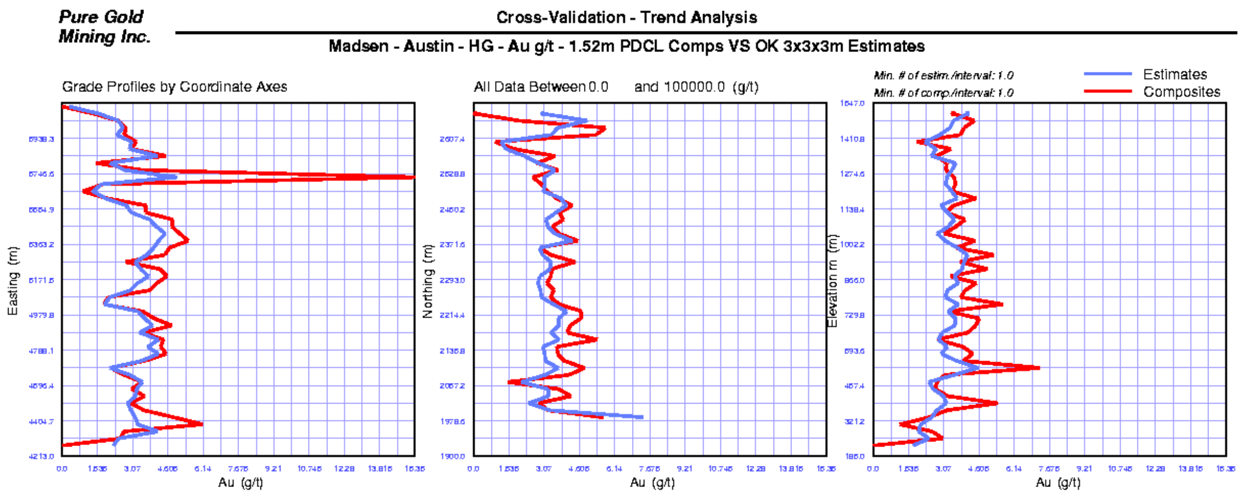
Source: Ginto Consulting Inc. (2019)

14.1.7.4 Grade Profile Reproducibility

The comparison of the grade profiles of the declustered composites with that of the estimates allows for a visual verification of an over- or under-estimation of the block estimates at the global and local scales. A qualitative assessment of the smoothing/variability of the estimates can also be observed from the plots. The output consists of three graphs displaying the average grade according to each of the coordinate axes (east, north, elevation). The ideal result is a grade profile from the estimates that follows that of the declustered composites along the three coordinate axes, in a way that the estimates have lower high-grade peaks than the composites, and higher low-grade peaks than the composites. A smoother grade profile for the estimates, from low to high grade areas, is also anticipated in order to reflect that these grades represent larger volumes than the composites.

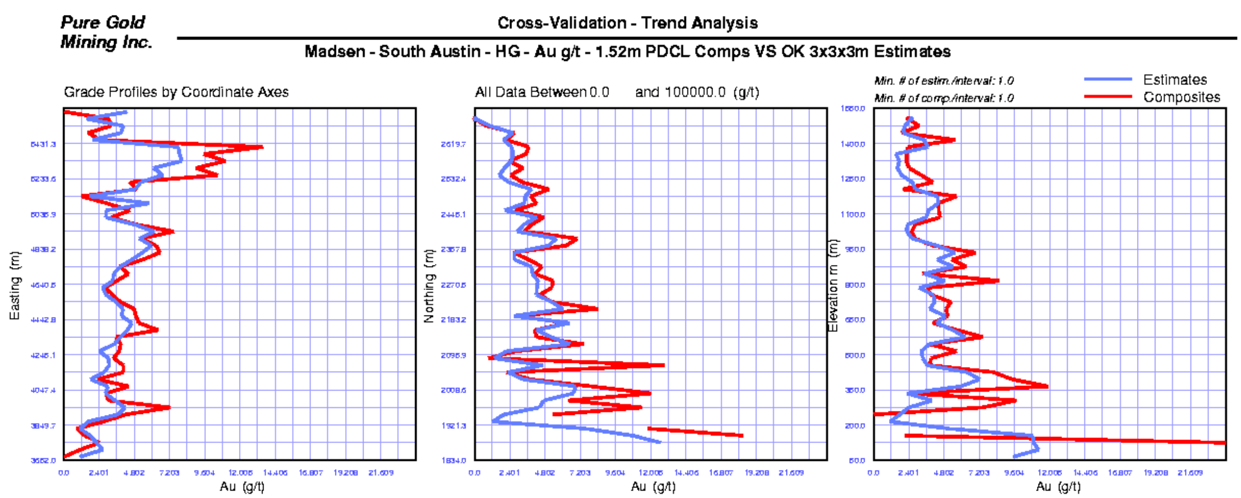
Gold grade profiles are presented in Figure 14-33 to Figure 14-36 for the high-grade zones at Madsen.

Figure 14-33: Gold Grade Profiles of Declustered Composites and Block Estimates – High-Grade Zones – Austin – Madsen Project



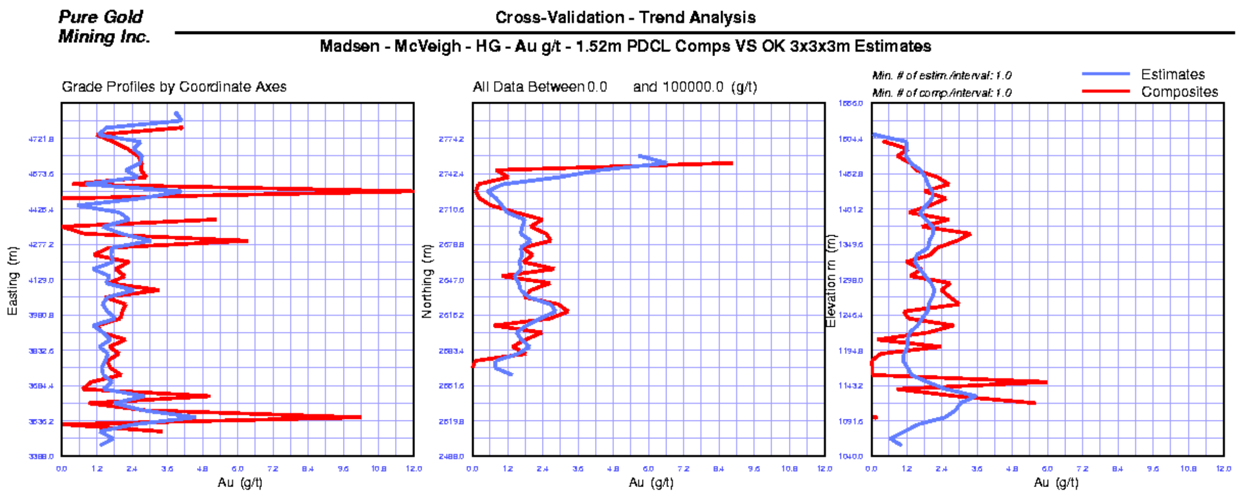
Source: Ginto Consulting Inc. (2019)

Figure 14-34: Gold Grade Profiles of Declustered Composites and Block Estimates – High-Grade Zones – South Austin and A3 – Madsen Project



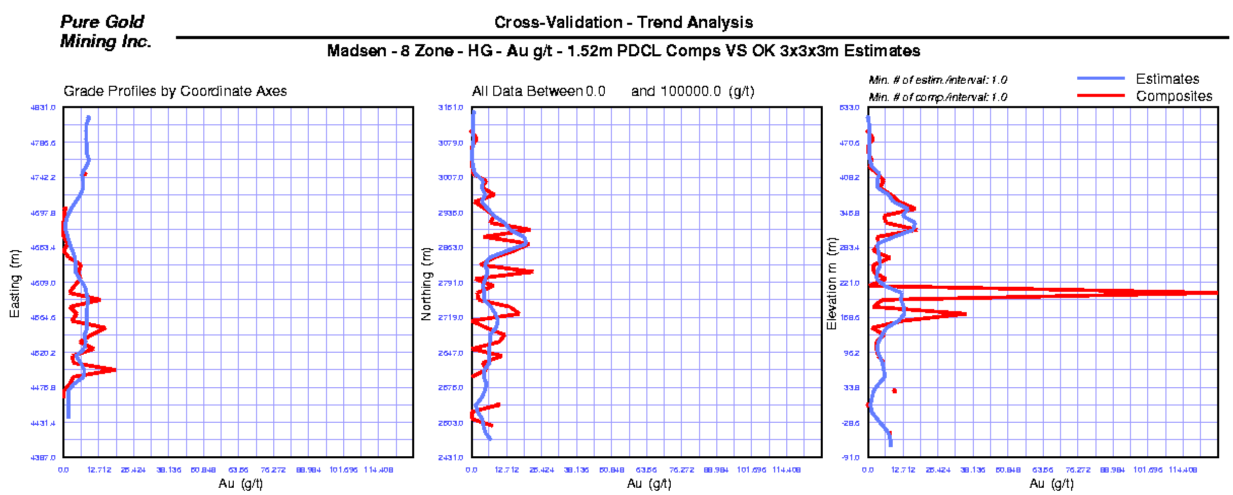
Source: Ginto Consulting Inc. (2019)

Figure 14-35: Gold Grade Profiles of Declustered Composites and Block Estimates – High-Grade Zones – McVeigh – Madsen Project



Source: Ginto Consulting Inc. (2019)

Figure 14-36: Gold Grade Profiles of Declustered Composites and Block Estimates – High-Grade Zone – 8 Zone



Source: Ginto Consulting Inc. (2019)

From the plots of Figure 14-33 to Figure 14-36, it can be seen that the grade profiles of the declustered composites are well reproduced by those of the block estimates and consequently that no global or local bias is observed. As anticipated, some smoothing of the block estimates can be seen in the profiles, where estimated grades are higher in lower grade areas and lower in higher grade areas. To quantify the level of smoothing of the estimates, further investigation is required (Section 14.1.7.5, Level of Smoothing / Variability).

14.1.7.5 Level of Smoothing / Variability

The level of smoothing / variability of the estimates can be measured by comparing a theoretical distribution of block grades with that of the actual estimates. The theoretical distribution of block grades is derived from that of the declustered composites, where a change of support algorithm is utilized for the transformation (Indirect Lognormal Correction). In this case, the variance of the composites' grade population is corrected (reduced) with the help of the variogram model, to reflect a distribution of block grades (3 m x 3 m x 3 m). The comparison of the coefficient of variation (CV) of this population with that of the actual block estimates provides a measure of smoothing. Ideally a lower CV from the estimates by 5 to 30% is targeted as a proper amount of smoothing. This smoothing of the estimates is desired as it allows for the following factors: the imperfect selection of blocks at the mining stage (misclassification), the block grades relate to much larger volumes than the volume of core (support effect), and the block grades are not perfectly known (information effect). A CV lower than 5 to 30% for the estimates would indicate a larger amount of smoothing, while a higher CV would represent a larger amount of variability. Too much smoothing would be characterized by grade estimates around the average grade, where too much variability would be represented by estimates with abrupt changes between lower and higher-grade areas.

Results of the level of smoothing / variability analysis are presented in Table 14-17 for the different high-grade zones at Madsen. As observed in this table, the CVs of the gold estimates are within or close to the targeted range, towards the higher end of the smoothing level. A possible measure to reduce this observed smoothing would be to decrease the number of samples at the grade estimation stage.

Table 14-17: Level of Smoothing / Variability of Gold Estimates

CV – Theoretical Block Grade Distribution	CV – Actual Block Grade Distribution	Difference
Austin		
1.738	1.325	-31.2%
South Austin and A3		
1.971	1.529	-28.9%
McVeigh		
1.967	1.542	-27.6%
8 Zone		
3.027	2.314	-30.8%

Source: Ginto Consulting Inc. (2019)

14.1.8 Resource Classification

The mineral resource was classified as indicated and inferred based on the variogram ranges of the second structures. The average distance of samples from the block center was utilized as the classification criterion. The classification distances for each mineralized zones are provided in Table 14-18. In addition to this resource classification strategy, all blocks located on the periphery of design stopes were made of the indicated category to allow these grade estimates to be utilized for the calculation of the mining dilution of the mineral reserves.

Table 14-18: Classification Distances

Mineralized Zone	Indicated	Inferred
Austin	≤ 30.0m	>30.0m
South Austin	≤ 30.0m	>30.0m
McVeigh	≤ 27.0m	>27.0m
8 Zone	≤ 25.0m	>25.0m

Source: Ginto Consulting Inc. (2019)

It should be noted that there are no mineral resources in the measured category, mainly due to some uncertainty associated with the historical data.

14.1.9 Editing of the Block Model

The block model of gold grade estimates was edited with the mined voids, the dykes, and the topographic surface at Madsen.

14.1.9.1 Underground Mined Voids

The underground voids from the historical mining at Madsen were provided as 3-D wireframes that were digitized by SRK for Claude Resources' 2009 resource estimate. As previously mentioned in Section 14.1.2.3, the mined voids were grouped into 5 different types: stopes, ramps, shafts, raises, and drifts. A geotechnical buffer of 5 m was added to the stope wireframes at Austin, South Austin, and McVeigh. No geotechnical buffer was developed for the stope wireframes of the 8 Zone due to the more discrete nature of the quartz vein-hosted mineralization. All wireframes of the underground mined voids were validated in Maptek™ Vulcan™.

The editing of the block model with the mined voids consisted of determining the exact fraction of the stope within each block and storing this information in a variable. This fraction variable with values ranging from 0.0 (no stopes) to 1.0 (all stope), was then utilized to affect the specific gravity in the calculation of the block tonnage: sg x (1.0 – stope fraction) x 3 m x 3 m x 3 m. The gold grade within the stope was assigned a 0.0 g/t value.

14.1.9.2 Dykes

The proportion of dyke within each block was estimated with an indicator approach, as described in Section 14.1.2.2. The dyke variable was utilized to dilute the gold grade estimates of the block model with a 0.0 g/t Au grade, since the dykes were considered barren. The dyke variable has values ranging from 0.0 (no dyke) to 1.0 (all dyke). The final gold grade was derived as follows: Au final = Au estimate x (1.0 – dyke fraction).

14.1.9.3 Topographic Surface

The percentage of block below the topographic surface was stored in a variable and utilized in the tonnage calculation of each block. The topography variable was utilized to affect the specific gravity in the calculation of the block tonnage: sg x (below topo percentage) / 100.0.

14.1.10 Mineral Resource Calculation

The mineral resource was calculated for 3 m (X) x 3 m (Y) x 3 m (Z) blocks with a constant specific gravity (SG) value of 2.94. Statistics on the specific gravity measurements from the Pure Gold drill holes with 3,288 determinations, from Claude Resources drill holes with 2,636 determinations, and from metallurgical testwork, indicated an average specific gravity of 2.94. This differs from the historical SG value of 2.84 utilized in the past.

For the mineral resource's tonnage calculation, the proportion of underground mined voids and proportion below topographic surface of each block was integrated in the tonnage computation. Only gold grade estimates outside the mined out stopes with a 15 ft geotechnical buffer and outside all other underground mined voids were kept for the reporting of the mineral resources. As well, the fraction of each block within the high-grade and low-grade zones was accounted for in the tonnage and grade calculations. For the mineral resource's gold grade calculation, the estimates were diluted to the proportion of dyke material within each block.

The indicated and inferred mineral resources for the different zones are presented in Table 14-19 to Table 14-23 and for the overall Madsen project in Table 14-24. The Madsen indicated mineral resources at a 4.0 g/t gold cut-off are 6.429 Mt at an average gold grade of 8.99 g/t, for a total of 1.857 million ounces of gold. The Madsen inferred mineral resources at a 4.0 g/t gold cut-off are 0.889 million tonnes at an average gold grade of 8.42 g/t, for a total of 0.241 million ounces of gold. The reported mineral resources are inclusive of the mineral reserves. A grade-tonnage curve of the resource is presented in Figure 14-37.

It should be noted that 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 estimated will be converted into mineral reserves. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The CIM definitions were followed for the classification of indicated and inferred mineral resources. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred mineral resources as an indicated mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated mineral resource category. All figures in Tables 14-19 to 14-24 have been rounded to reflect the relative accuracy of the estimates. Mineral resources are reported at a cut-off grade of 4.0 g/t gold based on US$1,200 per troy ounce gold and gold metallurgical recoveries of 92%.

Table 14-19: Mineral Resources* – Austin – Effective February 5, 2019

AUSTIN									
	INDICATED								
	HG1			HG2			HG3		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	1,980,000	6.15	391,000	2,053,000	6.40	422,000	1,459,000	6.70	314,000
4.0	1,263,000	7.68	312,000	1,441,000	7.64	354,000	991,000	8.23	262,000
5.0	861,000	9.19	254,000	1,026,000	8.93	295,000	713,000	9.71	223,000
	HG4			HG5			LG		
3.0	386,000	6.75	84,000	-	-	-	10,000	4.85	2,000
4.0	259,000	8.34	69,000	-	-	-	7,000	5.30	1,000
5.0	190,000	9.76	60,000	-	-	-	4,000	5.91	1,000
	TOTAL								
3.0	5,887,000	6.41	1,213,000						
4.0	3,962,000	7.84	999,000						
5.0	2,794,000	9.26	832,000						
	INFERRED								
	HG1			HG2			HG3		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	20,000	6.46	4,000	50,000	5.45	9,000	30,000	12.31	12,000
4.0	14,000	7.62	4,000	26,000	7.29	6,000	30,000	12.31	12,000
5.0	11,000	8.62	3,000	17,000	8.83	5,000	23,000	14.63	11,000
	HG4			HG5			LG		
3.0	84,000	8.21	22,000	292,000	6.53	61,000	191,000	4.59	28,000
4.0	84,000	8.21	22,000	292,000	6.53	61,000	100,000	5.58	18,000
5.0	68,000	9.09	20,000	211,000	7.31	50,000	55,000	6.51	12,000
	TOTAL								
3.0	667,000	6.36	136,000						
4.0	547,000	6.99	123,000						
5.0	386,000	8.05	100,000						

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-20: Mineral Resources* – South Austin – Effective February 5, 2019

SOUTH AUSTIN									
	INDICATED								
	HG1			HG2			HG3		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	196,000	6.25	39,000	1,331,000	7.32	313,000	133,000	8.56	37,000
4.0	137,000	7.45	33,000	955,000	8.83	271,000	104,000	9.98	33,000
5.0	102,000	8.48	28,000	711,000	10.33	236,000	86,000	11.11	31,000
	FING			FW2			LG		
3.0	157,000	9.81	49,000	61,000	7.77	15,000	3,000	3.73	0
4.0	133,000	10.93	47,000	48,000	8.87	14,000	1,000	5.73	0
5.0	115,000	11.94	44,000	38,000	10.01	12,000	0	6.96	0
	TOTAL								
3.0	1,880,000	7.51	454,000						
4.0	1,378,000	8.98	398,000						
5.0	1,053,000	10.38	351,000						
	INFERRED								
	HG1			HG2			HG3		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	5,000	5.10	1,000	16,000	5.29	3,000	1,000	9.94	0
4.0	3,000	6.27	1,000	9,000	6.69	2,000	1,000	9.94	0
5.0	2,000	6.82	1,000	6,000	8.09	1,000	1,000	9.94	0
	FING			FW2			LG		
3.0	41,000	10.74	14,000				51,000	4.19	7,000
4.0	35,000	11.90	13,000	-	-	-	18,000	5.42	3,000
5.0	31,000	12,78	13,000				7,000	7.09	2,000
	TOTAL								
3.0	114,000	6.78	25,000						
4.0	67,000	9.11	20,000						
5.0	47,000	11.02	17,000						

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-21: Mineral Resources* – McVeigh – Effective February 5, 2019

MCVEIGH						
	INDICATED					
	HG1			HG2		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	642,000	6.12	126,000	234,000	6.04	46,000
4.0	389,000	7.86	98,000	163,000	7.16	38,000
5.0	278,000	9.23	82,000	114,000	8.32	30,000
	LG			TOTAL		
3.0	-	-	-	877,000	6.10	172,000
4.0	-	-		552,000	7.65	136,000
5.0	-	-	-	392,000	8.97	113,000
	INFERRED					
	HG1			HG2		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	82,000	5.29	14,000	12,000	5.48	2,000
4.0	46,000	6.65	10,000	8,000	6.54	2,000
5.0	32,000	7.66	8,000	5,000	7.64	1,000
	LG			TOTAL		
3.0	7,000	3.62	1,000	100,000	5.20	17,000
4.0	2,000	4.50	0	56,000	6.57	12,000
5.0		-	-	37,000	7.65	9,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-22: Mineral Resources* – 8 Zone – Effective February 5, 2019

8 ZONE						
	INDICATED					
	INDICATED			INFERRED		
	HG			HG		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	500,000	19.03	306,000	231,000	11.25	83,000
4.0	458,000	20.47	301,000	202,000	12.36	80,000
5.0	414,000	22.16	295,000	175,000	13.56	76,000

Source: Ginto Consulting Inc. (2019)

Table 14-23: Mineral Resources* – A3 – Effective February 5, 2019

A3						
	INDICATED					
	HG			LG		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	94,000	8.25	25,000	-	-	-
4.0	79,000	9.17	23,000	-	-	-
5.0	68,000	9.93	22,000	-	-	-
	TOTAL					
3.0	94,000	8.25	25,000			
4.0	79,000	9.17	23,000			
5.0	68,000	9.93	22,000			
	INFERRED					
	HG			LG		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	19,000	10.34	6,000	-	-	-
4.0	18,000	10.97	6,000	-	-	-
5.0	16,000	11.52	6,000	-	-	-
	TOTAL					
3.0	19,000	10.34	6,000			
4.0	18,000	10.97	6,000			
5.0	16,000	11.52	6,000			

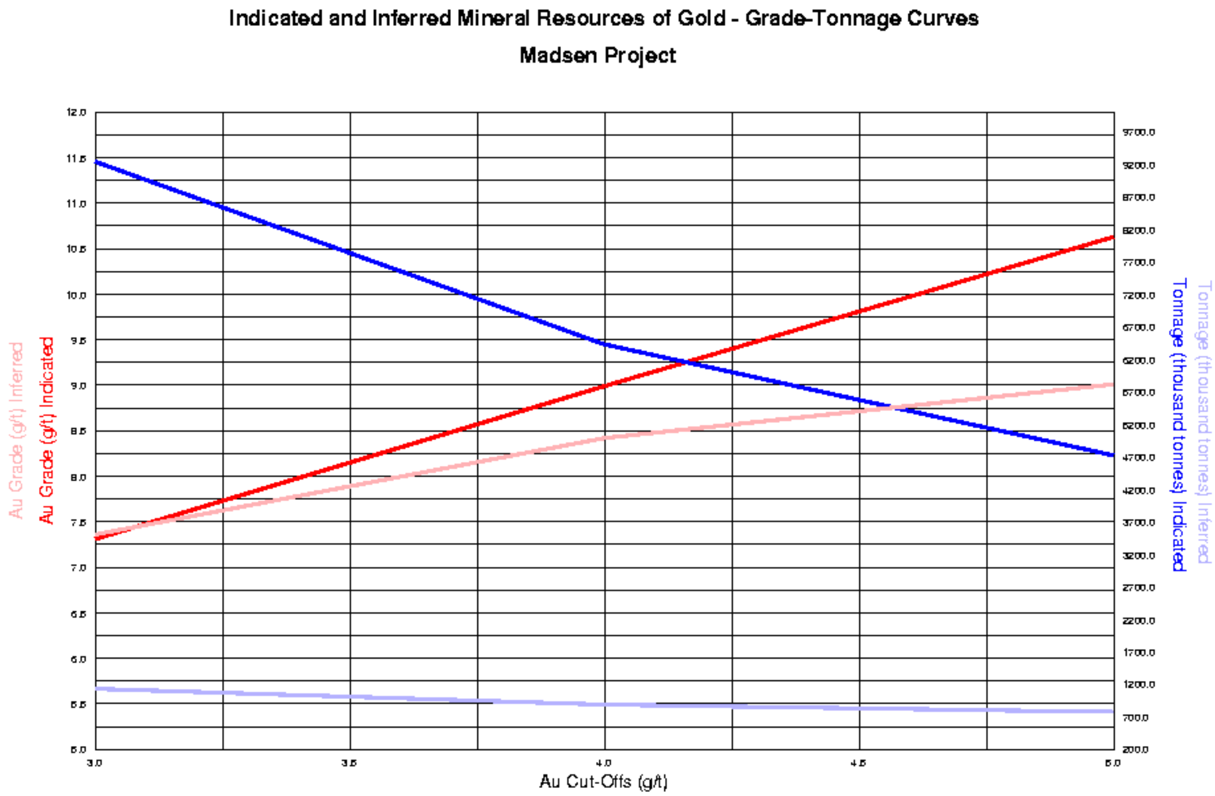
*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-24: Mineral Resources* – Effective February 5, 2019

MADSEN PROJECT						
	INDICATED					
	AUSTIN			SOUTH AUSTIN		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	5,887,000	6.41	1,213,000	1,880,000	7.51	454,000
4.0	3,962,000	7.84	999,000	1,378,000	8.98	398,000
5.0	2,794,000	9.26	832,000	1,053,000	10.38	351,000
	MCVEIGH			8 ZONE		
3.0	877,000	6.10	172,000	500,000	19.03	306,000
4.0	552,000	7.65	136,000	458,000	20.47	301,000
5.0	392,000	8.97	113,000	414,000	22.16	295,000
	A3			TOTAL		
3.0	94,000	8.25	25,000	9,239,000	7.31	2,170,000
4.0	79,000	9.17	23,000	6,429,000	8.99	1,857,000
5.0	68,000	9.93	22,000	4,721,000	10.63	1,613,000
	INFERRED					
	AUSTIN			SOUTH AUSTIN		
Au Cut-Off	Tonnage	Au Grade	Au Content	Tonnage	Au Grade	Au Content
g/t	tonnes	g/t	ounces	tonnes	g/t	ounces
3.0	667,000	6.36	136,000	114,000	6.78	25,000
4.0	547,000	6.99	123,000	67,000	9.11	20,000
5.0	386,000	8.05	100,000	47,000	11.02	17,000
	MCVEIGH			8 ZONE		
3.0	100,000	5.20	17,000	231,000	11.25	83,000
4.0	56,000	6.57	12,000	202,000	12.36	80,000
5.0	37,000	7.65	9,000	175,000	13.56	76,000
	A3			TOTAL		
3.0	19,000	10.34	6,000	1,132,000	7.36	268,000
4.0	18,000	10.97	6,000	889,000	8.42	241,000
5.0	16,000	11.52	6,000	662,000	9.78	208,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Figure 14-37: Gold Grade-Tonnage Curves of the Indicated and Inferred Mineral Resources Madsen Project



Source: Ginto Consulting Inc. (2019)

14.1.11 Comparison with the 2017 Mineral Resources

The current estimation of the mineral resources at the Madsen Gold Project was compared to the August 2017 estimate in Table 14-25.

Table 14-25: Comparison of Mineral Resources from 2017 and 2019

Au Cut-Off g/t	Mineral Resources	Indicated			Inferred		
		Tonnage (tonnes)	Avg Au Grade (g/t)	Au Content (ounces)	Tonnage (tonnes)	Avg Au Grade (g/t)	Au Content (ounces)
3.0	August 2017	8,374,000	7.19	1,936,000	1,017,000	6.90	225,000
	February 2019	9,239,000	7.31	2,170,000	1,132,000	7.36	268,000
	Difference	10.3%	1.7%	12.1%	11.3%	6.7%	19.1%
4.0	August 2017	5,785,000	8.86	1,648,000	587,000	9.42	178,000
	February 2019	6,429,000	8.99	1,857,000	889,000	8.42	241,000
	Difference	11.1%	1.5%	12.7%	51.4%	-10.6%	35.4%
5.0	August 2017	4,218,000	10.50	1,423,000	406,000	11.62	152,000
	February 2019	4,721,000	10.63	1,613,000	662,000	9.78	208,000
	Difference	11.9%	1.2%	13.4%	63.0%	-15.8%	36.8%

Source: Ginto Consulting Inc. (2019)

From Table 14-25, an increase of the tonnage, average gold grade, and metal content of the indicated mineral resources is observed for the current estimate. For the inferred mineral resources, an increase in tonnage and metal content, with a decrease in average gold grade is noted for the current estimate. These changes can be explained by a few factors such as the increase in the volumes of the high-grade and low-grade wireframes, the addition of the HG5 zone at Austin, the addition of estimates peripheral to the design stopes in the indicated resources, the increase of the overall specific gravity from 2.84 to 2.94, and an enhanced delineation of the high-grade zones from the additional holes drilled by Pure Gold.

14.1.12 Mineral Resources in Stopes

The mineral resources within the original stopes and within the stopes with a 15 ft geotechnical buffer were computed for comparison with the historical production from 1938 to 1999. The resources inside the other underground excavations were also added to that of the stopes. All gold grades within the underground voids are reported at a 0.0 g/t Au cut-off.

Comparisons show that the tonnages within the underground voids from the current mineral resource estimate and the historical production are very similar. However, the gold grades differ, with the past production results showing a higher average grade and metal content.

The mineral resources within the stopes with a 15 ft geotechnical buffer were calculated to quantify the impact of utilizing the buffered stopes for the final statement of the mineral resources. It is uncertain at this time if a portion or any of this material could potentially be added to the current mineral resources.

14.1.13 Mineral Resource Sensitivities

In addition to the base case scenario of the mineral resource presented above, two other cases were examined. A first scenario looked at the estimation of the mineral resource from samples located outside stopes only, while a second scenario looked at hard boundary conditions between veins from each domain. Results from each scenario are compared to the base case scenario in Tables 14-26 and 14-27.

Table 14-26: Comparison of Mineral Resources* with Samples Outside Stopes Only – 4.0 g/t Au Cut-Off

Mineral Resources	Indicated			Inferred		
	Tonnage tonnes	Avg Au Grade (g/t)	Au Content (ounces)	Tonnage tonnes	Avg Au Grade (g/t)	Au Content (ounces)
Base Case	6,128,000	8.90	1,754,000	659,000	10.10	214,000
Samples Outside Stopes	5,741,000	8.68	1,602,000	620,000	10.46	208,000
Difference	-6.3%	-2.5%	-8.7%	-5.9%	3.6%	-2.8%

Note: *mineral resources exclusive of HG5 zone at Austin and re-classified to indicated estimates in periphery of design stopes – both resources calculated with an SG of 2.84

Source: Ginto Consulting Inc. (2019)

Table 14-27: Comparison of Mineral Resources* with Vein Hard Boundaries – 4.0 g/t Au Cut-Off

Mineral Resources	Indicated			Inferred		
	Tonnage (tonnes)	Avg Au Grade (g/t)	Au Content (ounces)	Tonnage (tonnes)	Avg Au Grade (g/t)	Au Content (ounces)
Base Case	6,128,000	8.90	1,754,000	659,000	10.10	214,000
Vein Hard Boundaries	5,293,000	8.82	1,501,000	483,000	10.20	158,000
Difference	-13.6%	-0.9%	-14.4%	-26.7%	1.0%	-26.2%

Note: *mineral resources exclusive of HG5 zone at Austin and re-classified to indicated estimates in periphery of design stopes – both resources calculated with an SG of 2.84

Source: Ginto Consulting Inc. (2019)

As seen in Table 14-26, the exclusion of samples within mined out stopes does not have a significant impact on the mineral resources and especially on the average gold grade, where most of the impact would be expected. The minor effect seen in this case might be explained by the fact that estimates within a 15ft buffer around mined out stopes were removed in the base case scenario.

The usage of hard boundaries between the high-grade veins within each domain, as seen in Table 14-27, shows less tonnage estimated than the base case scenario at an elevated cut-off gold grade. It is however believed that soft boundary conditions between the high-grade veins internal to each high-grade domain is more reflective of the mineral resources found at Madsen, based on the current understanding.

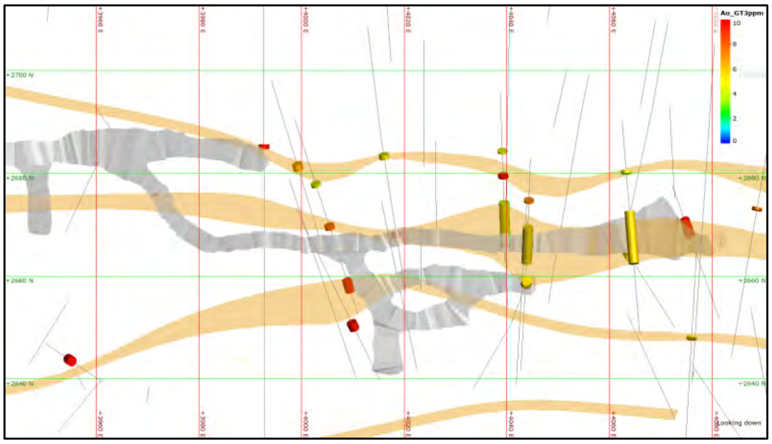
14.1.14 Mineral Resource Reconciliation to the Bulk Sample

14.1.14.1 Drill Data and Resource Modeling

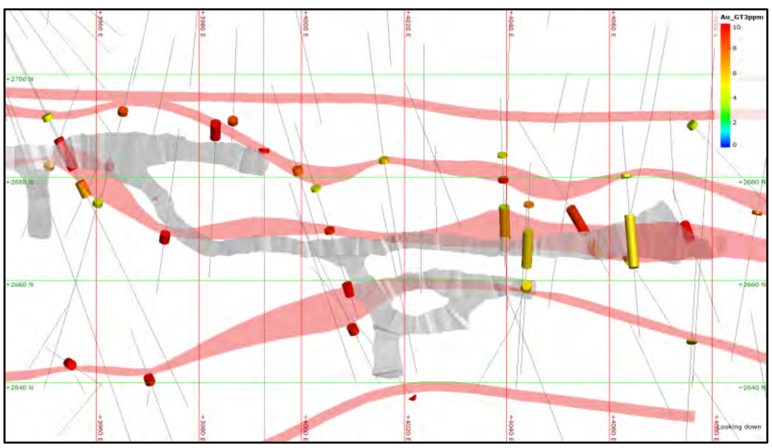
The 2018 underground bulk sample was collected in part to test and validate the mineral resource estimate. Mining targeted two proposed stopes defined in the 2017 PEA that were close to, and easily accessible from, the bottom of the existing McVeigh ramp. Despite the significant amount of drill support in these areas (Figure 14-38a), a number of gaps remained, which were infilled by underground drilling from the McVeigh ramp and newly developed exploration drift in the hanging wall of the zone. The resource wireframes were updated in January 2018 with the new drill holes resulting in some minor modifications to the shapes in this area and better definition of the continuity of the zones (Figure 14-38b). These wireframes were used for the current resource estimate described in this technical report.

Underground drilling continued through the spring of 2018 in advance of commencing the bulk sample in June, adding further definition to the zones (Figure 14-38c), and allowing more precise delineation of the mineable resources. For example, local deviation from the resource wireframes were recognized as the mapping, chip sampling and bazooka drilling progressed, allowing for real-time refinements to the mine plan to keep pace with development. Round-by-round bazooka drilling also led to the discovery of an additional 1,575 tonnes at 8.7 g/t Au in the hanging wall zone, which previously had insufficient drilling to allow estimation of any blocks above the cut-off grade of 4 g/t Au in the resource.

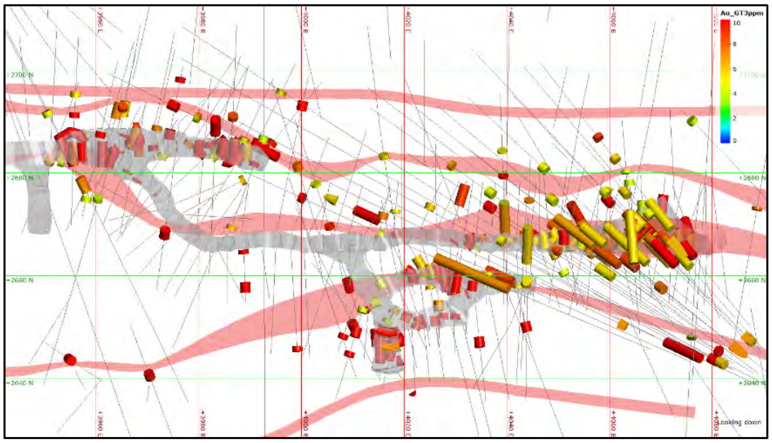
Figure 14-38: 2018 Bulk Sample



(a) Bulk sample asbuilt (grey) shown against 2017 PEA resource model wireframes (yellow) and drill hole data existing at that time.
Source: Pure Gold (2019)



(b) Updated resource wireframes (red) modified using new underground drilling available in January 2018.
Source: Pure Gold (2019)



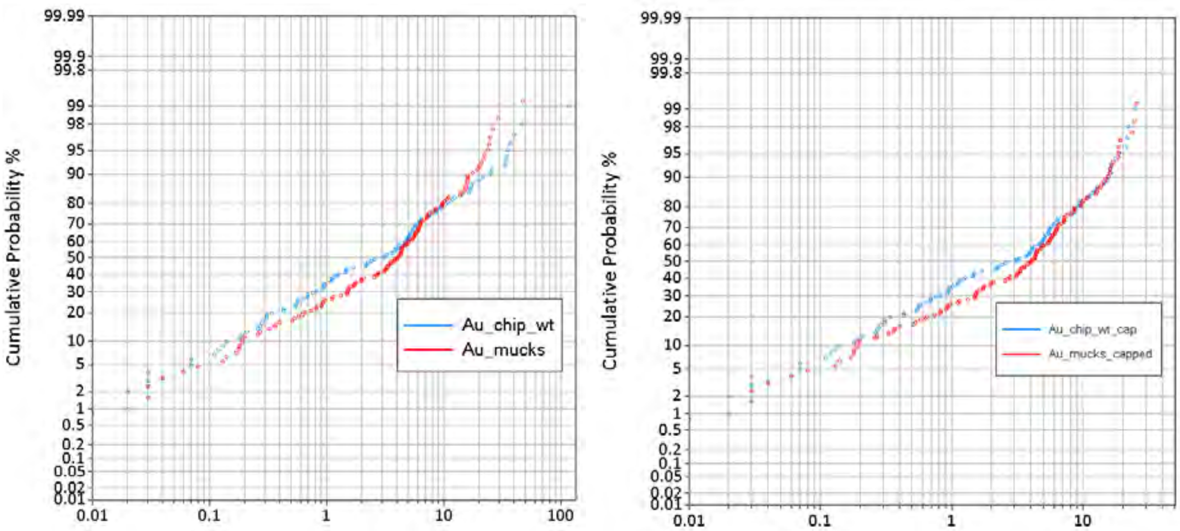
(c) All drill holes and sampling to date, including new underground drilling, chip sampling and bazooka holes
Source: Pure Gold (2019)

14.1.14.2 Bulk Sample Grade Estimation

Muck and chip samples were averaged on a round–by-round basis. Typically, around 25 muck samples were collected per round, with rounds averaging about 75 tonnes. Muck samples from each round were averaged to determine the grade of that round. Chip samples were collected along two sample lines across each face, with around 20 samples taken per face – the average width of 70 mining faces is 5.3 metres. The chip sample average grades for each round were length-weight; whereas, the muck average grades are arithmetic averages. All samples were capped at 100 g/t Au.

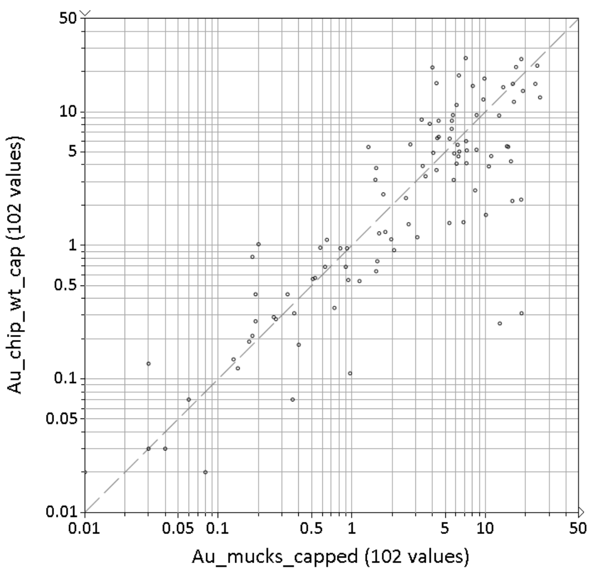
Chip samples generally show a higher degree of variability than the muck samples, with higher high grades and lower low grades; however, after capping, the grades of both sample types above a 4 g/t Au are very similar (Figure 14-39). Also, while there is some scatter seen between them on a scatter plot, the even distribution of muck versus chip sample grades about the 1:1 line indicates a lack of any significant bias (Figure 14-40).

Figure 14-39: Cumulative Probability Plots Comparing Muck versus Chip Sample Grades (Uncapped/Capped)



Source: Pure Gold (2019)

Figure 14-40: Scatterplot Showing Muck versus Chip Sample Grades on a Round by Round Basis



Source: Pure Gold (2019)

Bulk sample material was classified on the basis of the muck sample results into 'ore' (>4 g/t Au), 'low grade' (1-4 g/t), and 'un-mineralized' (<1 g/t Au). Figure 14-41 shows each round coloured by material type, with the high grade mineralization clustering into three distinct zones: West, East and Hanging Wall zones. The West and East zones correspond to >4 g/t Au areas in the resource block model, albeit at slightly modified locations as shown in Figure 14-42. As seen from the buildup of drill information in comparison to the resource modeling, this minor local variation is a product of more widely spaced drilling in these areas.

Precise location of the high grade material was enabled by a strong grade control program, guided by geological and structural geology mapping, chip sampling, and bazooka drilling.

In order to extract a meaningful portion of the resource block model for comparison to the tonnes and grade extracted for the bulk sample, a thick sub-horizontal slice, with the same width and dimensions of the as built, was cut through the block model in the test mining area. The tonnes and grade of this material above a cut-off grade of 4 g/t Au are reported in Table 14-28. Note that the resource model doesn't carry any mineable material in the Hanging Wall zone, but the totals of the East and West zones are very close, with the actual grades 15% higher than those predicted by the resource model. The Hanging Wall zone is an entirely unpredicted addition to the high grade material, adding more than 45% more tonnes at an 8% premium to the average grade predicted by the resource model in this area.

The same slice was cut through the design stopes for the feasibility reserves. The tonnes and grade from these stopes is reported at a zero cut-off grade in Table 14-29, since it is anticipated that the entire stope will be mined as ore. As expected, the design stope predicts higher tonnes and lower grade than the resource, as well as, the actual mined material.

Table 14-28: Comparison of Resource Model Predictions and Actuals from Mining

	Predicted from Resource			Actual (Capped Grades)*		
Stockpile	**Tonnes**	**Au (g/t)**	**Ounces**	**Tonnes**	**Au (g/t)**	**Ounces**
West Stoping Area	2,003	12.1	781	1,861	14.4	861
East Stoping Area	1,400	5.8	262	1,517	6.7	327
SubTotal	**3,403**	**9.5**	**1,043**	**3,378**	**10.9**	**1,188**
Variance From Resource				*-1%*	*15%*	*14%*
Hanging Wall	--	--	--	1,575	8.7	440
Total	**3,403**	**9.5**	**1,043**	**4,953**	**10.2**	**1,627**
Variance from Resource				*46%*	*8%*	*56%*

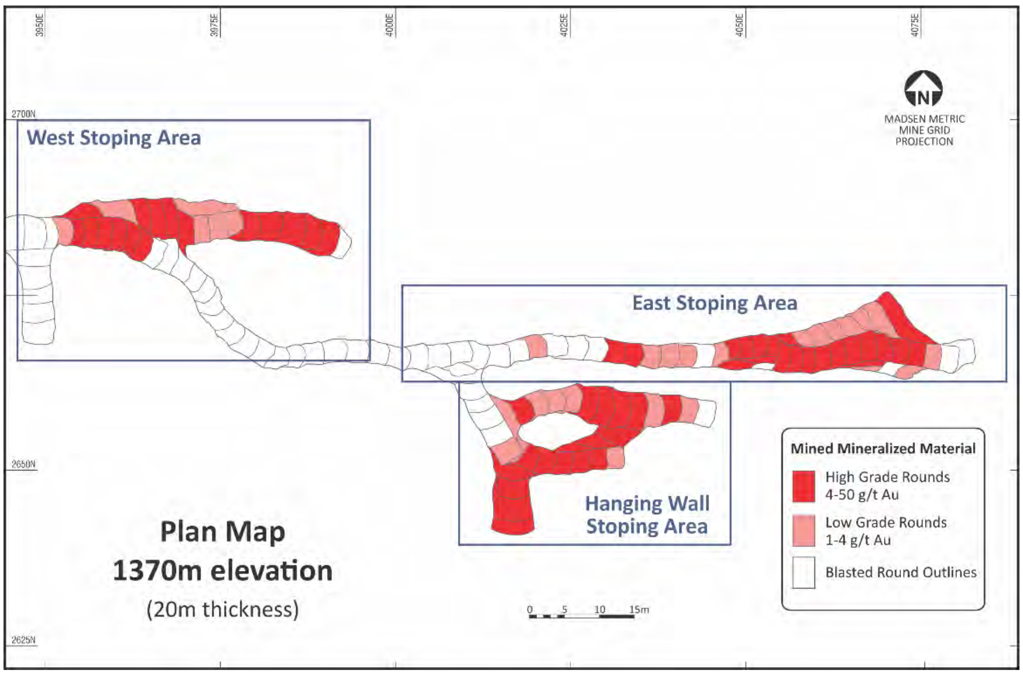
*Individual assay results were capped to 100 g/t gold
Source: Pure Gold (2019)

Table 14-29: Comparison of Resource Model Predictions and Actuals from Mining

	Stope Design			Actual (Capped Grades)*		
Stockpile	**Tonnes**	**Au (g/t)**	**Ounces**	**Tonnes**	**Au (g/t)**	**Ounces**
West Stoping Area	2,330	11.7	801	1,861	14.4	861
East Stoping Area	1,325	6.4	272	1,517	6.7	327
Subtotal	**3,655**	**9.1**	**1,073**	**3,378**	**10.9**	**1,188**
Variance From Resource				*-8%*	*20%*	*11%*
Hanging Wall	--	--	--	1,575	8.7	440
Total	**3,655**	**9.1**	**1,073**	**4,953**	**10.2**	**1,627**
Variance from Resource				*36%*	*12%*	*52%*

*Individual assay results were capped to 100 g/t gold
Source: Pure Gold (2019)

Figure 14-41: Plan Map of Bulk Sample Drift Showing Individual Rounds Coloured by Grade Category



Source: Pure Gold (2019)

Table 14-30 tabulates the total mineralized material mined for the bulk sample, including material currently stored in a ‘Low Grade’ stockpile.

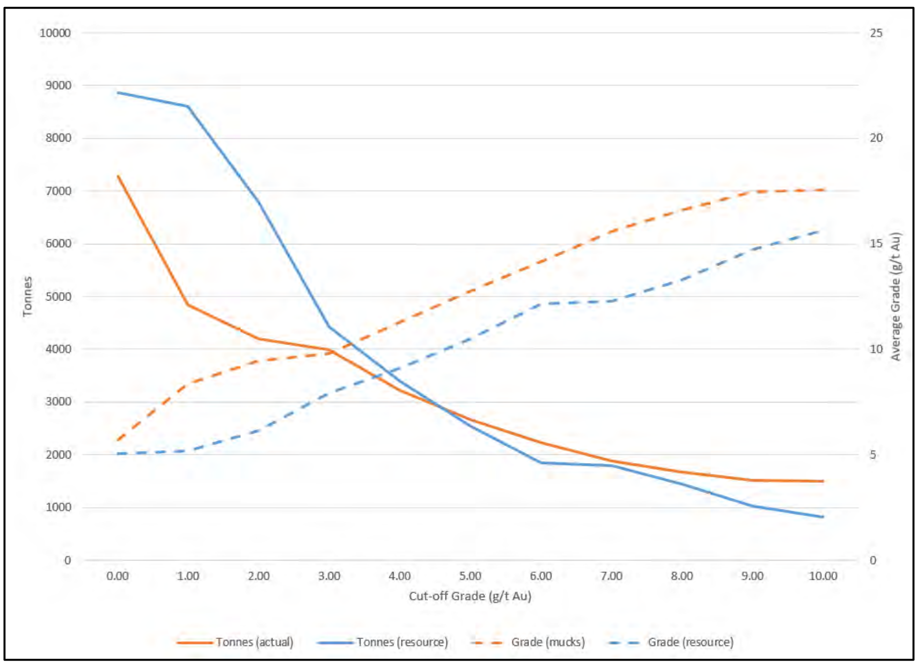
Table 14-30: Results from Test Mining - Total Mineralized Material

	Actual - Uncapped Grades			Actual - Capped Grades*		
Stockpile	**Tonnes**	**Au (g/t)**	**Ounces**	**Tonnes**	**Au (g/t)**	**Ounces**
High Grade (>4 g/t Au)	4,953	11.6	1,847	4,953	10.2	1,627
Low Grade (1-4 g/t Au)	2,143	2.5	169	2,143	2.5	169
Total	**7,096**	**8.8**	**2,016**	**7,096**	**7.9**	**1,797**

Source: Pure Gold (2019)

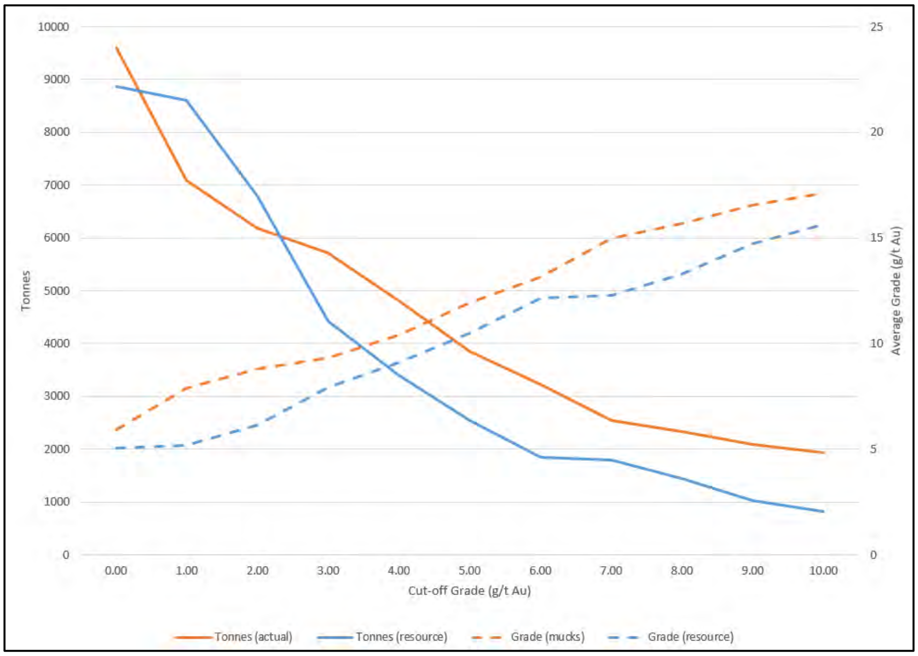
The actual tonnes mined in the bulk sample correlate very well with the resource model tonnes at all grades above a 4 g/t Au cut-off (Figure 14-43), with grade remaining consistently higher throughout. Adding in the tonnes from the Hanging Wall zone gives a slightly lower grade overall, but the increased tonnes provide significantly more gold ounces.

Figure 14-42: Grade Tonnage Curves Showing Actual versus Predicted Mined Material in the East and West Stoping Areas



Source: Pure Gold (2019)

Figure 14-43: Grade Tonnage Curves Showing Actual versus Predicted Mined Material Including All Zones



Source: Pure Gold (2019)

14.1.15 Discussion and Recommendations

The estimation of the mineral resources in an environment with a long and extensive mining history is a challenging assignment. It is believed that all reasonable efforts were carried out by SRK in 2009 and more recently by Pure Gold to gather all of the information available to a quality standard acceptable for the estimation of a Mineral Resource.

The uncertainty associated with the historical drilling was addressed by Pure Gold's recent drilling, where the overall grades were confirmed, and modern data was gathered to allow for the construction of a more robust geological model that relies on a property scale geological framework developed through extensive study. Comparisons of the statistics at Madsen with only the historical drill holes versus only the Pure Gold drill holes show similar results, providing additional confidence to the historical data.

A comparison of the mineral resources within the mined stopes and the historical production represents a valuable assessment of the estimation results. It was observed that the tonnage of all the underground excavations in the model (stopes, shaft, drifts, raises, and ramp) matches very well the tonnage of the historical production. However, in this comparison, it was also noted that the gold grades of the estimates and the past production are quite different, with the estimates being of much lower grade. One plausible explanation for this difference might come from the fact that the mined stope outlines were derived from more information than that provided by the drill holes. Examination of original paper sections and level plans indicate that channel, chip, and face samples were utilized to define the stope shapes in addition to the drill holes. In many cases, areas outside the currently modeled high-grade zones were added to the stopes based on channel/chip/face samples results indicating high-grade gold mineralization in these areas without any drill information. In the current estimate, these areas would be identified as part of the low-grade zone, which could explain the lower resulting gold grade observed from the current estimate within the underground excavations.

It should be noted that the tonnage and average gold grade calculated from the current estimate within the underground excavations matches that from the 2009 SRK estimate.

The statistics on gold grades within the high-grade zones indicated heterogeneous populations with higher coefficients of variability. Although an estimation technique, such as indicator kriging, could be recommended to provide more restrictive constraints to the higher grade portion, the validation tests on the current grade estimates show satisfying results without any overestimation of the higher-grade fraction. It is thus believed that the usage of ordinary kriging on capped gold grades provides satisfactory results in this case.

The large abundance of drill hole gold assays on a densely spaced grid within the resource area has allowed for well-defined variograms. This has brought additional confidence in the modeling of the experimental variograms.

The validation tests have indicated that in general the estimates have levels of smoothing closer to the upper limit of acceptability. For such, it is recommended that a reduction of the maximum number of samples, used in the grade estimation process, be investigated in order to decrease the level of smoothing.

This update of the mineral resource estimate has shown increases in tonnage, average gold grade, and metal content overall. These changes can be explained by the fine-tuning of the geologic model from the additional holes drilled by Pure Gold, the addition of the HG5 domain at Austin, and the re-classification of estimates peripheral to design stopes into the indicated category.

Similarly to previous estimates (2009 and 2017), there were no estimates categorized in the measured class. This is to account, in part, for some uncertainty associated with the historical data, but also to acknowledge the differences observed between the estimated grade in the stopes and the reported production. It is likely that Madsen, as with many other high-grade gold mines of this type, will require more detailed information such as underground development in mineralized zones, combined with closely-spaced chip sampling, to achieve a “Measured” level of definition.

The sensitivity studies carried out on the estimates address possible variations of the base case scenario. The usage of the samples outside mined stopes only for the grade estimation process has not shown any significant departure from the mineral resources of the base case, especially with regards to the average gold grades, where most of the differences would be expected. This observation can be mainly explained by the fact that although all samples inside and outside mined stopes were utilized in the base case scenario, the usage of the 15ft geotechnical buffer in the periphery of the mined stopes has dampened the effect of in-stope higher gold grades. A second sensitivity scenario examining the effect of hard boundaries between high-grade veins within each domain has shown a decrease in tonnage at approximately the same average gold grade. Based on the current understanding and experience gained on this project, the estimates derived from the base case scenario with soft boundaries between the high-grade veins of each domain, is believed to be more reflective of the mineralization encountered at Madsen.

The reconciliation of the bulk sample to the block model is a good indication of results expected at the mining stage. Local control from geological mapping and chip sampling has proven to be highly effective at tracking the mineralization on a round-by-round basis. The precise location of the high grade material may diverge slightly from the resource model in areas with less drilling; however, the overall tonnes and grade predicted by the model can be expected to be present at the scale of a defined stope. In addition to matching the tonnes and grade of the resource model, there are also many opportunities for finding more ore in areas that remain poorly drilled adjacent to the known resource areas. Nearly 50% more high grade material was identified using the short range mining scale data available as the bulk sample mining progressed. The key to future success will likewise depend on a strong grade control program, similar to that implemented during this bulk sample program, including detailed geological mapping, chip sampling, muck sampling and bazooka drilling.

Overall, it is believed that the current estimation is a realistic representation of the Mineral Resources at the Madsen project, based on the current geologic understanding and available information.

14.2 Fork and Russet South Deposits

14.2.1 Drill Hole Data

This section describes the mineral resource estimates for the Fork and Russet South gold deposits, which have been updated since the December 2017 mineral resource estimates. The drill hole databases were provided by Pure Gold with cut-off dates of July 10, 2018 for the Fork deposit and August 10, 2018 for the Russet South deposit. Five new holes were drilled at Fork, while 24 new holes were drilled at Russet South within the resource area. All collar coordinates are in a UTM projection. The content of the drill hole databases for each area is summarized in Table 14-31.

Table 14-31: Drill Hole Summary – Fork and Russet South Deposits – Resource Areas

Deposit	Number of Holes	Metres	Number of Assays
Fork	122	45,525.4	20,121
Russet South	105	26,566.7	15,199

Source: Ginto Consulting Inc. (2019)

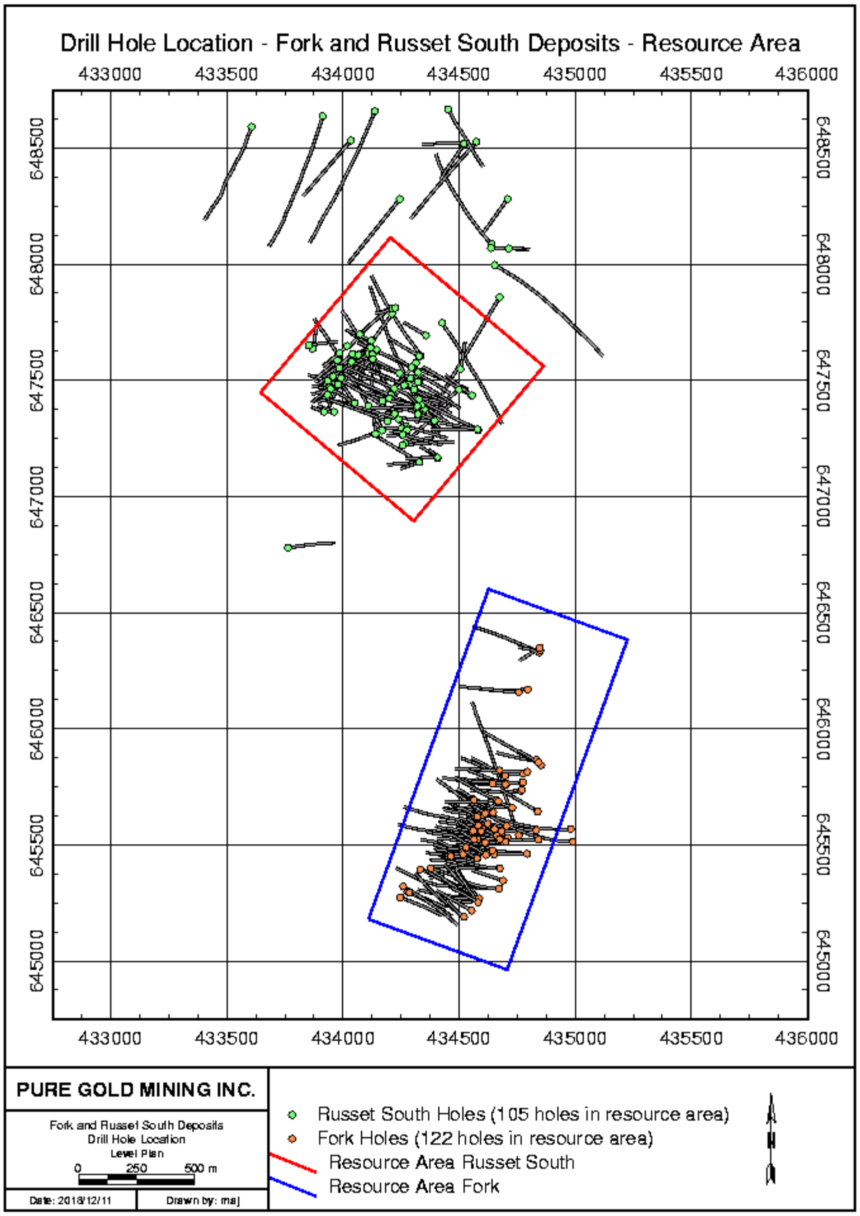
Statistics on drill hole spacing are presented in Table 14-32 for the Fork and Russet South deposits within the mineralized domains. The location of the drill holes in both deposits is shown in Figure 14-44 (note that the first digit of the northing coordinates was truncated in this figure).

Table 14-32: Drill Hole Spacing Statistics – Fork and Russet South Deposits

Deposit	Mean (m)	Median (m)
Fork	27.7	23.6
Russet South	21.0	20.1

Source: Ginto Consulting Inc. (2019)

Figure 14-44: Drill Hole Location – Fork and Russet South Deposits



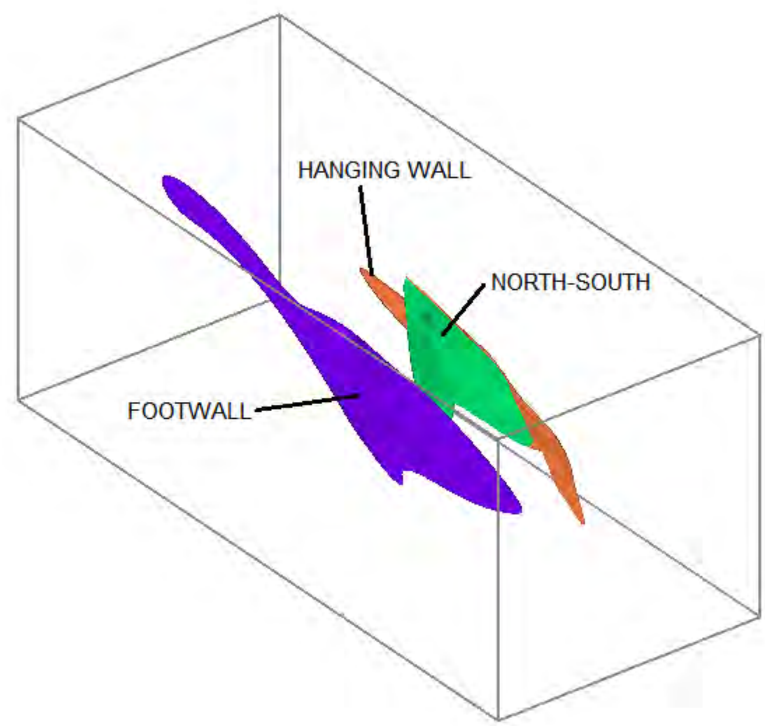
Source: Ginto Consulting Inc. (2019)

14.2.2 Geologic Modelling

The geologic model of the Fork and Russet South deposits were reviewed and updated with the new drill hole data. The domains modelled are comprised of the mineralized zones for each deposit.

For the Fork deposit, the geologic model is comprised of three domains: the hanging wall domain, the footwall domain, and the north-south domain. The hanging wall and footwall domains trend northeast and dip southeast and are spatially linked to discrete stratigraphic horizons. In the hanging wall domain, mineralization is predominantly localized in and around iron formation and ultramafic units; whereas, the footwall domain is hosted in basalt lenses within the deeper Russet Lake ultramafic body. The north-south domain is more steeply-dipping than the other two domains and transects the stratigraphy between them. It encloses a strongly silicified corridor that contains most of the higher grade intercepts in the deposit and is interpreted to represent a primary mineralizing structure. The wireframes of the three domains are displayed in Figure 14-45.

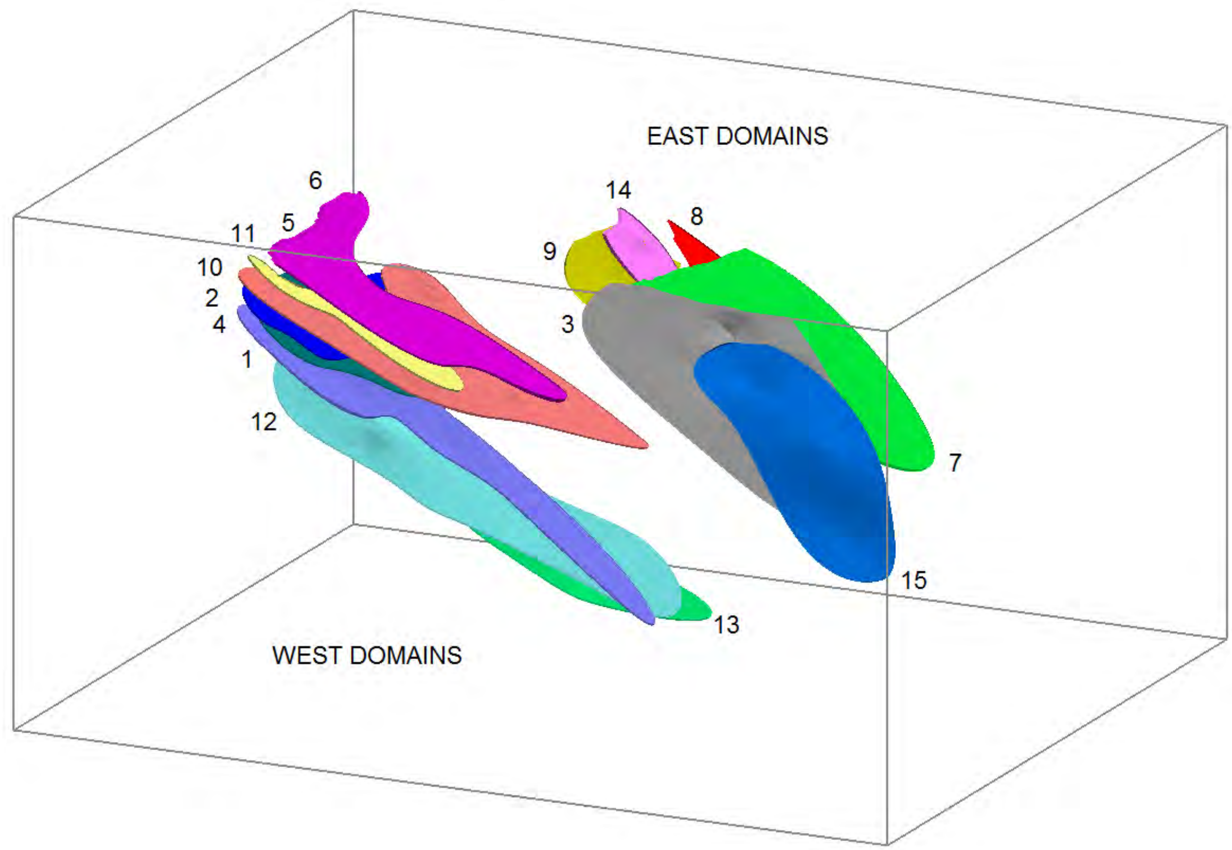
Figure 14-45: Geologic Model of the Fork Deposit: Hanging Wall, Footwall, and North-South Domains. Viewed to the Northeast



Source: Ginto Consulting Inc. (2019)

The geologic model of the Russet South deposit is comprised of fifteen domains that occupy the hinge zone of a broad, open F2 fold defined by iron formation and ultramafic units along with their enclosing volcanic stratigraphy. The deposit is separated into western and eastern areas, as seen in Figure 14-46. The western area is made up of nine sub-parallel domains with a northeast trend and shallow dip of approximately -40° to the southeast, sub-parallel to the plunging hinge line of the broad F2 fold. The eastern area is made up of six domains, where two of the domains are northeast-trending and dip to the southeast, two domains north-south trending and dipping to the east, and two other domains oriented parallel to the northwest-trending, steeply-dipping axial plane of the F2 fold. The latter domains correspond to a discrete quartz vein that outcrops over a 150 m strike length.

Figure 14-46: Geologic Model of the Russet South Deposit: 9 domains in the western area, 6 domains in the Eastern area. Viewed to the Northeast



Source: Ginto Consulting Inc. (2019)

As for the Madsen Main deposit, the three-dimensional modelling of the Fork and Russet South deposits was performed using Leapfrog Geo™ software; specifically, the Vein Modelling Tool. The domains were wireframed based on mineralized composites with a cut-off grade of 3 g/t Au. These composites were grouped according to their three dimensional position relative to each other and their surrounding geological context. While the mineralization in both deposits tends to occur in broad zones of millimetre- to centimetre-scale veining and disseminations, the domains attempted to restrict the domains to narrower zones of higher grade material, where the continuity is more easily recognized.

From the geologic models for the Fork and Russet South deposits, a set of rock codes was defined as presented in Table 14-33.

Table 14-33: Geologic Domain Codes for the Madsen Satellite Deposits

Deposit	Domain Codes	Volume m^3	Domain Codes	Volume m^3
Fork	1- footwall	914,457.8		
	2– hanging wall	652,699.4		
	3– north-south	316,085.9		
Russet South	1- domain 1 west	71,062.4	9- domain 9 east	21,441.0
	2– domain 2 west	28,979.2	10- domain 10 west	123,288.6
	3– domain 3 east	137,334.9	11- domain 11 west	23,187.6
	4– domain 4 west	41,463.5	12 – domain 12 west	81,074.9
	5– domain 5 west	5,736.8	13- domain 13 west	34,198.4
	6– domain 6 west	112,426.7	14- domain 14 east	64,298.3
	7- domain 7 east	175,560.5	15- domain 15 east	108,304.7
	8- domain 8 east	104,841.8		

Source: Ginto Consulting Inc. (2019)

The topographic surface for the Madsen project area was utilized to limit the grade estimates. It was obtained by a Lidar survey down-sampled to a 5 m resolution.

14.2.3 Compositing

Statistics were computed on the original sample lengths for each of the Fork and Russet South deposits.

For the Fork deposit, the most common sampling length was noted to be 1.52 m with 12% of the assays sampled at this interval. The original samples were thus composited to regular 1.52 m lengths, providing a satisfactory ratio of 1:2 with the block height (3 m), as well as preserving the intrinsic variability of the gold assay populations.

At Russet South, the most common sampling length was found to be 2.0 m with 33% of the data sampled on this interval. Due to the low ratio of composite length to block height of 1:1.5, it was decided to composite the original samples to 1.0 m regular intervals. This would provide a more acceptable ratio of composite length to block height (1:3), while still preserving the gold population's variability.

The compositing process consisted in starting the compositing at the collar of each hole with continuous composite intervals. At the contact with a different unit from the geology model, a last interval was composited, while a new set of regular composite lengths is generated within the other units. A summary of statistics on the composites at Fork and Russet South is presented in Table 14-34.

Table 14-34: Drill Hole Composites Summary at Fork and Russet South

Deposit	Domain	# of Holes	# of Composites	%	# of Metres	%	Average Au Grade g/t
Fork	mineralization	122	727	2.4	950.5	2.1 2	1.45
	total	122	30,206	100.0	44,931.9	100.0	0.07
Russet South	mineralization	105	1,700	3.5	1,590.7	3.3	1.36
	total	173	48,680	100.0	48,355.0	100.0	0.09

Source: Ginto Consulting Inc. (2019)

14.2.4 Exploratory Data Analysis (EDA)

A set of various statistical applications was utilized to provide a better understanding of the gold grade populations within the various mineralized zones.

14.2.4.1 Univariate Statistics

Basic statistics were performed on the gold grades of the Fork and Russet South composites. Histograms and probability plots indicated that the gold grade distributions resemble positively skewed lognormal populations. Basic statistics results were also obtained from boxplots for each mineralized domain modeled from each of the two deposits.

At Fork, statistical results show heterogeneous populations with high coefficients of variation (CV), varying from 3.2 to 4.3. Similar results are observed at Russet South with CVs of 3.3 and 5.1 for the western and eastern areas, respectively. The higher variability observed for the mineralized domains overall appears to be derived by high grade outliers.

14.2.4.2 Capping of High-Grade Outliers

It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results from specific statistical utilities. This procedure is performed on high-grade values that are considered outliers and that cannot be related to any geologic feature. In the case at Fork and Russet South, the higher gold grades were examined with three different tools: the probability plot, decile analysis, and cutting statistics. The usage of various investigating methods allows for a selection of the capping threshold in a more objective and justified manner. The resulting compilation of the capping thresholds is listed in Table 14-35.

Table 14-35: List of Capping Thresholds of Higher Gold Grade Outliers

Domain	Units	Capping Threshold g/t	% Metal Affected	Number of Comps Capped
Fork	1- footwall	30.0	14	1
	2– hanging wall	22.0	12	2
	3– north-south	32.0	6	3
Russet	1- domain 1 west	15.0	14	3
	2– domain 2 west	15.0	45	2
	3– domain 3 east	12.0	6	1
	4– domain 4 west	15.0	72	2
	5– domain 5 west	15.0	28	1
	6– domain 6 west	30.0	20	4
	7- domain 7 east	15.0	26	1
	8- domain 8 east	9.0	14	2
	9- domain 9 east	1.0	74	2
	10- domain 10 west	17.0	6	2
	11- domain 11 west	12.0	43	2
	12- domain 12 west	30.0	10	3
	13- domain 13 west	-	-	-
	14- domain 14 east	7.0	4	1
	15- domain 15 east	10.0	6	2

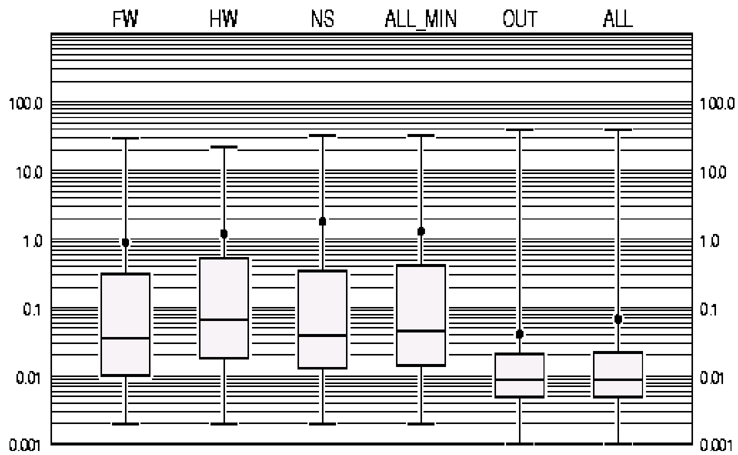
Source: Ginto Consulting Inc. (2019)

As seen in Table 14-35, the capping analysis showed that few high-grade outliers carry a significant portion of the metal content in some of the mineralized units.

Basic statistics were re-computed with the gold grades capped to the thresholds listed in Table 14-35. Boxplots of Figure 14-47 for Fork, and Figures 14-48 and 14-49 for Russet South, display the basic statistics resulting from the capping of the higher gold grade outliers. It can be observed from those figures that the coefficients of variation are in general below or close to 3.0 for the different gold grade populations. However, a few units display a coefficient of variation greater than 3.0, as seen for the footwall domain at Fork, and domains 3 and 8 at Russet South.

The effect of the capping of higher gold grade outliers has reduced the overall mean gold grade by 10.1 % at Fork, and by 23.7 % at Russet South.

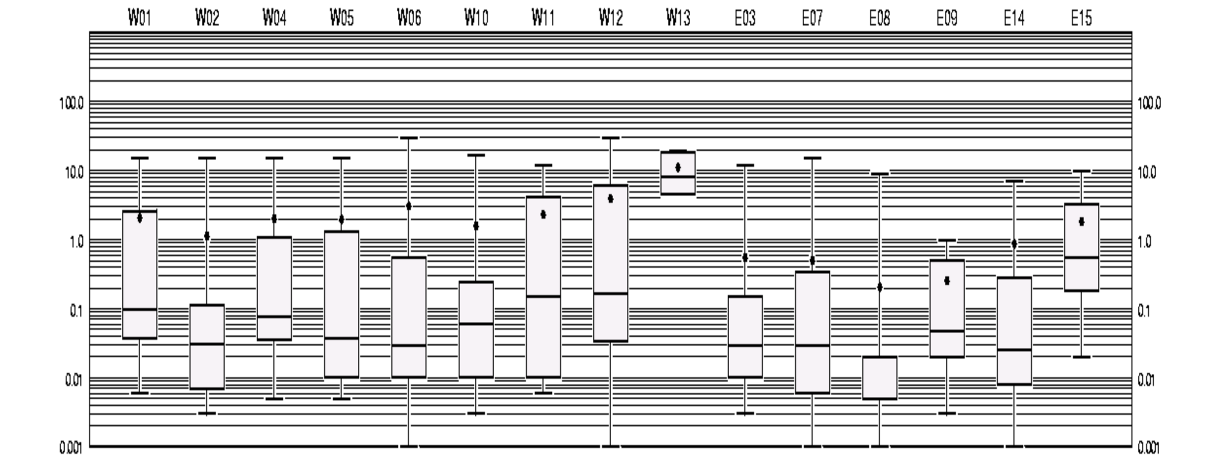
Figure 14-47: Boxplots of Capped Gold Composites – Fork Deposit – Au 1.52m Composites in g/t



	FW	HW	NS	ALL_MIN	OUT	ALL	
Number of data	198	308	221	727	29479	30206	Number of data
Mean	0.904	1.201	1.797	1.306	0.041	0.068	Mean
Std. Dev.	3.229	3.104	5.29	3.968	0.336	0.69	Std. Dev.
Coef. of Var.	3.573	2.586	2.943	3.037	8.186	10.184	Coef. of Var.
Maximum	30.0	22.0	32.0	32.0	39.712	39.712	Maximum
Upper quartile	0.312	0.519	0.338	0.418	0.021	0.022	Upper quartile
Median	0.035	0.065	0.039	0.045	0.009	0.009	Median
Lower quartile	0.01	0.018	0.013	0.014	0.005	0.005	Lower quartile
Minimum	0.002	0.002	0.002	0.002	0.001	0.001	Minimum
Number of holes	57	104	53	122	122	122	Number of holes

Source: Ginto Consulting Inc. (2019)

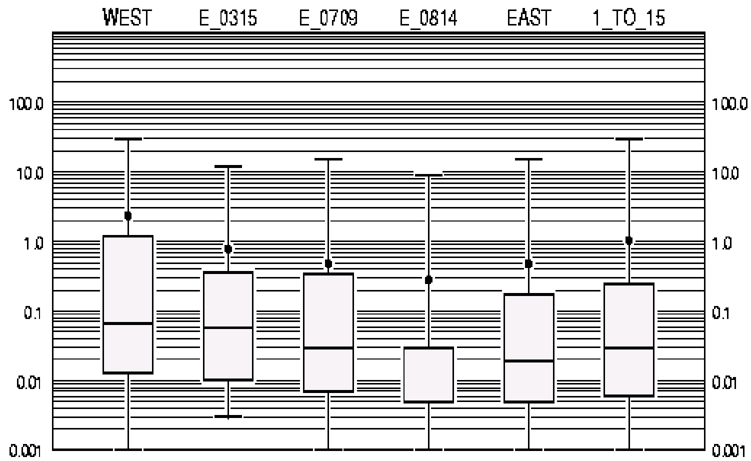
Figure 14-48: Boxplots of Capped Gold Composites – Russet South Deposit – Au 1.0m Composites in g/t by Domain



Number of data	61	47	39	30	125	151	28	52	6	251	330	433	27	61	59	Number of data
Mean	2.063	1.128	2.007	1.951	3.062	1.58	2.299	3.9	11.097	0.547	0.498	0.205	0.254	0.874	1.812	Mean
Std. Dev.	3.92	3.345	4.234	3.828	7.331	3.566	3.581	7.426	6.539	1.848	1.539	0.923	0.342	1.886	2.572	Std. Dev.
Coef. of Var.	1.9	2.964	2.109	1.962	2.394	2.257	1.557	1.904	0.589	3.381	3.092	4.51	1.346	2.157	1.419	Coef. of Var.
Maximum	15.0	15.0	15.0	15.0	30.0	17.0	12.0	30.0	19.64	12.0	15.0	9.0	1.0	7.0	10.0	Maximum
Upper quartile	2.621	0.113	1.071	1.283	0.56	0.24	4.152	6.09	18.571	0.151	0.341	0.02	0.499	0.281	3.323	Upper quartile
Median	0.099	0.031	0.078	0.038	0.03	0.059	0.153	0.168	7.99	0.03	0.03	0.005	0.047	0.025	0.562	Median
Lower quartile	0.037	0.007	0.036	0.01	0.01	0.01	0.01	0.034	4.519	0.01	0.006	0.005	0.02	0.008	0.18	Lower quartile
Minimum	0.006	0.003	0.005	0.005	0.001	0.003	0.006	0.001	4.503	0.003	0.001	0.001	0.003	0.001	0.02	Minimum
Number of holes	23	17	14	10	34	41	12	19	3	6	16	21	9	21	6	Number of holes

Source: Ginto Consulting Inc. (2019)

Figure 14-49: Boxplots of Capped Gold Composites - Russet South Deposit – Au 1.0m Composites in g/t Grouped Domains



	WEST	E_0315	E_0709	E_0814	EAST	1_TO_15	
Number of data	539	310	357	494	1161	1700	Number of data
Mean	2.351	0.782	0.481	0.277	0.477	1.035	Mean
Std. Dev.	5.363	2.062	1.49	1.09	1.543	3.311	Std. Dev.
Coef. of Var.	2.282	2.636	3.096	3.933	3.233	3.2	Coef. of Var.
Maximum	30.0	12.0	15.0	9.0	15.0	30.0	Maximum
Upper quartile	1.209	0.361	0.341	0.03	0.176	0.24	Upper quartile
Median	0.065	0.057	0.03	0.005	0.019	0.03	Median
Lower quartile	0.013	0.01	0.007	0.005	0.005	0.006	Lower quartile
Minimum	0.001	0.003	0.001	0.001	0.001	0.001	Minimum
Number of holes	66	12	25	30	52	105	Number of holes

Source: Ginto Consulting Inc. (2019)

14.2.4.3 Declustering

In general, there is a tendency to drill more holes in higher grade areas than in lower grade areas when delimiting a potential ore body. As a result, the higher grade portion of a deposit will be overly represented and would translate into a bias towards the higher grades when calculating statistical parameters of the population. Thus, a declustering method is utilized to generate a more representative set of statistical results within the zone of interest. In this case, a polygonal declustering technique was applied to the composites of the mineralized zones of Fork and Russet South.

Comparisons of average gold capped and declustered grades with the capped and un-declustered gold averages show some clustering for the two deposits with higher grades located in less densely drilled areas. As a result, the average gold grades of the declustered and capped composites increased by 44.1% at Fork, and 46.3% at Russet South.

The average grade from the declustered statistics provides an excellent comparison with the average grade of the interpolated blocks, as a way to assess any overall bias of the estimates.

14.2.5 Variography

A variographic analysis was carried out on the gold grade composites within the different mineralized units at Fork and Russet South. The objective of this analysis was to spatially establish the preferred directions of gold grade continuity. In turn, the variograms modeled along those directions would be later utilized to select and weigh the composites during the block grade interpolation process. For this exercise, all experimental variograms were of the type relative lag pairwise, which is considered robust for the assessment of gold grade continuity.

Due to the relatively low number of available composites in some of the mineralized units, it was unfeasible to develop conclusive variograms. In general, units with more than 300 composites, allowed for more definite variogram models.

The directions of gold grade continuity are in general agreement with the orientation of the mineralized zones, with best directions of continuity trending along strike and dip directions.

At Fork, a sufficient number of composites permitted for the definition of a variogram model for each of the three domains, with a better outcome for the hanging wall domain. Parameters from the variogram models are shown in Table 14-36.

Table 14-36: Modeled Variogram Parameters for Gold Composites at Fork

Parameters	1 – footwall domain			2 – hanging wall domain			3 – north-south domain		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	15°	105°	105°	30°	120°	120°	20°	110°	110°
Dip**	5°	-70°	20°	-5°	-60°	30°	-5°	-75°	15°
Nugget Effect C_0	0.096			0.250			0.222		
1st Structure C_1	0.973			0.931			0.592		
2nd Structure C_2	0.930			0.957			1.366		
1st Range A_1	38.1m	48.8m	2.1m	17.8m	14.6m	6.0m	60.9m	51.2m	8.2m
2nd Range A_2	48.8m	53.1m	9.2m	57.6m	39.4m	11.4m	65.2m	51.2m	13.5m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

At Russet South, the variographic analysis was conducted on domains grouped by west domains (domains 1,2,4,5,6,10,11,12,13) and east domains (domains 7,9; domains 3,15; and domains 8,14) to allow for the development of a variogram model. The variogram model parameters for the west and east areas are presented in Table 14-37.

Table 14-37: Modeled Variogram Parameters for Gold Composites at Russet South

Parameters	1 – West: Domains 1,2,4,5,6,10,11,12,13			2 – East: Domains 7,9		
	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	35°	125°	125°	45°	135°	135°
Dip**	0°	-40°	50°	0°	-45°	45°
Nugget Effect C_0	0.368			0.166		
1st Structure C_1	0.698			1.145		
2nd Structure C_2	1.049			1.240		
1st Range A_1	27.3m	14.5m	26.3m	18.1m	26.4m	8.8m
2nd Range A_2	53.0m	58.3m	29.5m	42.1m	46.7m	15.3m
Parameters	**3- East Domains 3,15**			**4- East Domains 8,14**		
	Principal	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**
Azimuth*	125°	35°	35°	10°	100°	100°
Dip**	0°	-65°	25°	0°	-40°	50°
Nugget Effect C_0	0.236			0.138		
1st Structure C_1	0.707			1.175		
2nd Structure C_2	1.208			0.951		
1st Range A_1	20.3m	25.3m	10.3m	40.8m	25.7m	13.1m
2nd Range A_2	44.8m	45.3m	17.5m	51.7m	39.1m	19.0m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

14.2.6 Gold Grade Estimation

The estimation of gold grades into a block model was carried out with the ordinary kriging technique for all domains. The estimation strategy and parameters were tailored to account for the various geometrical, geological, and geostatistical characteristics previously identified. A separate block model of gold grade estimates was assigned to each of the Fork and Russet South deposit zones with a total of 2 block models: Fork and Russet South. Each block model has a specific grid definition, as presented in Table 14-38. It should be noted that the origin of the block model corresponds to the lower left corner, the point of origin being the exterior edges of the first block. The same block size of 3 m (easting) x 3 m (northing) x 3 m (elevation) was selected for the Fork and Russet South deposits to better reflect the geometrical configuration and anticipated underground production rate. The block models for Fork and Russet South were rotated at angles of 20° clockwise and 40° clockwise, respectively.

Table 14-38: Block Grid Definitions

Coordinates	Origin m	Rotation (azimuth of X axis)	Distance m	Block Size m	Number of Blocks
Fork					
Easting (X)	434,110.0	110°	636.0	3.0	212
Northing (Y)	5,645,180.0		1,512.0	3.0	504
Elevation(Z)	-210.0		630.0	3.0	210
Number of Blocks		22,438,080			
Russet South					
Easting (X)	433,645.0	130°	900.0	3.0	300
Northing (Y)	5,647,448.0		750.0	3.0	250
Elevation(Z)	-50.0		501.0	3.0	167
Number of Blocks		12,525,000			

Source: Ginto Consulting Inc. (2019)

The respective databases of capped gold grade composites were utilized as input for the grade interpolation process at Fork and Russet South.

The size and orientation of the search ellipsoids for the estimation process was based on the variogram parameters modeled for gold. A minimum of 2 samples and maximum of 12 samples were selected for the block grade calculations. Hard boundaries were assigned in the estimation of each unit. A summary of the estimation parameters is presented in Table 14-39.

For certain domains with few composites and more isolated high grade intercepts, a restrictive search for high yielding composites was utilized to constrain their extrapolation during the estimation process. A summary of the constraints applied in such domains is presented in Table 14-40.

Table 14-39: Estimation Parameters for Gold

Estimation Parameters – Gold Grade									
Rock Code	Method	Min # of Samples	Max # of Samples	Search Ellipsoid – Long Axis – Azimuth / Dip	Search Ellipsoid – Long Axis - size	Search Ellipsoid – Short Axis – Azimuth / Dip	Search Ellipsoid – Short Axis - Size	Search Ellipsoid – Vertical Axis – Azimuth / Dip	Search Ellipsoid – Vertical Axis - Size
Fork									
1	OK	2	12	15°/5°	49.0m	105°/-70°	53.0m	105°/20°	10.0m
2	OK	2	12	30°/-5°	58.0m	120°/-60°	39.0m	120°/30°	11.0m
3	OK	2	12	20°/-5°	65.0m	110°/-75°	51.0m	110°/15°	14.0m
Russet South									
1	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
2	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
3	OK	2	12	125°/0°	45.0m	35°/ -65°	45.0m	35°/ 25°	18.0m
4	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
5	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
6	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
7	OK	2	12	45 °/ 0°	42.0m	135°/ -45°	47.0m	135°/ 45°	15.0m
8	OK	2	12	10°/0°	52.0m	100°/ -40°	39.0m	100°/ 50°	19.0m
9	OK	2	12	45 °/ 0°	42.0m	135°/ -45°	47.0m	135°/ 45°	15.0m
10	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
11	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
12	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
13	OK	2	12	35°/0°	53.0m	125°/-40°	58.0m	125°/50°	30.0m
14	OK	2	12	10°/0°	52.0m	100°/ -40°	39.0m	100°/ 50°	19.0m
15	OK	2	12	125°/0°	45.0m	35°/ -65°	45.0m	35°/ 25°	18.0m

Source: Ginto Consulting Inc. (2019)

Table 14-40: Restrictive Search for High Yield Gold Composites

Rock Code	Au cut-off (g/t)	Restrictive search distance (m)		
		Principal	Minor	Vertical
Fork				
1- footwall	≥ 20.0	20.0	20.0	20.0
Russet South				
6- domain 6	≥ 10.0	50.0	50.0	50.0
13- domain 13	≥ 5.0	50.0	50.0	50.0
15- domain 15	≥ 5.0	45.0	45.0	45.0

Source: Ginto Consulting Inc. (2019)

The grade estimation process consisted of a three-pass approach with the parameters of the first pass as presented in Table 14-39. The estimation parameters of the second and third passes are the same with the exception of an enlarged search ellipsoid by 1.5 times and 3 times the dimensions from the first pass, respectively. In this case, priority was given to estimates from the first pass, followed by estimates from the second pass for un-estimated blocks from the first pass, and finally the estimates of the third pass for un-estimated blocks from the first and second passes. Only blocks within the mineralized wireframes were estimated.

14.2.7 Validation of Grade Estimates

Validation tests were carried out on the estimates to examine the possible presence of a bias and to quantify the level of smoothing / variability.

14.2.7.1 Visual Inspection

A visual inspection of the block estimates with the drill hole grades on plans and cross-sections was performed as a first check of the estimates. Observations from stepping through the estimates along the different planes indicated that there was good agreement between the drill hole grades and the estimates. The orientations of the estimated grades were also as expected according to the projection angles defined by the search ellipsoid.

14.2.7.2 Global Bias Test

The comparison of the average gold grades from the declustered composites and the estimated block grades examines the possibility of a global bias of the estimates. As a guideline, a difference between the average gold grades of more than ± 10% would indicate a significant over- or under-estimation of the block grades and the possible presence of a bias

Results of the average gold grade comparison are presented in Table 14-41 for the different mineralized zones at Fork and Russet South.

Table 14-41: Average Gold Grade Comparison – Polygonal-declustered Composites with Block Estimates

Statistics	Declustered Composites	Block Estimates
Fork		
Average Gold Grade g/t	1.49	1.44
Difference	-3.6%	
Russet South		
Average Gold Grade g/t	2.35	2.29
Difference	-2.8%	

Source: Ginto Consulting Inc. (2019)

As seen in Table 14-41, the average gold grades between the declustered composites and the block estimates are within the acceptable limits of tolerance for the Fork and Russet South deposits. No global bias is thus present in the gold grade estimates.

14.2.7.3 Local Bias Test

A comparison of the grade from composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scatterplot gives a statistical valuation of the estimates. The estimated block grades should be similar to the composited grades within the block, without being exactly the same value. Thus, a high correlation coefficient will indicate satisfactory results in the interpolation process, while a medium to low correlation coefficient will be indicative of larger differences in the estimates and would require a further review of the interpolation process. Results from the pairing of composited and estimated grades within blocks pierced by a drill hole are presented in Table 14-42 for the mineralized zones at Fork and Russet South.

As seen in Table 14-42 for gold, the block grade estimates are very similar to the composite grades within blocks pierced by a drill hole, with high correlation coefficients, indicating satisfactory results from the estimation process. No local bias is observed.

Table 14-42: Gold Grade Comparison for Blocks Pierced by a Drill Hole – Paired Composite Grades with Block Grade Estimates

Data	Average Gold Grade g/t	Correlation Coefficient
Fork		
Composites	1.53	0.916
Block Estimates	1.50	
Russet		
Composites	1.35	0.935
Block Estimates	1.26	

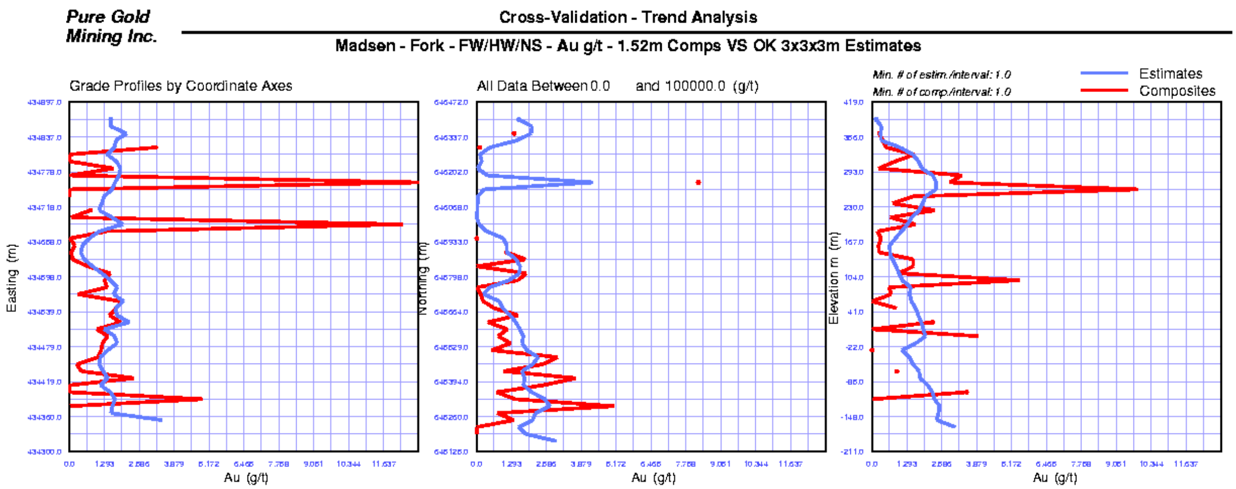
Source: Ginto Consulting Inc. (2019)

14.2.7.4 Grade Profile Reproducibility

The comparison of the grade profiles of the declustered composites with that of the estimates allows for a visual verification of an over- or under-estimation of the block estimates at the global and local scales. A qualitative assessment of the smoothing/variability of the estimates can also be observed from the plots. The output consists of three graphs displaying the average grade according to each of the coordinate axes (east, north, elevation). The ideal result is a grade profile from the estimates that follows that of the declustered composites along the three coordinate axes, in a way that the estimates have lower high-grade peaks than the composites, and higher low-grade peaks than the composites. A smoother grade profile for the estimates, from low to high grade areas, is also anticipated in order to reflect that these grades represent larger volumes than the composites.

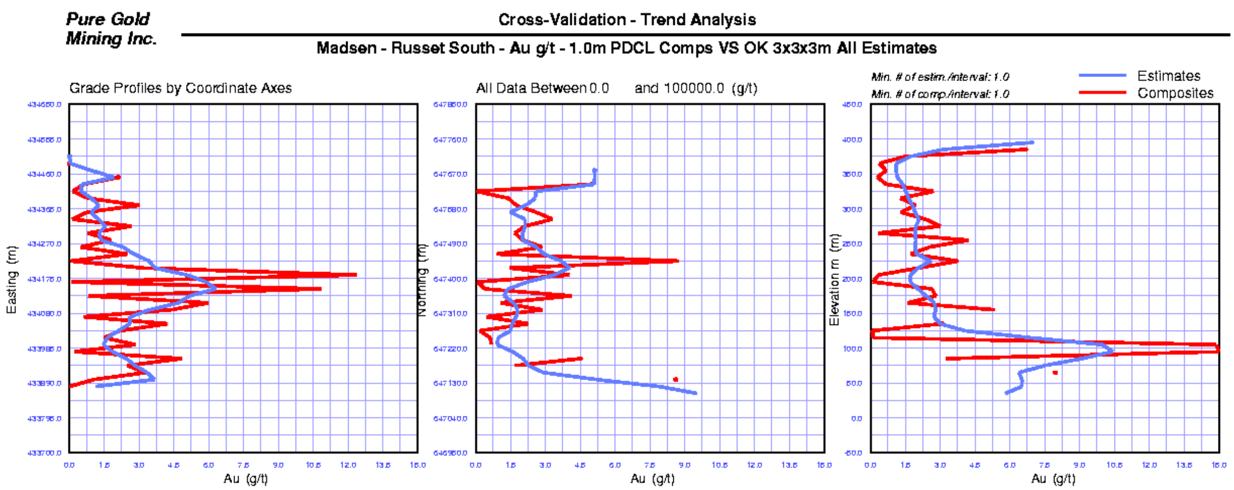
Gold grade profiles are presented in Figure 14-50 and 14-51 for the mineralized zones at Fork and Russet South, respectively.

Figure 14-50: Gold Grade Profiles of Declustered Composites and Block Estimates – Mineralized Zones at Fork



Source: Ginto Consulting Inc. (2019)

Figure 14-51: Gold Grade Profiles of Declustered Composites and Block Estimates – Mineralized Zones at Russet South



Source: Ginto Consulting Inc. (2019)

From the plots of Figure 14-50 and 14-51, it can be seen that the grade profiles of the declustered composites are well reproduced by those of the block estimates and consequently that no global or local bias is observed. As anticipated, some smoothing of the block estimates can be seen in the profiles, where estimated grades are higher in lower grade areas and lower in higher grade areas.

14.2.8 Resource Classification

The mineral resources were classified as indicated and inferred for both Fork and Russet South deposits. At Fork, a contour outlining areas of higher drill density was drawn for each domain. Gold grade estimates within the contour were classified as indicated, while those outside were classified as inferred. A different approach was selected for Russet South, where the distance from the second range of the variograms was chosen as the classification criterion. In this case, gold grade estimates with an average sample distance of 40 m or less were classified as indicated, while those with an average sample distance greater than 40 m were classified as inferred. All estimates from domains 3, 7, 12, 13, and 15 were classified as inferred due to the wider drilling spacing in these units. The mineral resource classification criteria for Fork and Russet South are summarized in Table 14-43. There were no mineral resources classified as measured at Fork and Russet South.

Table 14-43: Classification Distances

Mineralized Zone	Indicated	Inferred
Fork	Within higher drilling density contour	Outside higher drilling density contour
Russet South (domains 1,2,4,5,6,8,9,10,11,14)	≤ 40.0m	>40.0m

Source: Ginto Consulting Inc. (2019)

14.2.9 Editing of the Block Model

The block models of gold grade estimates for the Fork and Russet South deposits were edited to the topographic surface. The percentage of block below the topographic surface was stored in a variable and utilized in the tonnage calculation of each block. The topography variable was utilized to affect the specific gravity in the calculation of the block tonnage: sg x (below topo percentage) / 100.0. No previous mining has taken place at Fork or Russet South.

14.2.10 Mineral Resource Calculation

The mineral resource was calculated for 3 m (X) x 3 m (Y) x 3 m (Z) blocks with a constant specific gravity (SG) value of 2.94 for the Fork and Russet South deposits.

For the mineral resource's tonnage calculation, the fraction of each block within the mineralized zones and the block fraction below the topography surface were accounted for in the tonnage and grade calculations.

The indicated and inferred mineral resources for the Fork and Russet deposits are presented in Table 14-44 and 14-45, respectively. At Fork, the indicated mineral resources at a 4.0 g/t gold cut-off are 203,000 tonnes at an average gold grade of 6.62 g/t, for a total of 43,000 ounces of gold, while the inferred mineral resources at a 4.0 g/t gold cut-off are 331,000 tonnes at an average gold grade of 5.78 g/t, for a total of 61,000 ounces of gold. At Russet South, the indicated mineral resources at a 4.0 g/t gold cut-off are 242,000 tonnes at an average gold grade of 7.23 g/t, for a total of 56,000 ounces of gold, while the inferred mineral resources at a 4.0 g/t gold cut-off are 347,000 tonnes at an average gold grade of 7.58 g/t, for a total of 85,000 ounces of gold.

The mineral resources of the Madsen Main and Fork and Russet South deposits are presented in Table 14-46 at 4.0 g/t gold cut-off.

It should be noted that 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 estimated will be converted into mineral reserves. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The CIM definitions were followed for the classification of indicated and inferred mineral resources. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred mineral resources as an indicated mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated mineral resource category. All values in Table 14-44 to 14-46 have been rounded to reflect the relative accuracy of the estimates. Mineral resources are reported at a cut-off grade of 4.0 g/t gold based on US$1,200 per troy ounce gold and gold metallurgical recoveries of 92 percent.

Table 14-44: Mineral Resources* by Au Cut-off Grades - Effective February 5, 2019 – Fork Deposit

Au Cut-Off Grade (g/t)	Indicated			Inferred		
	Tonnage tonnes	Au Grade g/t	Au Content ounces	Tonnage tonnes	Au Grade g/t	Au Content ounces
1.0	577,000	3.71	69,000	1,393,000	3.13	140,000
2.0	384,000	4.84	60,000	932,000	3.94	118,000
3.0	273,000	5.81	51,000	606,000	4.73	92,000
4.0	203,000	6.62	43,000	331,000	5.78	61,000
5.0	149,000	7.41	35,000	170,000	7.12	39,000
6.0	91,000	8.64	25,000	85,000	8.70	24,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-45: Mineral Resources* by Au Cut-off Grades - Effective February 5, 2019 – Russet South Deposit

Au Cut-Off Grade (g/t)	Indicated			Inferred		
	Tonnage tonnes	Au Grade g/t	Au Content ounces	Tonnage tonnes	Au Grade g/t	Au Content ounces
1.0	670,000	4.07	88,000	908,000	4.36	127,000
2.0	494,000	4.99	79,000	667,000	5.37	115,000
3.0	340,000	6.14	67,000	517,000	6.23	104,000
4.0	241,000	7.23	56,000	352,000	7.54	85,000
5.0	169,000	8.42	46,000	277,000	8.38	75,000
6.0	119,000	9.68	37,000	194,000	9.60	60,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-46: Mineral Resources* at a 4.0 g/t Au Cut-off Grades - Effective December 1, 2018 – Madsen, Fork, and Russet South Deposits

Deposit	Indicated			Inferred		
	Tonnage tonnes	Au Grade g/t	Au Content ounces	Tonnage tonnes	Au Grade g/t	Au Content ounces
Madsen	6,429,000	8.99	1,857,000	889,000	8.42	241,000
Fork	203,000	6.62	43,000	331,000	5.78	61,000
Russet South	241,000	7.23	56,000	352,000	7.54	85,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

14.2.11 Discussion and Recommendations

The validation of the gold grade estimates at Fork and Russet South has shown satisfactory results with no global or local biases. The relatively heterogeneous nature of the gold mineralization from the Fork and Russet South deposits showed higher gold grades in drilled areas of lesser density. This particular situation was dealt with by utilizing a restrictive search for high yield assays in order to limit the over-generation of high-grade block estimates in these regions.

Although the drill hole spacing is adequate on average, the thinness of the mineralized zones does not allow for many intercepts from a single drill hole. Therefore, few samples are available for the assessment of gold grade continuity for each individual vein. In this study, the vein samples were grouped to provide a larger number of samples and allow for a variographic analysis. For such, both deposits would benefit from additional drill holes, especially in areas of isolated higher grades.

Overall, it is believed that the estimation of the mineral resources at Fork and Russet South are realistic representations, based on the current geologic understanding and available information.

14.3 Wedge Deposit

14.3.1 Drill Hole Data

This is the first mineral resource estimate of the Wedge deposit. The procedures and steps undertaken in the estimation of the gold grades into a block model are described below. The drill hole database for the Wedge deposit was provided by Pure Gold with a cut-off date of August 29, 2018. All collar coordinates are in a UTM projection. There is a total of 201 holes in the Wedge database with 26,126 assays for gold in g/t. These holes represent 52,237.7 m of drilling with an average drill hole length of 259.9 m. The content of the drill hole database is summarized in Figure 14-52.

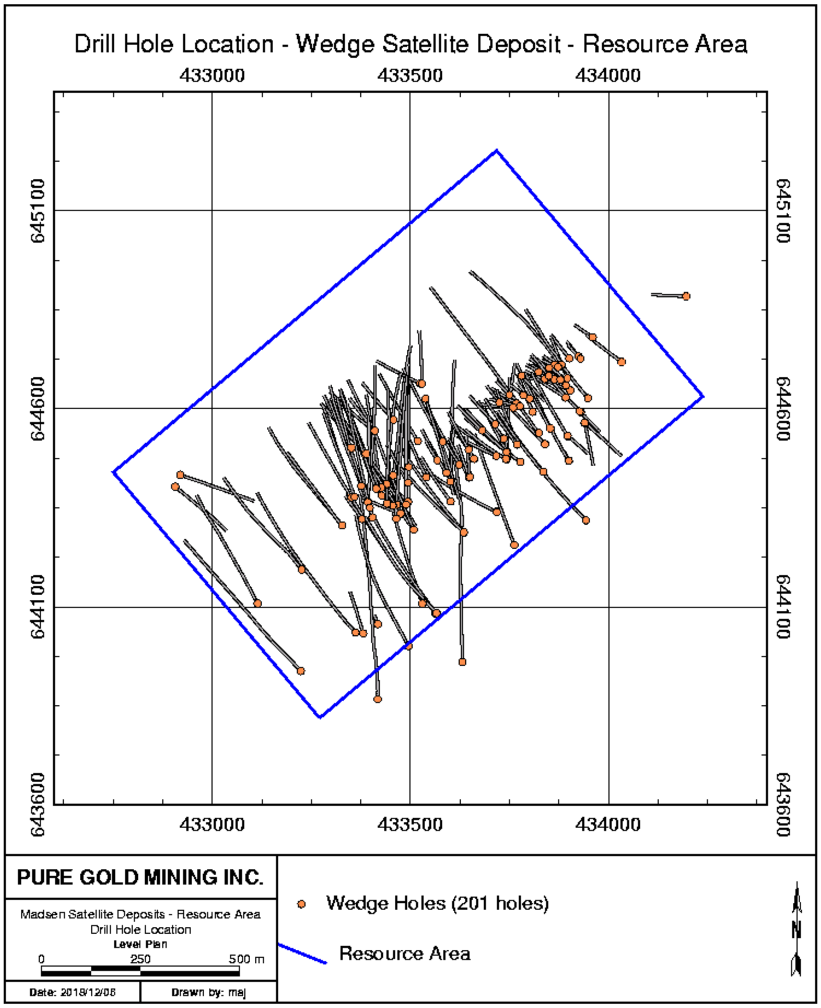
Figure 14-52: Drill Hole Database Summary – Wedge Deposit – Exploratory Data Analysis and Drill Hole Data Statistics

Collar Data	Number of Data	Mean	Standard Deviation	Coefficient of Variation	Minimum	Lower Quartile	Median	Upper Quartile	Maximum	Number of 0.0 values	Number of < 0.0 values
Easting (X)	201	433805.0	442.903	0.001	432906.0	433476.0	433744.0	433948.0	435929.0	—	—
Northing (Y)	201	645248.0	1401.7	0.002	643867.0	644400.0	644546.0	645770.0	649014.0	—	—
Elevation (Z)	201	390.993	10.376	0.027	373.43	381.92	393.04	398.43	424.3	—	—
Hole Depth	201	259.889	176.652	0.68	5.6	123.55	224.5	367.24	1001.97	—	—
Azimuth	201	270.817	95.834	0.354	0.0	269.67	306.5	324.51	360.0	—	—
Dip	201	-52.483	8.614	-0.164	-81.01	-57.83	-49.66	-45.15	-41.2	—	—
Overburden	201	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	—	—
Survey Data											
Azimuth	7740	277.411	92.783	0.334	0.0	271.45	311.4	330.38	359.98	—	—
Dip	7740	-52.795	8.617	-0.163	0.0	0.0	0.0	0.0	0.0	—	—
Assay Data											
Interval Length (from-to)	25753	1.826	1.256	0.688	0.15	1.22	1.87	2.0	28.1	0	0
AU_GPT	25753	0.172	3.065	17.865	0.001	0.005	0.01	0.03	354.0	0	373

Source: Ginto Consulting Inc. (2019)

The drill hole spacing is 23.4 m on average with a median of 21.5 m. The drill holes are mainly drilled at azimuths varying from 270° to 360°, with dips varying from -45° to -75°. A smaller set of holes are drilled at azimuth between 160° and 170° with dips between -45° and -50°. The location of the drill holes is shown in Figure 14-53 (note that the first digit of the northing coordinates was truncated in this figure).

Figure 14-53: Drill Hole Location – Wedge Deposit

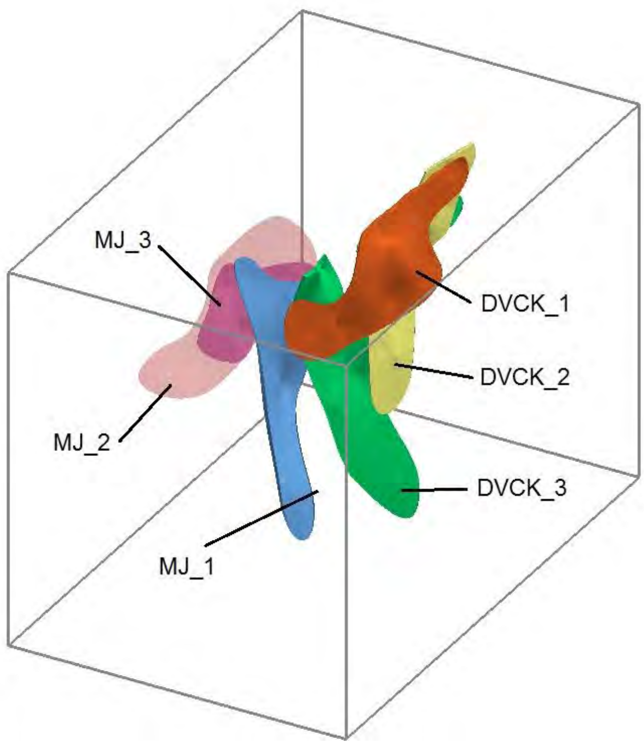


Source: Ginto Consulting Inc. (2019)

14.3.2 Geologic Modelling

The domains modelled for the geologic model at Wedge are made of six mineralized veins: the DVCK 1, 2, and 3 veins, and the MJ 1, 2, and 3 veins. These zones are oriented northeast at an azimuth of approximately 50°, steeply dipping at angles between 60° and 80° to the southeast. The wireframes of the various mineralized veins at Wedge are presented in Figure 14-54. The geology codes for each vein and their volume are found in Table 14-47.

Figure 14-54: Geologic Model of the Wedge Deposit - Viewed to the Northeast



Source: Ginto Consulting Inc. (2019)

Table 14-47: Geologic Domain Codes for the Wedge Deposit

Domain Codes	Volume (m^3)
1- DVCK-1 Vein	300,049.7
2- DVCK-2 Vein	411,662.8
3- DVCK-3 Vein	488,199.6
4- MJ-1 Vein	469,095.3
5- MJ-2 Vein	242,319.2
6- MJ-3 Vein	109,745.4

Source: Ginto Consulting Inc. (2019)

The topographic surface for the Madsen project area was utilized to limit the grade estimates at Wedge. It was obtained by a Lidar survey down-sampled to a 5 m resolution.

14.3.3 Compositing

Statistics were computed on the original sample lengths at Wedge. The most common sampling length was observed to be 1.0 m with 25% of the samples. Consequently, the samples were composited to regular intervals of 1.0 m to reflect the most common sampling length and to preserve the intrinsic variability of the gold assay populations.

The compositing process consisted in starting the compositing at the collar of each hole with continuous composite intervals. At the contact with a different unit from the geology model, a last interval was composited, while a new set of regular composite lengths is generated within the other units. A summary of statistics on the composites at Wedge is presented in Table 14-48.

Table 14-48: Drill Hole Composites Summary at Wedge

Deposit	Domain	# of Holes	# of Composites	%	# of Metres	%	Average Au Grade g/t
Wedge	mineralization	115	1,381	2.6	1,278.0	2.4	2.00
Wedge	total	201	52,550	100.0	52,238.4	100.0	0.10

Source: Ginto Consulting Inc. (2019)

14.3.4 Exploratory Data Analysis (EDA)

A set of various statistical applications was utilized to provide a better understanding of the gold grade populations within the mineralized zones.

14.3.4.1 Univariate Statistics

Basic statistics were performed on the gold grades of the Wedge composites. Histograms and probability plots indicated that the gold grade distributions resemble positively skewed lognormal populations. Basic statistics results were also obtained from boxplots for each mineralized domain modeled.

At Wedge, statistical results show heterogeneous populations with high coefficients of variation (CV), varying from 2.0 to 7.3. The higher variability observed for the mineralized domains overall appear to be derived by high grade outliers.

14.3.4.2 Capping of High-Grade Outliers

It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results from specific statistical utilities. This procedure is performed on high-grade values that are considered outliers and that cannot be related to any geologic feature. In the case at Wedge, the higher gold grades were examined with two different tools: the probability plot and cutting statistics. The usage of various investigating methods allows for a selection of the capping threshold in a more objective and justified manner. The resulting compilation of the capping thresholds is listed in Table 14-49.

Table 14-49: List of Capping Thresholds of Higher Gold Grade Outliers – Wedge Deposit

Domains	Capping Threshold g/t	% Metal Affected	Number of Comps Capped
1- DVCK-1 Vein	60.0	10	3
2– DVCK-2 Vein	30.0	2	3
3– DVCK-3 Vein	80.0	30	3
4- MJ-1 Vein	6.5	38	8
5- MJ-2 Vein	19.0	46	1
6- MJ-3 Vein	7.0	18	3

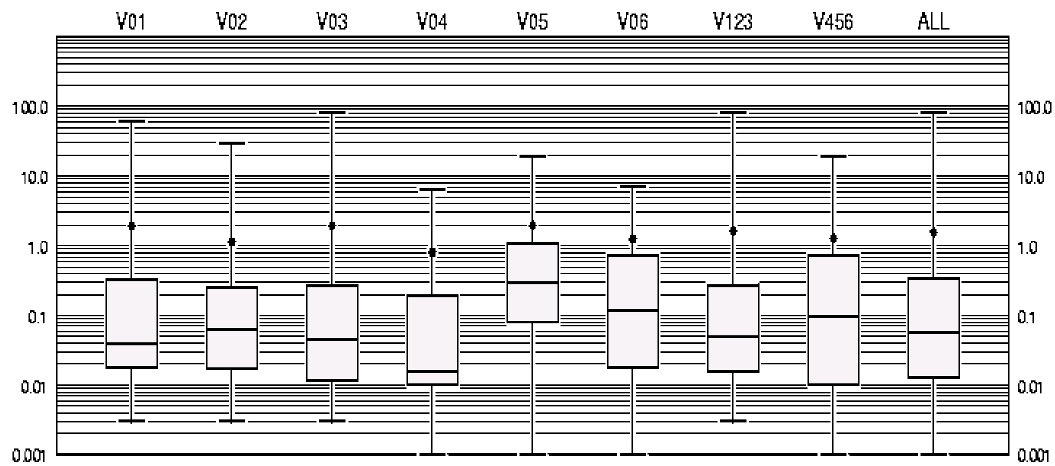
Source: Ginto Consulting Inc. (2019)

As seen in Table 14-49, the capping analysis showed that few high-grade outliers carry a significant portion of the metal content in some of the mineralized units.

Basic statistics were re-computed with the gold grades capped to the thresholds listed in Table 14-49. Boxplots of Figure 14-55 display the basic statistics resulting from the capping of the higher gold grade outliers. It can be observed from this figure that the coefficients of variation were reduced below a value of 3.0 for the MJ veins, while those of the DVCK veins remained above 3.0.

The effect of the capping of higher gold grade outliers has reduced the overall mean gold grade by 22% at Wedge.

Figure 14-55: Boxplots of Capped Gold Composites – Wedge Deposit – Au 1.0m Composites in g/t



	V01	V02	V03	V04	V05	V06	V123	V456	ALL	
Number of data	280	420	397	131	94	59	1097	284	1381	Number of data
Mean	1.922	1.141	1.942	0.816	1.951	1.248	1.627	1.274	1.557	Mean
Std. Dev.	7.633	3.707	8.939	1.813	3.941	2.276	7.005	2.813	6.391	Std. Dev.
Coef. of Var.	3.971	3.249	4.604	2.222	2.02	1.824	4.304	2.207	4.105	Coef. of Var.
Maximum	60.0	30.0	80.0	6.5	19.0	7.0	80.0	19.0	80.0	Maximum
Upper quartile	0.322	0.254	0.265	0.194	1.073	0.745	0.274	0.719	0.339	Upper quartile
Median	0.04	0.063	0.045	0.016	0.298	0.117	0.05	0.097	0.058	Median
Lower quartile	0.018	0.017	0.012	0.01	0.079	0.018	0.016	0.01	0.013	Lower quartile
Minimum	0.003	0.003	0.003	0.001	0.001	0.001	0.003	0.001	0.001	Minimum
Number of holes	56	52	63	28	23	19	97	38	201	Number of holes

Source: Ginto Consulting Inc. (2019)

14.3.4.3 Declustering

In general, there is a tendency to drill more holes in higher grade areas than in lower grade areas when delimiting a potential ore body. As a result, the higher grade portion of a deposit will be overly represented and would translate into a bias towards the higher grades when calculating statistical parameters of the population. Thus, a declustering method is utilized to generate a more representative set of statistical results within the zone of interest. In this case, a polygonal declustering technique was applied to the composites of the mineralized zones at Wedge.

Comparisons of average gold capped and declustered grades with the capped and un-declustered gold averages show some clustering with higher grades located in less densely drilled areas. As a result, the average gold grades of the declustered and capped composites increased by 109%.

The average grade from the declustered statistics provides an excellent comparison with the average grade of the interpolated blocks, as a way to assess any overall bias of the estimates.

14.3.5 Variography

A variographic analysis was carried out on the gold grade composites within the different mineralized domains at Wedge. The objective of this analysis was to spatially establish the preferred directions of gold grade continuity. In turn, the variograms modeled along those directions would be later utilized to select and weigh the composites during the block grade interpolation process. For this exercise, all experimental variograms were of the type relative lag pairwise, which is considered robust for the assessment of gold grade continuity.

Due to the relatively low number of available composites in the individual MJ veins, their composites were grouped for the variographic analysis. Variograms for the DVCK veins were individually calculated.

The directions of gold grade continuity are in general agreement with the orientation of the mineralized zones, with best directions of continuity trending along strike and dip directions.

Parameters from the variogram models are shown in Table 14-50.

Table 14-50: Modeled Variogram Parameters for Gold Composites at Wedge

Parameters	1 – DVCK-1			2 – DVCK-2			3 – DVCK-3		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	60°	150°	150°	50°	140°	140°	50°	140°	140°
Dip**	0°	-70°	20°	15°	-65°	25°	10°	-70°	20°
Nugget Effect C_0	0.286			0.234			0.193		
1st Structure C_1	1.120			1.072			0.886		
2nd Structure C_2	0.642			0.797			0.825		
1st Range A_1	13.5m	37.1m	13.5m	20.0m	14.6m	10.3m	12.6m	13.7m	8.2m
2nd Range A_2	51.1m	47.9m	15.7m	52.3m	28.6m	15.7m	58.2m	45.2m	18.0m
Parameters	**4- MJ-1**			**5- MJ-2**			**6- MJ-3**		
	Principal	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**
Azimuth*	55°	145°	145°	55°	145°	145°	55°	145°	145°
Dip**	10°	-70°	20°	10°	-70°	20°	10°	-70°	20°
Nugget Effect C_0	0.215			0.215			0.215		
1st Structure C_1	1.152			1.152			1.152		
2nd Structure C_2	1.001			1.001			1.001		
1st Range A_1	22.8m	41.9m	17.5m	22.8m	41.9m	17.5m	22.8m	41.9m	17.5m
2nd Range A_2	55.6m	52.5m	23.9m	55.6m	52.5m	23.9m	55.6m	52.5m	23.9m

*positive clockwise from north
**negative below horizontal
Source: Ginto Consulting Inc. (2019)

14.3.6 Gold Grade Estimation

The estimation of gold grades into a block model was carried out with the ordinary kriging technique on capped composites. The estimation strategy and parameters were tailored to account for the geometrical, geological, and geostatistical characteristics previously identified. The block model's grid definition is presented in Table 14-51. It should be noted that the origin of the block model corresponds to the lower left corner, the point of origin being the exterior edges of the first block. A block size of 3 m (easting) x 3 m (northing) x 3 m (elevation) was selected for the Wedge deposit to better reflect the geometrical configuration and anticipated underground production rate. The Wedge block model was rotated at an angle of 50° clockwise, with the X axis along a 140° azimuth.

Table 14-51: Block Grid Definition – Wedge Deposit

Coordinates	Origin (m)	Rotation (azimuth of X axis)	Distance (m)	Block Size (m)	Number of Blocks
Easting (X)	432,750.0	140°	810.0	3.0	270
Northing (Y)	5,644,440.0		1,263.0	3.0	421
Elevation(Z)	-400.0		900.0	3.0	300
Number of Blocks		34,101,000			

Source: Ginto Consulting Inc. (2019)

The size and orientation of the search ellipsoids for the estimation process was based on the variogram parameters modeled for gold. A minimum of 2 samples and maximum of 12 samples were selected for the block grade calculations. Hard boundaries were assigned in the estimation of each mineralized domain. A summary of the estimation parameters is presented in Table 14-52.

A restrictive search for high yielding composites was utilized to constrain their extrapolation during the estimation process, especially in areas of sparser drilling. An additional constraint, consisting of a 25 m x 25 m x 25 m restrictive search, was applied in one local area of domain DVCK-1, around hole PG18616, and two areas of domain DVCK-2, around holes PG16215 and PG18540. A summary of the constraints is presented in Table 14-53.

Table 14-52: Estimation Parameters for Gold – Wedge Deposit

Estimation Parameters – Gold Grade									
Rock Code	Method	Min # of Samples	Max # of Samples	Search Ellipsoid – Long Axis – Azimuth / Dip	Search Ellipsoid – Long Axis - size	Search Ellipsoid – Short Axis – Azimuth / Dip	Search Ellipsoid – Short Axis - Size	Search Ellipsoid – Vertical Axis – Azimuth / Dip	Search Ellipsoid – Vertical Axis - Size
1	OK	2	12	60°/0°	51.0m	150°/-70°	48.0m	150°/20°	16.0m
2	OK	2	12	50°/15°	52.0m	140°/-65°	29.0m	140°/25°	16.0m
3	OK	2	12	50°/10°	58.0m	140°/-70°	45.0m	140°/20°	18.0m
4	OK	2	12	55°/10°	56.0m	145°/-70°	53.0m	145°/20°	24.0m
5	OK	2	12	55°/10°	56.0m	145°/-70°	53.0m	145°/20°	24.0m
6	OK	2	12	55°/10°	56.0m	145°/-70°	53.0m	145°/20°	24.0m

Source: Ginto Consulting Inc. (2019)

Table 14-53: Restrictive Search for High Yield Gold Composites – Wedge Deposit

Rock Code	Au cut-off (g/t)	Restrictive search distance (m)		
		Principal	Minor	Vertical
1- DVCK-1	≥ 5.0	50.0	50.0	50.0
2- DVCK-2	≥ 3.0	50.0	30.0	30.0
3- DVCK-3	≥ 3.0	50.0	50.0	50.0
4- MJ-1	-	-	-	-
5- MJ-2	≥ 3.0	50.0	50.0	50.0
6- MJ-3	≥ 3.0	50.0	50.0	50.0

Source: Ginto Consulting Inc. (2019)

The grade estimation process consisted of a three-pass approach with the parameters of the first pass as presented in Table 14-53. The estimation parameters of the second and third passes are the same with the exception of an enlarged search ellipsoid by 1.5 times and 3 times the dimensions from the first pass, respectively. In this case, priority was given to estimates from the first pass, followed by estimates from the second pass for un-estimated blocks from the first pass, and finally the estimates of the third pass for un-estimated blocks from the first and second passes. Only blocks within the mineralized wireframes were estimated.

14.3.7 Validation of Grade Estimates

Validation tests were carried out on the estimates to examine the possible presence of a bias and to quantify the level of smoothing / variability.

14.3.7.1 Visual Inspection

A visual inspection of the block estimates with the drill hole grades on plans and cross-sections was performed as a first check of the estimates. Observations from stepping through the estimates along the different planes indicated that there was good agreement between the drill hole grades and the estimates.

The orientations of the estimated grades were also as expected according to the projection angles defined by the search ellipsoid. Higher grade estimates in areas of sparser drilling were constrained in an acceptable way.

14.3.7.2 Global Bias Test

The comparison of the average gold grades from the declustered composites and the estimated block grades examines the possibility of a global bias of the estimates. As a guideline, a difference between the average gold grades of more than ± 10% would indicate a significant over- or under-estimation of the block grades and the possible presence of a bias

Results of the average gold grade comparison are presented in Table 14-54 for the different mineralized domains at Wedge.

Table 14-54: Average Gold Grade Comparison – Polygonal-declustered Composites with Block Estimates

Statistics	Declustered Composites	Block Estimates
Average Gold Grade g/t	3.26	2.06
Difference	-36.8%	

Source: Ginto Consulting Inc. (2019)

As seen in Table 14-54, the average gold grades of the estimates is quite lower than that of the declustered composites. This is most likely due to the restrictive search for the high-grade yielding composites, which constrained the generation of higher grade estimates in areas of sparser drilling. This approach was deemed acceptable at this stage.

14.3.7.3 Local Bias Test

A comparison of the grade from composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scatterplot gives a statistical valuation of the estimates. The estimated block grades should be similar to the composited grades within the block, without being exactly the same value. Thus, a high correlation coefficient will indicate satisfactory results in the interpolation process, while a medium to low correlation coefficient will be indicative of larger differences in the estimates and would require a further review of the interpolation process. Results from the pairing of composited and estimated grades within blocks pierced by a drill hole are presented in Table 14-55 for the Wedge deposit

Table 14-55: Gold Grade Comparison for Blocks Pierced by a Drill Hole – Paired Composite Grades with Block Grade Estimates

Data	Average Gold Grade g/t	Correlation Coefficient
Composites	2.05	0.912
Block Estimates	2.08	

Source: Ginto Consulting Inc. (2019)

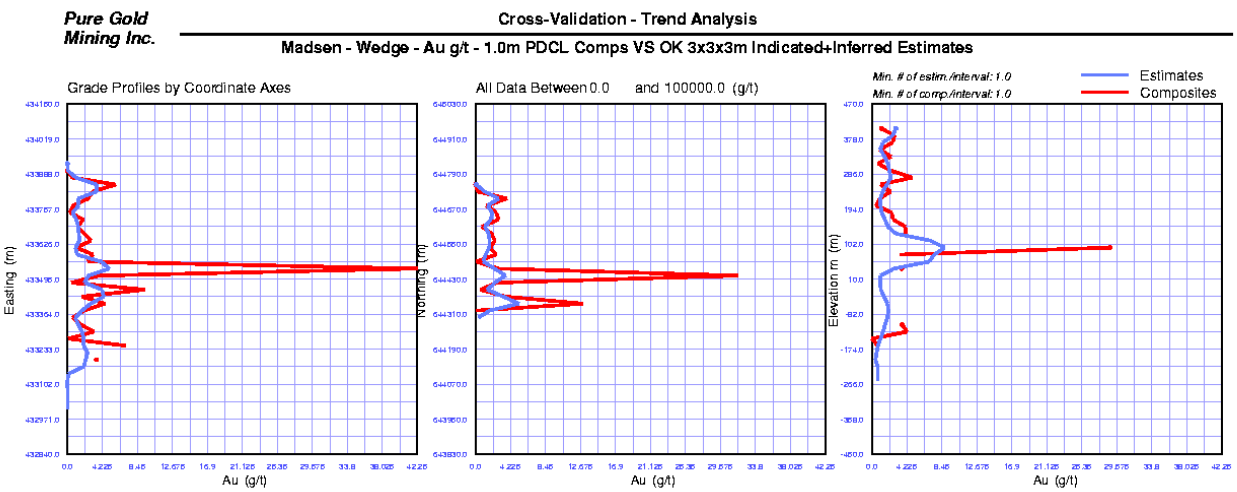
As seen in Table 14-55 for gold, the block grade estimates are very similar to the composite grades within blocks pierced by a drill hole, with a high correlation coefficient, indicating satisfactory results from the estimation process.

14.3.7.4 Grade Profile Reproducibility

The comparison of the grade profiles of the declustered composites with that of the estimates allows for a visual verification of an over- or under-estimation of the block estimates at the global and local scales. A qualitative assessment of the smoothing/variability of the estimates can also be observed from the plots. The output consists of three graphs displaying the average grade according to each of the coordinate axes (east, north, elevation). The ideal result is a grade profile from the estimates that follows that of the declustered composites along the three coordinate axes, in a way that the estimates have lower high-grade peaks than the composites, and higher low-grade peaks than the composites. A smoother grade profile for the estimates, from low to high grade areas, is also anticipated in order to reflect that these grades represent larger volumes than the composites.

Gold grade profiles are presented in Figure 14-56 for the mineralized domains at Wedge.

Figure 14-56: Gold Grade Profiles of Declustered Composites and Block Estimates – Mineralized Domains at Wedge



Source: Ginto Consulting Inc. (2019)

From the plots of Figure 14-56, it can be seen that the grade profiles of the block estimates reproduce adequately those of the declustered composites overall. Consequently, no global or local bias is observed. As anticipated, some smoothing of the block estimates can be seen in the profiles, where estimated grades are higher in lower grade areas and lower in higher grade areas. The constraining effect of the restrictive search on the estimates can also be observed in areas of higher gold grade.

14.3.8 Resource Classification

The mineral resource was classified as indicated and inferred based on the drilling density within each mineralized domain. For such, an outline was drawn around areas of denser drilling for each individual domain. Estimates within these outlines were then classified as indicated, while estimates outside the outlines were classified as inferred. There were no resources classified as measured at Wedge.

14.3.9 Editing of the Block Model

The block model of gold grade estimates for the Wedge deposit was edited to the topographic surface. The percentage of block below the topographic surface was stored in a variable and utilized in the tonnage calculation of each block. The topography variable was utilized to affect the specific gravity in the calculation of the block tonnage: sg x (below topo percentage) / 100.0. No previous mining has taken place at Wedge.

14.3.10 Mineral Resource Calculation

The mineral resource was calculated for 3 m (X) x 3 m (Y) x 3 m (Z) blocks with a constant specific gravity (SG) value of 2.94.

For the mineral resource's tonnage calculation, the fraction of each block within the mineralized zones and the block fraction below the topography surface were accounted for in the tonnage and grade calculations.

The indicated and inferred mineral resources for the Wedge deposit are presented in Table 14-56. The indicated mineral resources of the Wedge deposit at a 4.0 g/t gold cut-off are 322,000 tonnes at an average gold grade of 10.28 g/t, for a total of 107,000 ounces of gold, while the inferred mineral resources at a 4.0 g/t gold cut-off are 307,000 tonnes at an average gold grade of 7.99 g/t, for a total of 79,000 ounces of gold.

The mineral resources of the Madsen, Fork, Russet South, and Wedge deposits are presented in Table 14-57 at 4.0 g/t gold cut-off.

It should be noted that 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 estimated will be converted into mineral reserves. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The CIM definitions were followed for the classification of indicated and inferred mineral resources. The quantity and grade of reported inferred mineral resources in this estimation are uncertain in nature and there has been insufficient exploration to define these inferred mineral resources as an indicated mineral resource and it is uncertain if further exploration will result in upgrading them to an indicated mineral resource category.

All values in Table 14-56 and 14-57 have been rounded to reflect the relative accuracy of the estimates. Mineral resources are reported at a cut-off grade of 4.0 g/t gold based on US$1,200 per troy ounce gold and gold metallurgical recoveries of 92 percent.

Table 14-56: Mineral Resources* by Au Cut-off Grades - Effective February 5, 2019 – Wedge Deposit

Au Cut-Off Grade (g/t)	Indicated			Inferred		
	Tonnage (tonnes)	Au Grade (g/t)	Au Content (ounces)	Tonnage (tonnes)	Au Grade (g/t)	Au Content (ounces)
1.0	1,116,000	4.51	162,000	1,386,000	3.54	158,000
2.0	721,000	6.19	144,000	842,000	4.78	129,000
3.0	481,000	8.05	125,000	599,000	5.72	110,000
4.0	322,000	10.28	107,000	307,000	7.99	79,000
5.0	236,000	12.42	94,000	164,000	11.26	59,000
6.0	158,000	15.87	80,000	93,000	15.59	47,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

Table 14-57: Mineral Resources* at a 4.0 g/t Au Cut-off Grades – Effective February 5, 2019 – Madsen, Fork, Russet South, and Wedge Deposits

Deposit	Indicated			Inferred		
	Tonnage tonnes	Au Grade g/t	Au Content ounces	Tonnage tonnes	Au Grade g/t	Au Content ounces
Madsen	6,429,000	8.99	1,857,000	889,000	8.42	241,000
Fork	203,000	6.62	43,000	331,000	5.78	61,000
Russet South	241,000	7.23	56,000	352,000	7.54	85,000
Wedge	322,000	10.28	107,000	307,000	7.99	79,000

*mineral resources' tonnage and ounces have been rounded to the nearest thousand
Source: Ginto Consulting Inc. (2019)

14.3.11 Discussion and Recommendations

The validation of the gold grade estimates at Wedge has shown satisfactory results overall. The heterogeneous nature of the gold mineralization at Wedge showed higher gold grades in areas of lesser drilling density. This particular situation was dealt with by utilizing a restrictive search for high yield assays in order to limit the over-generation of high-grade block estimates in these regions. This approach was considered appropriate for the treatment of higher gold grades in this case.

Although the drill hole spacing is adequate on average, the thinness of the mineralized zones does not allow for many intercepts from a single drill hole. Therefore, few samples are available for the assessment of gold grade continuity for each individual vein. In this study, the vein samples were grouped for some domains to provide a larger number of samples and allow for a variographic analysis. For such, the deposit would benefit from additional drill holes, especially in areas of isolated higher grades.

Overall, it is believed that the estimation of the mineral resource at Wedge is a realistic representation, based on the current geologic understanding and available information.

15 Mineral Reserve Estimate

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study (PFS). This Feasibility Study includes adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage, and grade that is the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral Reserves are subdivided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

The reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 resource and reserve definitions and Companion Policy 43-101CP. These are listed below.

A “Proven Mineral Reserve” is the economically mineable part of a Measured Mineral Resource demonstrated by at least a PFS. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the Proven Mineral Reserve category implies that the QP has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A “Probable Mineral Reserve” is the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource, demonstrated by at least a PFS. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.1 Cut-off Grade Criteria

Mining reserve values were calculated from the four resource block model tonnes and grades using a gold cut-off grade (COG) to determine the mineable portions of the Madsen deposit. Mineable stopes were defined based on COG values greater than 4.0 g/t Au for the McVeigh zone and 4.75 g/t Au for Austin, South Austin (including A3 domain), and 8 zones after dilution and mining recoveries are applied.

The parameters used for the calculation were based on the data shown in Table 15-1.

Table 15-1: Cut-Off Grade Criteria

Item	Unit	Value
Gold Price	US$/oz	1,275
Exchange Rate	CDN:USD	0.75
Payable Metal	% Au	99
Refining / Transportation	US$/oz	6.00
Total Operating Cost	CDN$/t milled	226
Process Recovery	%	95.0

Note: Assumptions stated in this table were used to establish mining cut-off grades only.
Source: JDS (2019)

15.2 Dilution

Two types of dilution were applied to the stope and development designs:

- External dilution – Additional material (overbreak) that is mined outside of the mineralized zone; this includes backfill dilution from adjacent stopes. All external dilution was assumed as zero Au grade; and
- Design and/or internal dilution – Additional lower than COG material that is within the planned stope or development design shape.

The total external, backfill, planned and inferred dilution is approximately 13% of the total mining reserve.

15.2.1 External Dilution

Dilution for longhole stoping will primarily come in the form of sloughing from the hangingwall and footwall. Rib pillars between adjacent stopes are being utilized which will minimize any potential fill dilution. Hanging wall and footwall dilution for longhole stopes was estimated using the equivalent linear overbreak / slough (ELOS) method described in Section 16.3.6 and is based on the depth of the excavated stope. Table 16-4 shows the linear hanging wall and foot wall dilution applied to stopes. A minimum dilution factor of 5% was also applied to account for any backfill mucking dilution in the longhole stopes. Average external dilution for longhole stopes is 7.9%

For all cut and fill stopes (mechanized and conventional) and ore development a dilution factor of 10% was applied. For the typical cut and fill stope profile of 2 m W x 2.4 m H this is the equivalent of 0.2 m total sloughing / overbreak on the hangingwall and footwall, and 0.24 m of mucking dilution.

15.2.2 Design and Internal Dilution

Additional sources of dilution include Inferred Resource dilution. Any Inferred Resource class material within the mining reserve stope and development shapes has been treated as un-mineralized and has been assigned zero metal grades. Inferred dilution comprises approximately 1.8 kt or 0.05% of the reserve. Non classified material also carries no grade for purposes of reserve estimation, non-classified material accounts for 1.9 kt or 0.05% of the reserve. Dilution from material taken outside the resource wireframes is 114.0 kt or 3.25% of the reserve, 94% of this outside the resource wireframe dilution occurs within the cut and fill stopes. This is due to the areas that require this mining method are generally shallower dipping and may be more challenging geometry.

15.2.3 Mining Recovery

Mining or extraction recovery is a function of mineralized material left behind due to operational constraints typical in the mining process.

The longhole mining method is largely dependent on the accuracy of longhole drilling and explosive detonation to properly fracture the ore. Where holes deviate from the ore limits, some material will remain hung up and may never report to the stope floor for recovery.

Lesser factors considered to affect recoveries of longhole stoping and both cut and fill methods utilized in ore extraction include ragged mucking floors, limited visibility for remote mucking and operator error.

A mining recovery of 95% based on industry norms as well as JDS operational experience in stopes and drifts of similar size and dip has been applied to all mining and ore development methods, with the exception of the 8 Zone. In the 8 Zone some stopes were modelled directly adjacent to historical production stopes. These stopes have a lower 85% mining recovery applied to account for any ore losses that may occur. This accounts for 18.6 kt in the reserve.

15.3 Mineral Reserve Estimate

The mining stope and level designs with dilution and mining recovery factors applied determined the mineral reserve estimate shown in Tables 15-2 and 15–3.

Table 15-2: Madsen Mineral Reserve Estimate

Class	Diluted Tonnes (kt)	Au Grade (g/t)	Au Ounces (koz)
Probable	3,512	8.97	1,013
Total	**3,512**	**8.97**	**1,013**

Notes:

1. The Qualified Person for the Mineral Reserve estimate is Michael Makarenko, P. Eng., of JDS Energy & Mining Inc.
2. Effective date: February 5, 2019. All Mineral Reserves have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under NI 43-101.
3. Mineral Reserves were estimated using a $1,275 /oz gold price and gold cut-off grade of 4.0 g/t for McVeigh Zone and 4.75 g/t for all other zone. Other costs and factors used for gold cut-off grade determination were mining, process and other costs of $226/t, transport and treatment charges of $6.00 /oz Au, no royalty, and a gold metallurgical recovery of 95%.
4. Tonnages are rounded to the nearest 1,000 t, metal grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces.
5. Rounding as required by reporting guidelines may result in summation differences.

Source: JDS (2019)

Table 15-3: Madsen Mineral Reserve Estimate by Zone

Zone	Class	Diluted Tonnes (kt)	Au Grade (g/t)	Au Ounces (koz)
Austin	Probable	1,847	7.85	466
8 Zone	Probable	421	16.87	228
South Austin	Probable	791	8.60	219
McVeigh	Probable	386	6.70	83
A3	Probable	68	7.61	17
Grand Total	**Probable**	**3,512**	**8.97**	**1,013**

Source: JDS (2019)

The Mineral Reserves identified in Tables 15-2 and 15-3 comply with CIM definitions and standards for a NI 43-101 Technical Report. Detailed information on mining, processing, metallurgical, and other relevant factors are contained in the followings sections of this report and demonstrate, at the time of this report, that economic extraction is justified.

This study did not identify any mining, metallurgical, infrastructure or other relevant factors that may materially affect the estimates of the Mineral Reserves or potential production.

16 Mining Methods

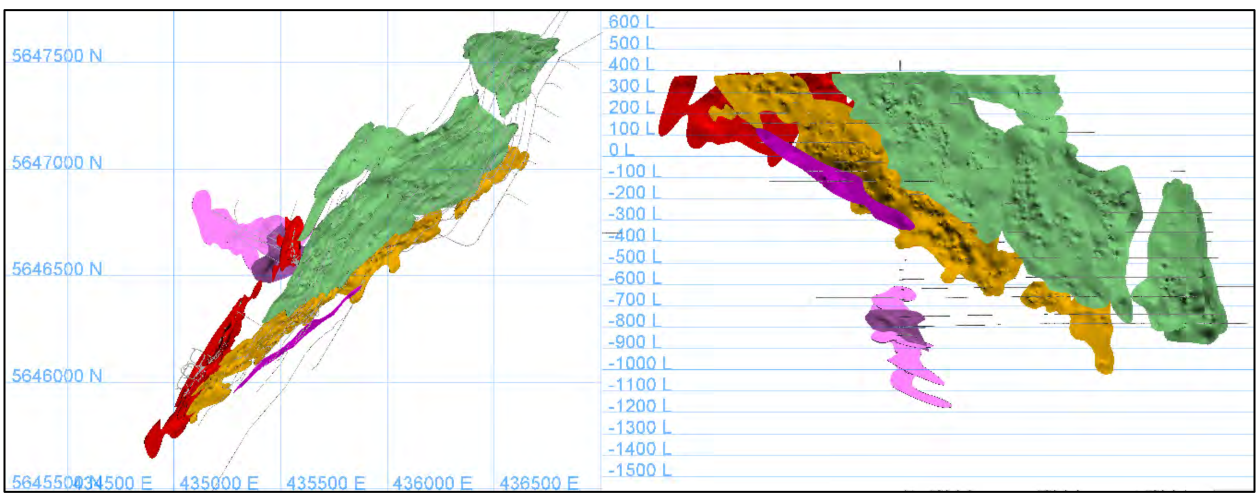
16.1 Introduction

The mine design and planning for Madsen is based on the resource model completed by Ginto Consulting as detailed in Section 14 of this report. Three underground mining methods were selected for ore extraction at Madsen: sub-level longhole stoping (LH); mechanized cut and fill (MCF); and conventional cut and fill (CCF). Mining method was driven primarily by mineralization geometry and continuity, and efficiency of development access. For design purposes, LH was the preferred method of extraction followed by MCF then CCF.

16.2 Deposit Characteristics

High grade mineralization at the Madsen deposit occurs in four identified zones: McVeigh, Austin, South Austin (including the A3 domain), and 8 Zone (Figure 16-1). Within each zone there are multiple parallel veins hosted within a highly altered and weakly mineralized corridor. In the McVeigh, Austin, and South Austin, the veins trend in the NW-SW dipping between 90° to 60° and are generally between 2 m to 10 m thick. In the 8 Zone, the modelled veins dip at around 40° with a lower grade perpendicular structure intersecting the main mineralized zone. The thickness of the veins in the 8 Zone range between 2 m to 8 m true width.

Figure 16-1: Plan View and Long Section of the Modelled High Grade Veins



Source: JDS (2019)

16.3 Geotechnical Analysis and Recommendations

16.3.1 Geotechnical Characterization Data

Minimal previous geotechnical studies or reports are available from the historical mine. As such JDS conducted geotechnical core logging and point load testing of select drill hole intervals as well as geotechnical mapping of underground exposures to develop a sufficient geotechnical database for development of FS geotechnical design parameters. Overall, the following information was used as the basis of the FS geotechnical design:

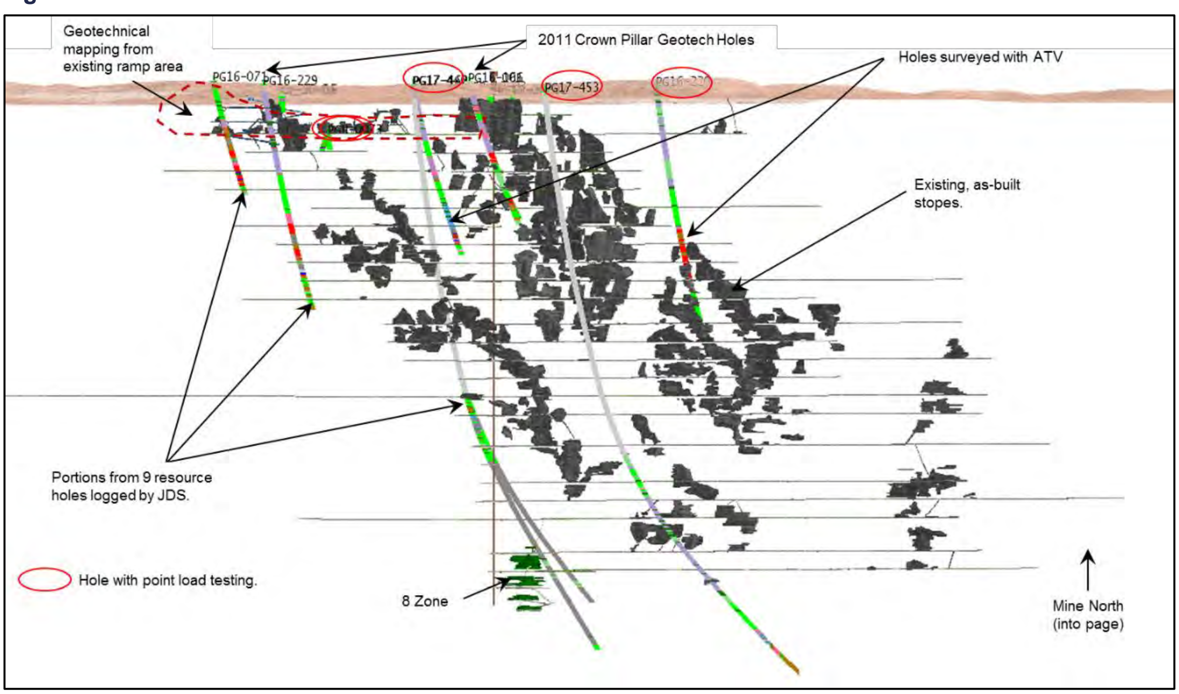
- Rock quality designation (RQD), core recovery and fracture frequency data collected by Pure Gold for 344 resource drill holes located within the FS area. A significant portion of the Pure Gold drill holes are oriented but typically only mineralized veins and lithologic contacts were captured during the resource logging program;
- Geotechnical core logging data from 6 shallow drill holes near the shaft the Stope 2A, 2B and 2-15 crown pillar study carried out by AMEC (2003) and updated by Wood (2011). The core was logged for Barton's Q (Barton & Grimstad, 1994) but fractures were not oriented;
- Geotechnical mapping and discontinuity orientations obtained by JDS from geotechnical mapping of 15 stations along the current ramp system down to Level 2 and along Level 2 out to near the shaft. Additional areas were not accessible at the time of the mapping campaign;
- Geotechnical core logging by JDS of intervals from a total of 9 resource drill holes using the Q' system. Half core was used for 8 holes as the core had previously been split for assay; however, this was possible given the excellent quality and storage of the core. The half core was fully intact with very little disturbance from the core cutting (Figure 16-2). Orientation of fractures on the split core was not possible but was possible for the hole that was logged with whole core;
- Point load tests (PLTs) on ½ core from intervals for three historical resource drill holes and whole core from two recent resource drill holes;
- Laboratory uniaxial strength (UCS) and Brazilian indirect tensile strength (BTS) testing on samples of whole core obtained from two current resource drill holes;
- Discontinuity orientations and information obtained from optical (OTV) and acoustical televiewer (ATV) surveys of three resource drill holes carried out by DGI Geoscience Inc.(2018); and
- Anecdotal information regarding ground support and ground conditions reported by previous mine workers.

Figure 16-3 shows the existing as-built development and stoping areas with the distribution of the various geotechnical data sources described above.

Figure 16-2: Example of Typical Half Core used for Logging and Point Load Testing

Source: JDS (2019)

Figure 16-3: Distribution of Geotechnical Data Sources



Source: JDS (2019)

16.3.2 Geotechnical Domains and Rock Mass Properties

Based on the geologic structural and lithology models and the geotechnical characterization data described above, the deposit was divided into separate geotechnical domains. Each of the domains grouped volumes where overall rock mass quality and ground conditions are anticipated to be similar. Geotechnical design parameters were then developed for each of the domains. Lithology was identified as the dominant factor controlling the geotechnical domains for the main vein systems while the relationship to the mineralized structural corridor (i.e. hanging wall/footwall) appears to control rock quality at 8 Zone. There have been very few brittle fault structures encountered at Madsen due to its location within the Canadian Shield and as such, major fault structures did not play a significant role in the selection of geotechnical domains.

Table 16-1 contains a summary of the key rock mass properties by rock type derived from the core logging data while Table 16-2 summarizes the mean UCS values by rock type. The data in Table 16-1 represents the mean value of all the core runs drilled within the respective domains or lithology. Local variations will occur from the values listed in Table 16-1 and Table 16-2 but the mean values summarized are expected to be indicative of the overall rock mass behavior.

Geotechnical characteristics of the primary rock types are described below. Additional details regarding geology and structural geology are contained in Section 7.

16.3.2.1 Madsen Deposit: Austin, South Austin, and McVeigh Zones

Balmer Assemblage Basalts (BSLA & BSLT):

Basalt is the most common lithology in the Balmer Assemblage forming the hanging wall and footwall of the main mineralized veins systems. The basalts are typically a competent, slightly blocky to massive rock mass with strong to very strong intact rock strengths. The basalts have a slight vein-parallel foliation similar to, but much less intense than, the SAFZ. While geologically different, both the altered basalt (BSLA) and non-altered basalt (BSLT) demonstrate similar geotechnical characteristics. Analysis of the core logging and mapping data indicate average Q' values of 15.4 and 15.1, respectively, corresponding to 'Good' rock quality according to Barton's Q system (Barton & Grimstad, 1994). UCS test values are typically in the 100 to 150 MPa range or Very Strong based on the ISRM (1981) suggested methods.

Strongly Altered Foliated Zones (SAFZ):

The gold mineralization is contained within SAFZ zones that represent structural corridors exploited by alteration and subsequently gold bearing fluids. The SAFZ zones are coherent domains that are altered and foliated to a degree that the protolith is unrecognizable. Although strongly foliated, the SAFZ has a high silica content and is typically massive with few fractures or joints of any persistence. Foliation does not appear to form a dominant plane of weakness in the SAFZ, based on observations of existing exposures underground.

Analysis of the core logging and mapping data indicate average Q' values of 14.6 and 15.3, respectively, corresponding to 'Good' rock quality according to Barton's Q system (Barton & Grimstad, 1994). The intact rock typically has UCS values between 100 and 200 MPa range which is described as Very Strong according to the ISRM (1981) suggested methods.

Some historical open stopes are visible from the ramp that have remained open since 1999. Where the inside of stopes could be safely viewed, they appeared stable with only minor, localized sloughing apparent.

Intrusive Dykes (MINT & IINT):

Mafic intrusive dykes (MINT) and intermediate intrusive dykes (IINT) are common around the main vein systems. The dykes have sharp, typically chilled, margins and post-date mineralization. They typically are well jointed and blocky near contacts with the host rock. They are commonly oriented sub-parallel to foliation and cross-cut the Balmer Assemblage. Development through the dykes can result in over-break or over-scaling where they strike oblique to drifts. The intrusive dykes are too numerous and complex to accurately model in 3D or predict their occurrence. An average Q' value of 13.3 was calculated from the core logging data ('Good' rock quality) while drift mapping of MINT exposures indicated a lower mean Q' value of 8.1 classifying as 'Fair' rock quality according to Barton's Q system (Barton & Grimstad, 1994). UCS test values are typically in the 100 to 200 MPa range or Very Strong based on the ISRM (1981) suggested methods.

Ultramafic Sills (UMAF):

Five intrusive ultramafic sills (UMAF) have been delineated in the Madsen Mine area (Venus, Pluto, Neptune, Mercury and Jupiter) but only the Venus and a small portion of the Mercury sill are near the deposit. The Jupiter and Neptune sills are crossed briefly by access tunnels to the 8 Zone. The UMAF sills consist of peridotite, pyroxenite, and gabbro and can be well foliated. The sills are generally oriented parallel to foliation and the host basalt.

The UMAF sills represent the poorest rock quality at Madsen. Rock strength is typically strong to very strong based on PLTs and lab testing but joint conditions can be quite poor or weak. The sills can have intense veining and microdefects with foliation representing tightly spaced weak planes in the rock mass. Talc alteration of the sills can be very intense near contacts. Borehole deformation occurred in zones where two of the ATV holes crossed through the Venus UMAF. An average Q' of 4.6 resulted from the core logging data and an average Q' of 5.9 was estimated based on the mapping. Both sources would be classified as 'Fair' rock quality according to Barton's Q system (Barton & Grimstad, 1994). UCS test values are typically in the 100 to 200 MPa range or Very Strong based on the ISRM (1981) suggested methods.

Point load testing indicates an average UCS value of 159 MPa for passing tests; however, 13 of the 29 tests (45%) were considered failing tests because they broke preferentially across foliation rather than between the platens. The 13 foliation breaks averaged only 34 MPa, 21% of the passing test average strength (Table 16-2). This indicates that the UMAFs have potential for high strength anisotropy and frequent foliation fractures should be anticipated. Given the steep dip and location of the sills outside the mineralization, the most significant impact of the foliation would be on footwall development where drifts are within the Venus and Mercury sills and parallel or are slightly oblique to the foliation direction. The walls of these drifts may require additional ground support which has been accounted for in ground support estimates discussed in Section 16.3.8.

The 2 Level North cross cut crosses the Venus UMAF sill at an angle approximately perpendicular to the strike of the sill. This drift connects the McVeigh and South Austin zones and provides a good cross section exposure of the Venus ultramafic in contact with BSLA and SAFZ. Although the cross cut dimensions are relatively small (approximately 2.4 m x 2.4 m), the drift appears to have performed well since it was driven in the 1930's with no obvious signs of deformation around the UMAF contacts, even where talcose, and no ground support installed.

Conclusions from Geotechnical Domains of Austin, South Austin, and McVeigh Zones:

Austin, South Austin, McVeigh zone mining and development will occur primarily in the basalts and SAFZ units. While geologically different, both units are similar in overall rock mass quality and strength and have been grouped together into a single geotechnical domain for analysis. The domain is very competent and of 'Good' geo-mechanical quality with Q' typically between 10 and 17 according to the Barton classification system (Barton & Grimstad, 1994). Intact rock strength is classified as Strong to Very Strong (ISRM, 1981) with typical UCS values between 100 and greater than 200 MPa.

Although the intrusive dykes are slightly lower in rock quality, they have also been included in the basalt / SAFZ domain since they represent a comparably small percentage of the overall rock mass and that their location cannot reliably be predicted at this stage of the project. The dykes typically have slightly lower Q' values between 8 and 16 ('Fair' to 'Good' rock quality) due primarily to increased fracturing and blockiness.

The ultramafic sills represent the lowest rock quality at Madsen and are considered a second domain. The sills are typically 'Poor' to 'Fair' geo-mechanical quality (Q' between 3 and 6) and can have weak intact strengths where intensely altered. The ultramafics also tend to have more frequent foliation joints and can have particularly weak joint conditions being very smooth with various levels of talc, chlorite and/or biotite.

16.3.2.2 8 Zone:

The 8 Zone mineralization occurs within strongly altered and veined peridotite of the Russet Lake Ultramafic which is believed to be of extrusive origin. Gold mineralization typically occurs within biotite-amphibole altered peridotite (PRBA) which is moderately to strongly altered and foliated in comparison to the surrounding peridotite. These zones of alteration and foliation manifest along structural corridors that were also exploited by the gold bearing fluids. Talc alteration can be very intense particularly near contacts, significantly weakening the rock mass.

Relatively few holes have been drilled through the 8 Zone by Pure Gold due to the significant depth of the zone, and as such, significantly less geotechnical information exists for the 8 Zone than exists for the other deposits. In addition to the drill holes, JDS has also used historical information available from Pure Gold regarding ground conditions encountered when accessing and mining the 8 Zone as part of the characterization process. The 8 Zone is currently flooded and therefore was not accessible for any underground mapping as part of the FS. No information regarding joint orientations is available for the 8 Zone; however, the worst case scenario that the dominant rock mass structure is parallel to the HW has been assumed for the analyses.

Gold mineralization at 8 Zone is generally contained within three parallel structures as shown in Figure 16-4. The rock quality varies significantly between the hanging wall, footwall and mineralized structures and as such, each was considered a different geotechnical domain for analysis. A description of each of the 3 geotechnical domains follows.

8 Zone Hanging Wall Geotechnical Domain:

Based on the available data the overall hanging wall above the top of the 3 high grade vein domains is of variable quality ranging from 'Fair' to 'Good' geo-mechanical quality with Q' values typically ranging between 8 and 17. Estimates of UCS were made for one drill hole by correlating PLTs with a correlation factor developed using laboratory UCS test results. The correlated UCS data indicate a lower average UCS in the immediate hanging wall (first 10 m above the top vein) with the rock strength increasing significantly beyond. The correlated average UCS for the immediate hanging wall was 79 MPa while the UCS beyond 10 m averaged 196 MPa.

8 Zone Mineralized Zone Geotechnical Domain:

The quality of the rock mass within and between the 3 high grade vein domains is generally 'Poor' to 'Fair' geo-mechanical quality with Q' ranging between 4 and 8 based on the core logging data. Intense talc alteration and foliation is common within this zone.

Point load testing indicated an average UCS value of 83 MPa for passing tests; however, 17 of the 19 tests were considered failing tests because they broke preferentially across foliation averaging only 20 MPa (Table 16-2). This indicates that the rock within the mineralized zone is highly anisotropic and frequent foliation fractures should be anticipated. Given the shallow, approximately 45° dip, the foliation will limit any potential span of unsupported ground.

As illustrated in Figure 16-4, some portions of the veins do not contain economic mineralization and therefore will not be mined. As a result, the 'hanging wall' of stopes mined in the middle of the three veins will consist of rock within the mineralized zone domain and will likely be of significantly lower rock quality compared to stopes mined in the upper most vein. As such, the weaker mineralized domain strength was assumed for the hanging wall for design of all 8 Zone stopes.

Previous reports indicate that the historical mining encountered ground control challenges when accessing the 8 Zone ore body from the hanging wall due to what was described as a talc schist encountered near the hanging wall of the ore body. Mining conditions were reportedly improved significantly by changing access to the ore body from the footwall side.

8 Zone Footwall Geotechnical Domain:

The footwall appears to be the highest rock quality at the 8 Zone deposit. The footwall is typically of 'Good' geo-mechanical quality with Q' ranging between 16 and 21 and an average UCS of 91 MPa. Historic reports indicated that ground control issues were avoided by accessing the 8 Zone ore from the footwall.

Table 16-1: Rock Mass Properties from Geotechnical Core Logging Data by Rock Types and Vein System

Domain	No. of Runs	Mean Rec. (%)	Mean RQD (%)	Mean FF/m	Q' (Barton & Grimstad, 1994) [A,B]					RMR_{76} (Bieniawski, 1976) [C]		
					Min.	20^{th} %	Mean	80^{th} %	Max.	Min.	Mean	Max.
Basalts	164	100	98	1.7	1.9	10.9	15.4	17	33.9	50	68	76
Intrusive Dikes	14	99	95	2.8	5.2	8.0	13.3	16.4	20.5	59	67	71
Russet Lake UMAF	190	100	96	2	2.8	10.4	14.5	16.7	32.9	53	67	75
Venus UMAF	35	100	88	3.7	2.4	3.3	4.6	5.6	7.4	52	57	62
Veins	6	100	99	1.5	10.5	10.7	13.6	16.5	16.5	65	67	69
Confederation	26	100	95	2.2	3.4	8.4	13.7	16.3	33	55	67	76
Austin	45	100	96	3.5	2.8	5.5	11.5	16.5	35	53	64	76
McVeigh	98	100	98	2.2	1.9	10.8	14.4	16.7	33.9	50	67	76
South Austin	16	100	95	2.2	6.7	9.8	18.1	28.9	33.9	61	69	76
8 Zone HW	32	100	98	1.6	7.5	8.3	14	16.6	17.1	62	63	63
8 Zone Mineralized Zone	17	97	87	3.6	2.8	4.8	6.7	8.4	12.2	53	61	67
8 Zone HG FW	34	100	95	1.9	10.9	15.6	18.3	21.3	32.9	66	70	75

[A] 3 joint sets (Jn = 9) estimated for Q' equation.
[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 x ln Q + 44).

Source: JDS (2019)

Table 16-2: Mean PLT/UCS Values per Rock Type and Domains

Lithology Solid	No. Tests	T1 (Valid Test) UCS (MPa)[A]	T3 (failed on pre-existing defect) UCS (MPa) [A]	T1 & T3 Combined UCS (MPa) [A]	T3:T1 [B] (Strength)	% T3 Tests [C]
Balmer 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%
8 Zone Distant HW (>10 m)	21	196	93	166	47%	29%
8 Zone Immediate HW (<10 m)	8	79	41	65	52%	38%
8 Zone Mineralized Zone	19	83	20	26	24%	91%
8 Zone Footwall	36	91	34	51	37%	69%

[A] UCS calculated by converting PLTs using a correlation factor of 14.6.

[B] Ratio of the average T3 (broke on foliation) UCS strength to the average T1 (valid test, broke perpendicular to foliation) strength. This provides a general indication as to how much weaker the rock may be in the foliation direction.

[C] Percent of overall number of tests that were T3 breaks (broke on foliation). This provides a general indication as to how frequent the weak foliation planes occur.

Source: JDS (2019)

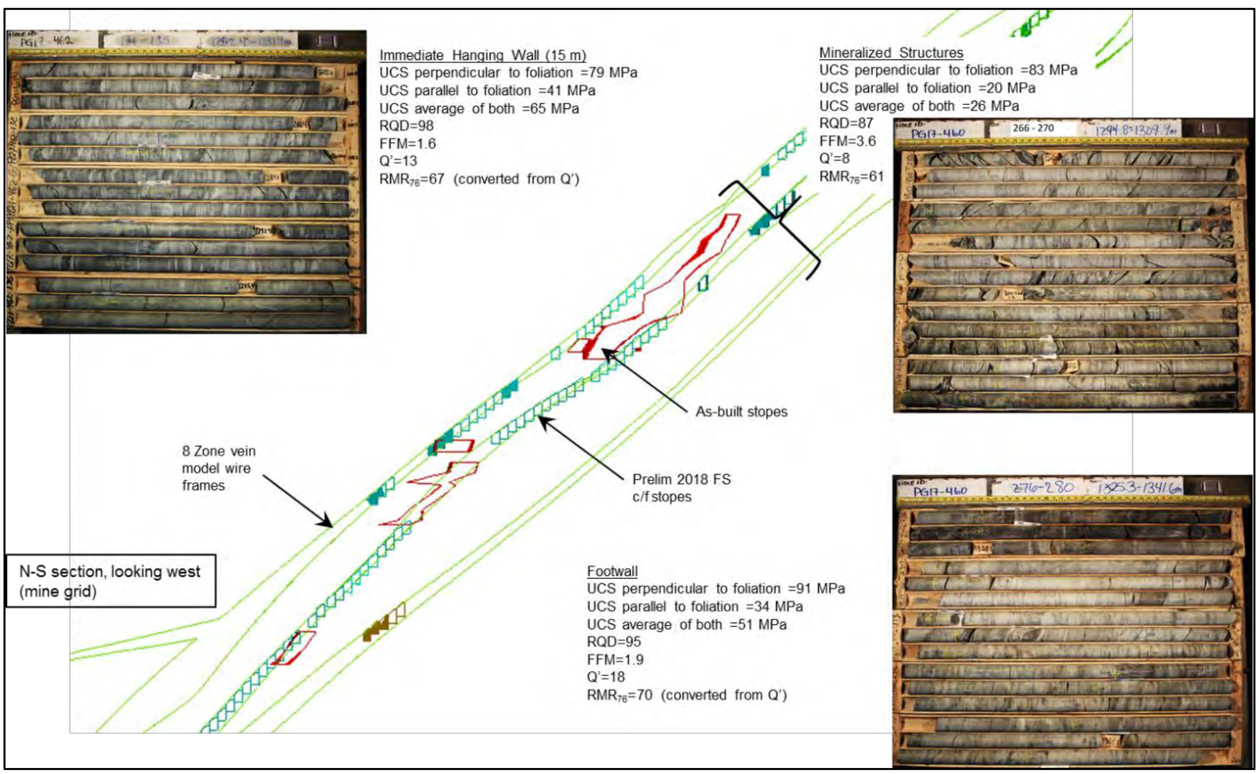
Rock quality parameter Q' obtained from the geotechnical core logging is compared with data obtained from underground geotechnical mapping in Table 16-3 as a means of confirming data consistency and quality. The mean values for the basalts, the SAFZ and the UMAF sills show excellent correlation between the two data sources while the intrusive dykes appear slightly lower quality from the underground mapping. Although the mapping and core logging for the MINT still compare reasonably well, the mapping may return slightly lower values than what was observed in core logging due to blasting effects in the excavation.

Table 16-3: Comparison of Geotechnical Mapping and Core Logging Data

Lithology	Q' (Mapping)				Q' (Core Logging)			
	No. of Cells	Min.	Mean	Max.	No. of Runs	Min.	Mean	Max.
Basalts	5	7.9	**15.1**	30	164	1.9	**15.4**	33.9
SAFZ	4	8.3	**15.3**	21.3	44	3.4	**14.6**	34.6
UMAF	3	5	**5.9**	7.5	35	2.4	**4.6**	7.4
MINT	3	7.9	**8.1**	8.3	14	5.2	**13.3**	20.5

Source: JDS (2019)

Figure 16-4: 8 Zone Geotechnical Domains



Source: JDS (2019)

16.3.3 Discontinuity Sets

Discontinuity orientations and conditions have been characterized by ATV surveys of three drill holes as well as underground geotechnical mapping. The locations of the drill holes and mapping area are shown on Figure 16-3. A significant portion of the core from Pure Gold drill holes were also oriented using the Reflex Act III core orientation tool but typically only local mineralized veins were measured during the resource logging program. Review and comparison of the ATV logs, mapping data, and known geologic trends in the deposit indicates the data is consistent and of good quality. Discontinuity sets appear similarly oriented across the main vein systems. The following conclusions were made from the overall data set:

- Foliation structures (J1) have a steep SE dip and are the most dominant pattern and are parallel to mineralized vein structures;
- A flat to shallow northwest dipping set (J2) is common throughout mine. This set has potential to leave unstable blocks in development backs but will be mitigated with the ground support recommended in Section 16.3.8;
- A steep northeast dipping set (J3) exists and has potential to form rectangular blocks with foliation structures (J1) and the flat set (J2);
- One or two additional sets across the mine are much less dominant and exhibit a higher variability in their orientation; and
- Based on review of existing excavations and joint data, large, unstable wedges or blocks is not anticipated. This is due to the joint orientations described above as well as the overall high rock quality, low joint persistence and anticipated clamping stresses.

16.3.4 Stress Regime and Numerical Modeling

16.3.4.1 Stress Regime

Stress conditions for the mine have been estimated based on regional geologic information and site topographic features as well as the World Stress Map (Heidbach, et al, 2016). The World Stress Map database contains two measurements of in-situ stresses obtained nearby: one at the Madsen Mine and the other approximately 150 km to the southwest at the Atomic Energy of Canada Limited (AECL) Underground Research Laboratory (URL). Both indicate that the major horizontal stress (σ_H) is oriented northeast – southwest, sub-parallel to the vein strike, at least regionally. Stresses may have rotated locally around the mine openings.

Based on the measured stress magnitudes and test depths, the magnitude of the major horizontal stress (σ_H) is approximately two times the vertical stress (σ_v) and the minor horizontal stress (σ_h) is approximately 1.3 times the vertical stress (σ_v).

No observations were been made while underground indicating high horizontal stress conditions; however only the upper two levels were accessible. Based on discussions with previous miners, the mine did not have significant issues with stresses in the main vein systems but access tunnels to the 8 Zone from the shaft did experience high rates of closure in local areas. These areas are believed to correlate with certain ultramafic sills as well as the contact of the Russet Lake UMAF with the stiffer Balmer Basalt. Historical maps showing these locations and the ground support types were reviewed as part of the FS geotechnical assessment.

The anticipated range of stresses has been considered in the calculation of stress parameter A for stope design as discussed below. Horizontal stresses may need to be assessed in more detail during operation.

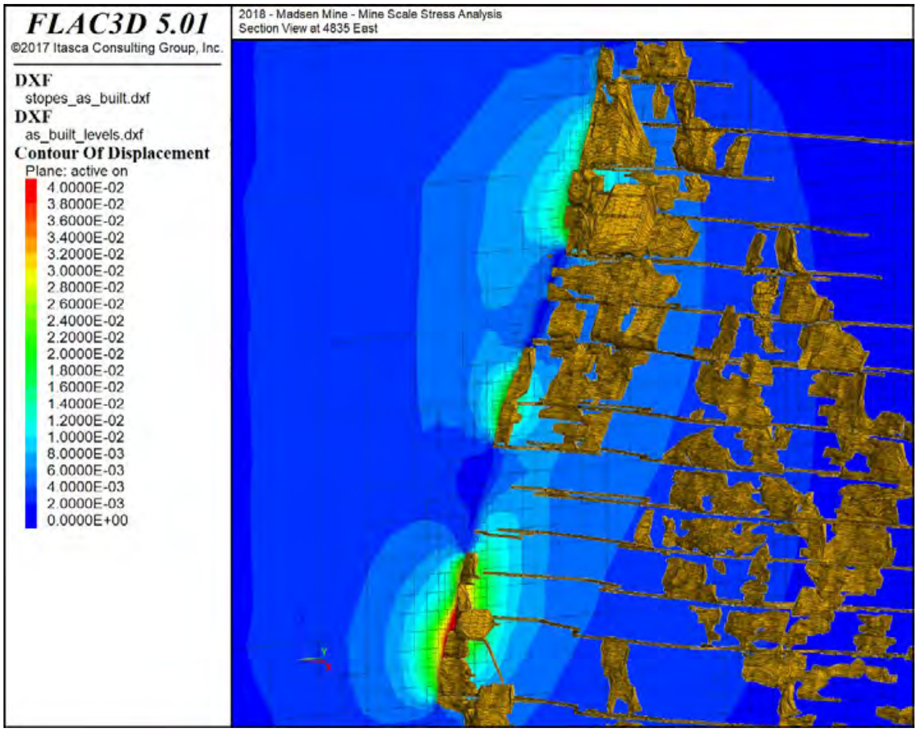
16.3.4.2 Mine-Scale Numerical Stress Model

Given the potential for elevated horizontal stresses, a mine-scale 3D numerical stress model was developed by SRK Consulting (U.S.) Inc. for the main vein systems (Austin, South Austin, McVeigh and A3 deposits). The purpose of the model was to estimate stress distributions around historical stoping areas and to better understand where difficult ground conditions may be encountered. An example of the completed model showing total displacement on a north-south (4835 E) cross section is shown on Figure 16-5.

The model used geotechnical properties and material models that were based on the descriptions above. The properties were calibrated as close as possible to the reported squeezing conditions that occurred in the access tunnels to the 8 Zone and the borehole breakouts observed in the 2 holes recently surveyed with the acoustic televiewer. Additional details regarding material properties and model details can be found in (SRK, 2018A). The conclusions from the numerical model are summarized below:

- Displacements around the historical mining areas were relatively small and the depth of rock mass yielding into stope walls was limited due to the high competency of the basalt and SAFZ rock mass;
- Stress concentrations remain relatively close to existing stope walls;
- Maximum principal stress (σ_1) concentrations near existing stopes range between approximately 20 MPa in the upper levels and 80 MPa in the lower levels;
- Rock mass experiences high stress near stopes but generally remains intact (zones in model do not show yielding) where pillars between stopes are 20 m or more;
- New development and stopes less than 20 m from existing stopes are not anticipated to encounter unmanageable ground conditions given the good rock quality but may require destress blasting; and
- Concentrated stresses may be locked-in in remnant pillars left by past mining and may require de-stressing blasting ahead of mining.

Figure 16-5: View of Displacements around Existing Stopes from 3D Numerical Model



Source: JDS (2019)

16.3.5 Empirical Stope Design Analysis

Empirical stope design analyses are based on stability graphs where the stability number (N') is plotted on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope face on the horizontal axis. The stability number is calculated based on the rock mass quality (Q'), and three empirical factors: A (induced stress conditions), B (geologic structure orientation) and C (dip angle of the stope face).

Maximum stope dimensions were estimated using the Potvin (2001) method for the anticipated range of rock mass conditions and stope sizes. The Trueman & Mawdesley (2003) 'Stable' line was then used as a check against the Potvin (2001) results. Upper and lower bound estimates of stope face dimensions and rock quality (Q') were analyzed for each of the main vein systems and 8 Zone geotechnical domains.

Empirical factors for the calculation of stability number (N') were based on the following assumptions:

- Induced Stress Factor, A, varied from 0.8 for shallow stopes to 0.2 for deep stopes where horizontal stresses may be high. The A parameter was calculated based on sigma 1 near stopes in the mine-scale numerical model and the average UCS strength for each domain;
- Joint Orientation Factor, B, varied from 0.2 to 0.3 for the hanging wall and footwall, and was calculated based on the dominant discontinuity orientation being sub-parallel to veins; and

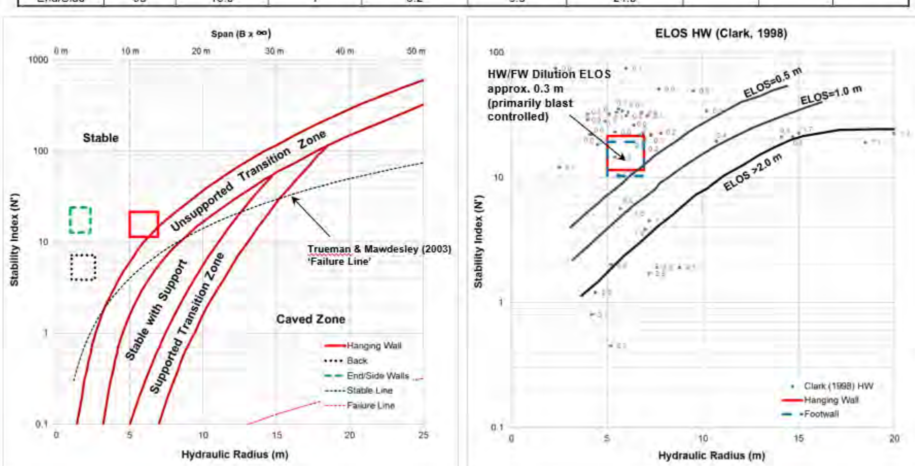
- Gravity Factor, C was calculated for each stope wall, except for the footwall, based on dip angle using the following equation: C =8 – 6 cos (face dip angle). The maximum C value of 8.0 was assumed for the footwall due to its favorable dip orientation.

The maximum level spacing was set at 20 m and the maximum stable lengths and widths were then determined from the stability graph for the hanging walls and backs, respectively. Figures 16-6, 16-7 and 16-8 contain output plots from the empirical stability analyses for the main vein systems for depths of 300 m, 600 m and 900 m, respectively. A plot for the 8 Zone cut and fill mining is shown in Figure 16-9. Based on the results of the empirical stope stability analyses, the following conclusions were made:

- For longitudinal mining, stopes can be up to 45 m long at 300 m depth, 30 m long at 600 m depth and 15 m long at 900 m depth based on the 20 m stope height;
- Cable bolts will not be required for stope backs for typical stope widths (approx. 2 m to 6 m); and
- Cut and fill stopes can be mined 2.4 m high by up to 6 m wide and 200 m long with pattern bolting.

Figure 16-6: Empirical Stope Stability Analyses and Dilution Estimate for 300 m (Longitudinal LHOS)

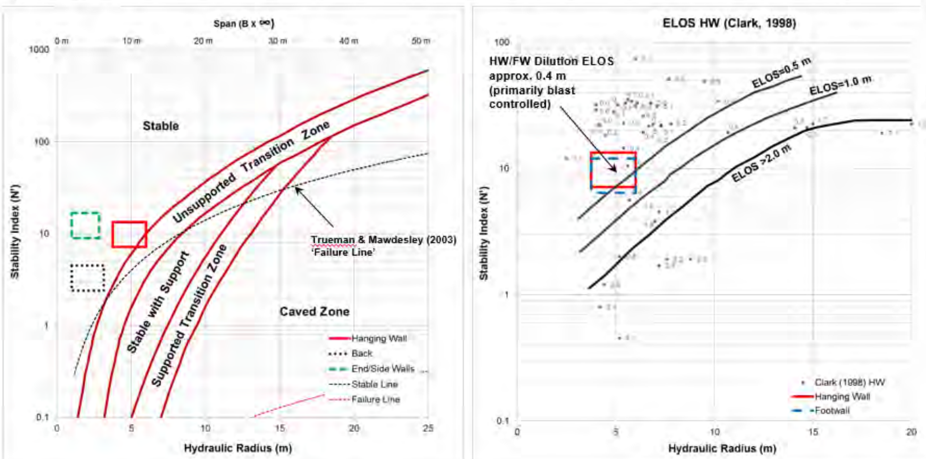
Wall	Dip	Q'	Empirical Parameters			N'	Max. and Min. Ranges		
			A (Stress)	B (Joint Orientation)	C (Dip Angle)		Width	Height	Length
HW	70	8.0	0.8	0.3	5.9	11.4	6.0	20.0	45.0
FW	70	8.0	0.8	0.2	8.0	10.2			
Back	0	8.0	0.8	0.3	2.0	3.8			
End/Side	90	8.0	1	0.2	8.0	12.8			
HW	70	15.0	0.8	0.3	5.9	21.4	2.3	15.0	30.0
FW	70	15.0	0.8	0.2	8.0	19.2			
Back	0	15.0	0.8	0.3	2.0	7.2			
End/Side	90	15.0	1	0.2	8.0	24.0			



Source: JDS (2019)

Figure 16-7: Empirical Stope Stability Analyses and Dilution Estimate for 600 m (Longitudinal LHOS)

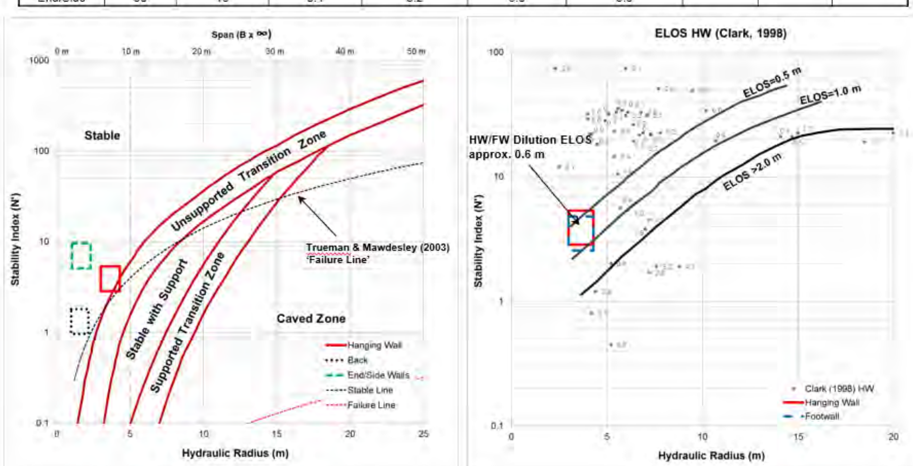
Wall	Dip	Q'	Empirical Parameters			N'	Max. and Min. Ranges		
			A (Stress)	B (Joint Orientation)	C (Dip Angle)		Width	Height	Length
HW	70	8	0.5	0.3	5.9	7.1	8.0	20.0	30.0
FW	70	8	0.5	0.2	8.0	6.4			
Back	0	8	0.5	0.3	2.0	2.4			
End/Side	90	8	0.7	0.2	8.0	9.0			
HW	70	15	0.5	0.3	5.9	13.4	2.3	15.0	15.0
FW	70	15	0.5	0.2	8.0	12.0			
Back	0	15	0.5	0.3	2.0	4.5			
End/Side	90	15	0.7	0.2	8.0	16.8			



Source: JDS (2019)

Figure 16-8: Empirical Stope Stability Analyses and Dilution Estimate for 900 m (Longitudinal LHOS)

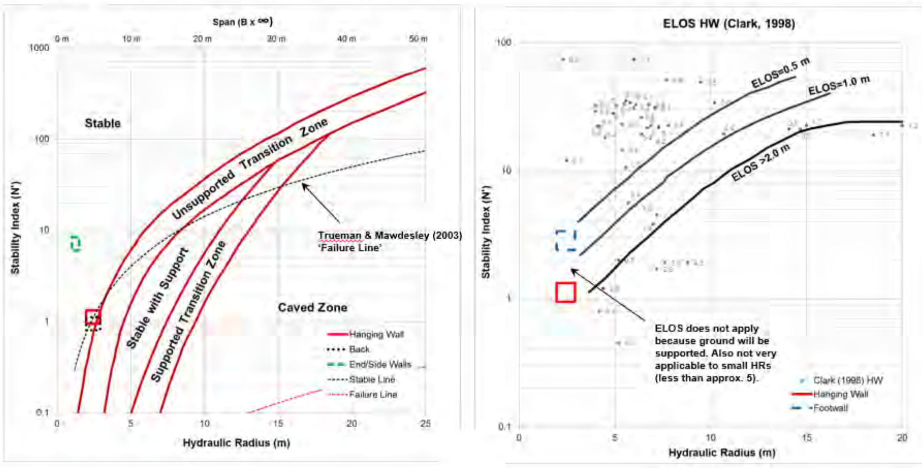
Wall	Dip	Q'	Empirical Parameters			N'	Max. and Min. Ranges		
			A (Stress)	B (Joint Orientation)	C (Dip Angle)		Width	Height	Length
HW	70	8	0.2	0.3	5.9	2.9	6.0	20.0	15.0
FW	70	8	0.2	0.2	8.0	2.6			
Back	0	8	0.2	0.3	2.0	1.0			
End/Side	90	8	0.4	0.2	8.0	5.1			
HW	70	15	0.2	0.3	5.9	5.4	2.4	15.0	10.0
FW	70	15	0.2	0.2	8.0	4.8			
Back	0	15	0.2	0.3	2.0	1.8			
End/Side	90	15	0.4	0.2	8.0	9.6			



Source: JDS (2019)

Figure 16-9: Empirical Stope Stability Analyses and Dilution Estimate for 1200 m (Cut and Fill)

Wall	Dip	Q'	Empirical Parameters			N'	Max. and Min. Ranges		
			A (Stress)	B (Joint Orientation)	C (Dip Angle)		Width	Height	Length
HW	36	5	0.2	0.3	3.1	0.9	6.0	6.0	200
FW	90	5	0.2	0.3	8.0	2.4			
Back	0	5	0.2	0.4	2.0	0.8			
End/Side	90	5	0.3	0.5	8.0	6.0			
HW	36	7	0.2	0.3	3.1	1.3	4.0	4.0	200
FW	90	7	0.2	0.3	8.0	3.4			
Back	0	7	0.2	0.4	2.0	1.1			
End/Side	90	7	0.3	0.5	8.0	8.4			



Source: JDS (2019)

16.3.6 Estimates of Unplanned Dilution

The potential for unplanned dilution was estimated for stope hanging walls and footwalls using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1998). The method is similar to the empirical stope stability charts previously discussed with Stability Number (N') plotted on the vertical axis and the hydraulic radius (wall area divided by wall perimeter) of the stope face on the horizontal axis but rather than showing stable/unstable conditions, the plot contains contours of unplanned dilution. The estimates are approximated in terms of the average thickness spread over the entire hanging wall or footwall.

The ELOS graph is not applicable to small hydraulic radii less than 4 or 5 m. For small hydraulic radii such as these, blasting quality will be the primary control on dilution. Unplanned dilution for the 8 Zone is anticipated to be controlled primarily by blasting practices and as such, has been estimated based on planned blasting and excavation practices for the deposit.

Unplanned dilution in the upper portion of the main vein systems is also anticipated to be mostly blast controlled due to the lower anticipated stress and good rock mass quality. The lowest portion of the main vein systems may experience slightly higher dilution due to high horizontal stresses. An unplanned dilution of 0.6 m was estimated for stope hanging walls and footwalls for the 900 m depth based on the ELOS method.

Figures 16-6, 16-7 and 16-8 contain ELOS plots for the main vein systems at depths of 300 m, 600 m and 900 m, respectively. An ELOS plot for the 8 Zone cut and fill mining is shown in Figure 16-9; however, ELOS is not applicable to cut and fill mining as the ground will be supported. Additional details regarding dilution assumptions for cut and fill mining are contained in Section 15.2.1.

Table 16-4: Estimates of Unplanned Dilution for Long Hole Stopes by Depth

Mining Type and Deposit	Stope Wall	Depth Below Ground Surface (m)		
		300 m	600 m	900 m
LHOS (Main Vein Systems)	Hanging Wall	0.3	0.4	0.6
	Footwall	0.3	0.4	0.6

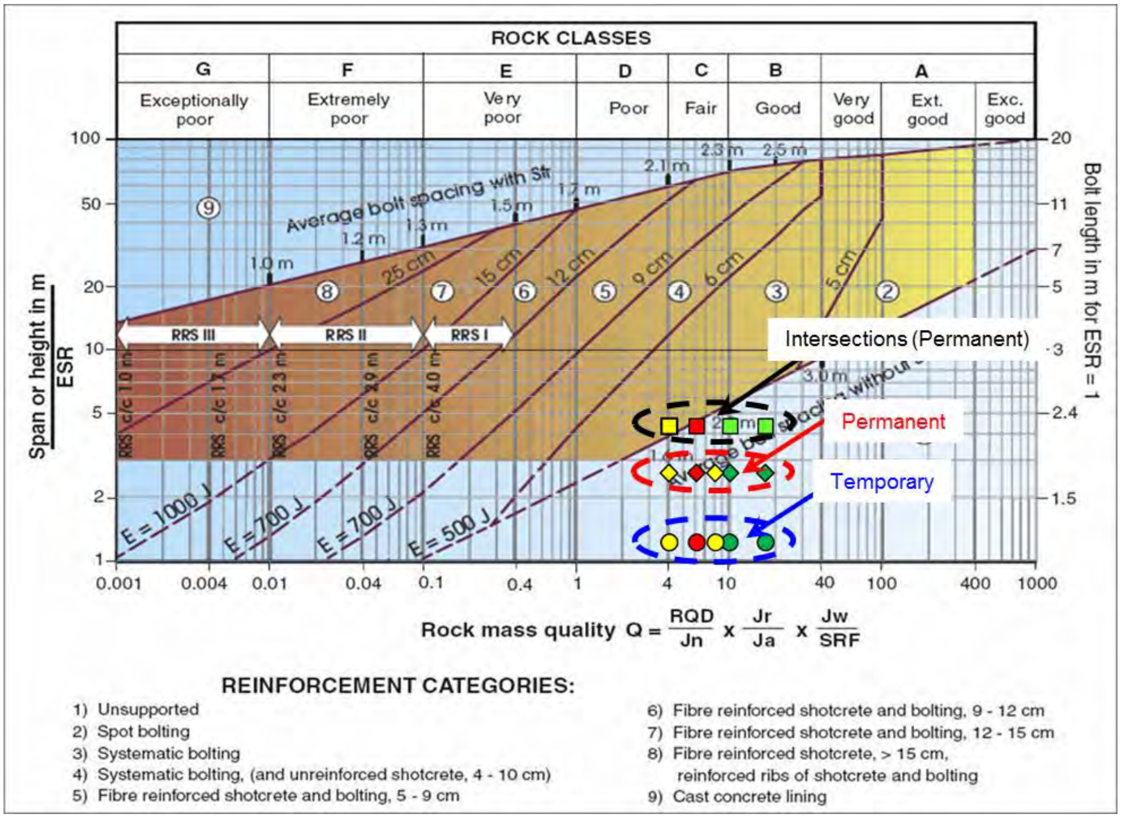
Source: JDS (2019)

To reduce potential stress impacts on mining, stopes should be sequenced to direct stresses away from the central mining areas and toward outer abutments. Stopes are mined in a hanging wall to footwall retreat sequence to cut-off, or divert, stresses away from the remaining stopes on the level.

16.3.7 Ground Support

Based on the range of anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support requirements were initially designed according to the Barton & Grimstad (1994) criteria (Figure 16-10). The Q-system also accounts for the life and use of the opening (e.g. man-entry or equipment only) with the excavation support ratio (ESR) parameter. The ESR is used to adjust the design span, in order to obtain the equivalent span for use in the Q Support Diagram; in effect, it imposes a higher factor of safety on critical structures with long life (such as an underground nuclear power station with an ESR rating of 0.5 to 0.8) than on temporary tunnels (such as temporary mine workings with an ESR rating of 2 to 5). An ESR of 1.6 was used for permanent development and temporary or short term ore development was assessed assuming an ESR of 3. Based on the Barton & Grimstad criteria, most of the non-intersection development would only require spot bolting as a minimum to remain open; however, pattern bolting with welded wire mesh is recommended to control loose material for all development where miners will enter. The ground support recommended for each type of permanent and temporary/ore development are summarized in Tables 16-5 and 16-6, respectively.

Figure 16-10: Barton Q Ground Support Chart



Source: JDS (2019)

16.3.8 Squeezing Ground

Based on discussions with former mine personnel at Madsen there were local zones of significant squeezing in access tunnels to the 8 Zone from the shaft. Review of historical maps indicates that the squeezing may be related to two specific geologic zones within the Russet Lake UMAF. The first is in the Russet Lake UMAF, near the contact with the Balmer Basalt where competence contrasts between the units likely resulted in preferential strain and enhanced alteration. The second zone is in the hanging wall of the 8 Zone mineralization. Figure 16-11 contains a reproduction by Pure Gold of one of the historical level maps showing the areas where steel sets, timber and/or shotcrete were required to control closure.

For the current FS, a hybrid bolt has been selected for approximately 50% of the 8 Zone development to accommodate for potential squeezing ground issues. The hybrid bolt is based on experience in similar ground conditions and stresses at the LaRonde mine in Northern Quebec as described by Mercier-Langevin and Turcotte (2007).

The hybrid bolt consists of a installing a resin bolt installed inside of a split set. The combination takes advantage of the benefits of both bolts without the limitations. As described by Mercier-Langevin and Turcotte (2007) the hybrid bolt has the following advantages:

- Yields at a constant high tonnage;

- High shear resistance;
- Can be successfully installed in badly crushed rock (reconditioning);
- Uses low cost, familiar, easy to install equipment;
- Quality control similar to rebar; and
- Corrosion resistance.

The bolt is installed according to the following steps:

1. A 37 mm hole is drilled and a 2.1 m long 39 mm split set is installed with the resin already inside the split set in order to prevent broken fragments of rock from entering the bolt and hampering the next steps. The friction bolt is installed with a plate to hold the screen;
2. A 2.0 m long 22 mm rebar is installed inside the friction bolt and spun following the specific requirements of the resin used. The rebar has its own plate to prevent early failure of the friction bolt head; and
3. The rebar nut is tightened to push the plate against the head of the friction bolt.

The specifics of the methodology was optimized for ground support elements and procedures already in use at the LaRonde mine and adjustments to the process may be specific to Madsen.

Table 16-5: Ground Support Recommendations for Permanent Development

Drift Development (Long term / permanent)	Austin, S. Austin, McVeigh & A3 Veins	8 Zone (assume 50% each)	
		Good Ground	Squeezing Ground
Height	4.2 m	4.2 m	4.2 m
Width	4.2 m	4.2 m	4.2 m
Back profile (arched / flat)	arched	arched	arched
Primary Back Bolts - Type	Resin Bolt	Resin Bolt	Split Set/Resin Bolt
Primary Back Bolts - Diameter	22 mm (#7)	22 mm (#7)	39 mm / 22 mm
Primary Back Bolts - Length	1.8 m	2.1 m	2.1 m
Primary Back Bolts - Spacing	1.2 m	1.0 m	1.0 m
Secondary 3-Way Intersection Back Bolts - Type	Resin Bolt	Resin Bolt	Cable Bolts
Secondary 3-Way Intersection Back Bolts - Diameter	22 mm (#7)	22 mm (#7)	Single Strand
Secondary 3-Way Intersection Back Bolts - Length	2.4 m	2.4 m	5.0 m
Secondary 3-Way Intersection Back Bolts - Spacing	1.8 m	1.8 m	2.0 m
* Notes: -'Secondary 3-Way Intersection Back Bolts' are in addition to the 'Primary Back Bolts' above at intersections. -Intersections in the UMAFs should be avoided in the Austin, S. Austin, McVeigh & A3 Veins otherwise additional cables and/or shotcrete may be necessary.			
Secondary 4-Way Intersection Back Bolts - Type	Resin Bolt	Resin Bolt	Cable Bolts
Secondary 4-Way Intersection Back Bolts - Diameter	22 mm (#7)	22 mm (#7)	Single Strand
Secondary 4-Way Intersection Back Bolts - Length	3.0 m	3.0 m	5.0 m
Secondary 4-Way Intersection Back Bolts - Spacing	2.0 m	2.0 m	2.0 m
* Notes: -'Secondary 4-Way Intersection Back Bolts' are in addition to the 'Primary Back Bolts' above at intersections. Intersections in the UMAFs should be avoided in the Austin, S. Austin, McVeigh & A3 Veins otherwise additional cables and/or shotcrete may be necessary.			
Wall Bolts - Type	Resin Bolt	Resin Bolt	Split Set/Resin Bolt
Wall Bolts - Diameter	22 mm (#7)	22 mm (#7)	39 mm / 22 mm
Wall Bolts - Length	1.8 m	1.8 m	2.1 m
Wall Bolts - Spacing	1.2 m	1.2 m	1.0 m
Wall Bolts - Distance From Floor*	1.5 m	1.5 m	0.5 m
Screen - Gauge	#6, Galvanized	#6, Galvanized	#6, Galvanized
Screen - Wire Spacing	10 cm	10 cm	10 cm
Screen - Distance from Floor	1.5 m	1.5 m	0.5 m
Shotcrete - Requirement	5 %	5 %	25 %
Shotcrete - Thickness	5 cm	5 cm	5 cm
Shotcrete - Maximum Distance From Floor	0.0 m	0.0 m	0.0 m

Source: JDS (2019)

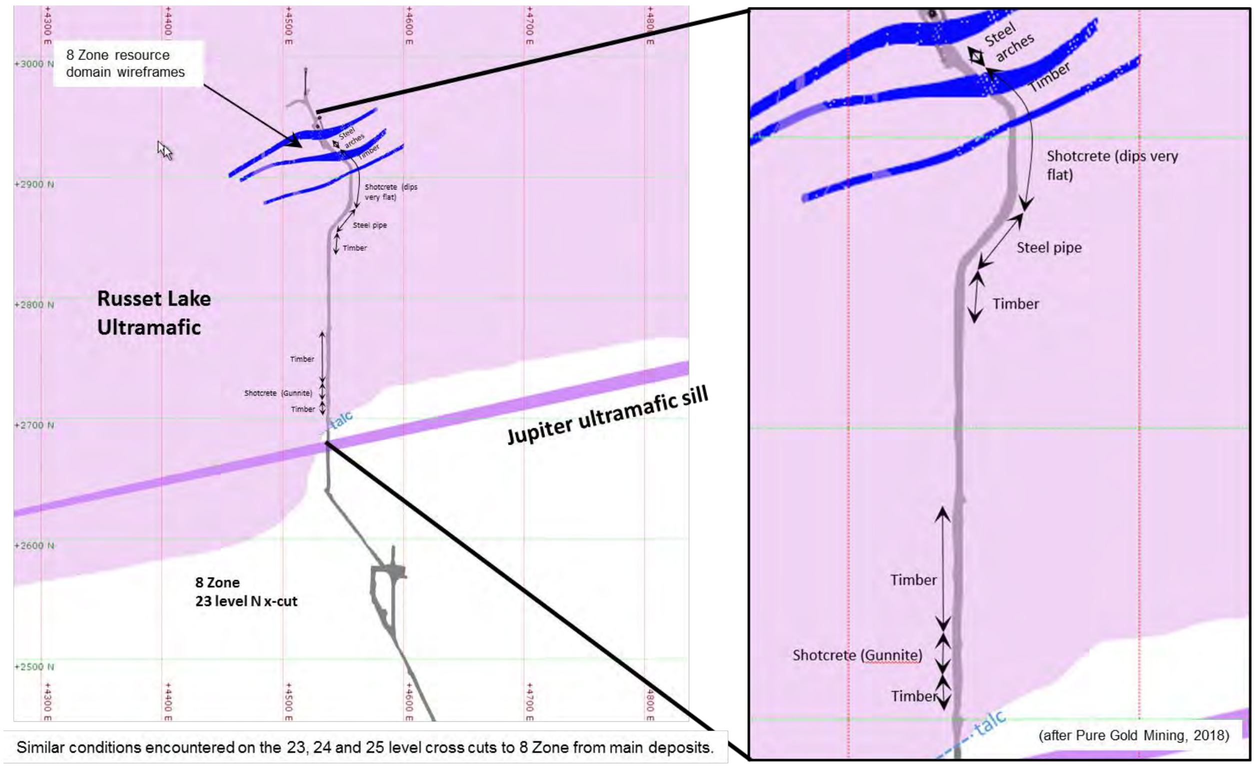
Table 16-6: Ground Support Recommendations for Ore Development

Drift Development (short term / ore)	Austin, S. Austin, McVeigh & A3 Veins		8 Zone (Drift & Fill)			
	LHOS Development	Conventional Cut & Fill	Good Ground		Squeezing Ground	
Maximum Height*	3.5 m	2.4 to 3.5 m	2.4 m		2.4 m	
Maximum Width*	3.0 m	2.0 to 3.0 m	2.0 to 4.0 m	4.0 to 6.0 m	2.0 to 4.0 m	4.0 to 6.0 m
Shape (arched / flat / shanty)	Flat	Flat	Shanty		Shanty	
Dip - Footwall	90.0	90.0	90.0		90.0	
Dip – Hanging Wall	90.0	90.0	40.0		40.0	
*Maximum dimensions refer to largest stable excavation without application of heavy ground support and may not reflect actual stope dimensions or ground support applied in the mine design.						
Primary Back Bolts - Type	Swellex Pm12	Swellex Pm12	Swellex Pm12	Swellex Pm12	Swellex Pm12	Swellex Pm12
Primary Back Bolts - Diameter	27.5 mm	27.5 mm	27.5 mm	27.5 mm	27.5 mm	27.5 mm
Primary Back Bolts - Length	1.5 m	1.5 m	1.5 m	2.1 m	2.1 m	2.1 m
Primary Back Bolts - Spacing	1.2 m	1.2 m	1.2 m	1.2 m	1.0 m	1.0 m
Secondary 3-Way Intersection Back Bolts - Type	-	-	-	Cable Bolts	-	Cable Bolts
Secondary 3-Way Intersection Back Bolts - Diameter	-	-	-	Single Strand	-	Single Strand
Secondary 3-Way Intersection Back Bolts - Length	-	-	-	4.0 m	-	4.0 m
Secondary 3-Way Intersection Back Bolts - Spacing	-	-	-	1.5 m	-	1.5 m
* Notes: -'Secondary 3-Way Intersection Back Bolts' are in addition to the 'Primary Back Bolts' above at intersections. -Intersections in the UMAFs should be avoided in the Austin, S. Austin, McVeigh & A3 Veins otherwise additional cables and/or shotcrete may be necessary.						
Secondary 4-Way Intersection Back Bolts - Type	-	-	-	Cable Bolts	-	Cable Bolts
Secondary 4-Way Intersection Back Bolts - Diameter	-	-	-	Single Strand	-	Single Strand
Secondary 4-Way Intersection Back Bolts - Length	-	-	-	5.0 m	-	5.0 m
Secondary 4-Way Intersection Back Bolts - Spacing	-	-	-	1.5 m	-	1.5 m
* Notes: -'Secondary 4-Way Intersection Back Bolts' are in addition to the 'Primary Back Bolts' above at intersections. -Intersections in the UMAFs should be avoided in the Austin, S. Austin, McVeigh & A3 Veins otherwise additional cables and/or shotcrete may be necessary.						

Drift Development (short term / ore)	Austin, S. Austin, McVeigh & A3 Veins		8 Zone (Drift & Fill)		
	LHOS Development	Conventional Cut & Fill	Good Ground	Squeezing Ground	
Wall Bolts - Type	Swellex Pm12	Swellex Pm12	Swellex Pm12	Split Set/Resin Hybrid	Split Set /Resin Hybrid
Wall Bolts - Diameter	27.5 mm	27.5 mm	27.5 mm	39/22 mm	39/22 mm
Wall Bolts - Length	1.5 mm	1.5 mm	1.5 mm	2.1 mm	2.1 mm
Wall Bolts - Spacing	1.2 m	1.2 m	1.2 m	1.2 m	1.0 m
Wall Bolts – Maximum Distance from Floor	1.5 m	1.5 m	1.5 m	0.5 m	0.5 m
Screen - Gauge	#6	#6	#6	#6	
Screen - Opening Size	10 cm	10 cm	10 cm	10 cm	
Screen - Distance from Floor	1.5 m	1.5 m	1.5 m	0.5 m	
Shotcrete - Requirement	-	-	5 %	25 %	
Shotcrete - Thickness	-	-	5 cm	5.0 cm	
Shotcrete – Maximum Distance from Floor	-	-	0.0 m	0.0 m	

Source: JDS (2019)

Figure 16-11: Map of 8 Zone Access Tunnel Ground Support



Source: JDS (2019)

16.4 Mine Planning Criteria

The mine planning criteria for the Madsen Project are listed below:

- The pre-production mine development period will be approximately 5 months. Any development ore mined will be stockpiled on surface during this period with no stopes scheduled to start mining until the mill gets commissioned and the primary ventilation circuit is established;
- For all planned raise development raisebore contractors will be utilized;
- Owner operator development and production mining has been assumed for the duration of the mine life;
- A combination of diesel and battery powered haulage fleet will be utilized; and
- Mined voids will be filled with hydraulic fill, or un-mineralized development rock.

Other key mine planning criteria are summarized in Table 16-7.

Table 16-7: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	365
Shifts per Day	Shifts	2
Hours per Shift	Hour	12
Work Roster	Days on-off	4x4
Nominal Ore Mining Average Rate	t/d	800
Annual Ore Mining Average Rate	t/a	~292,000
Ore Specific Gravity	t/m^3	2.94
Un-mineralized Rock Specific Gravity	t/m^3	2.94
Swell Factor		30%

Source: JDS (2019)

Gold grade cut-off value, dilution and mining ore recovery criteria have been defined previously in Sections 15.1 to 15.3 of this report.

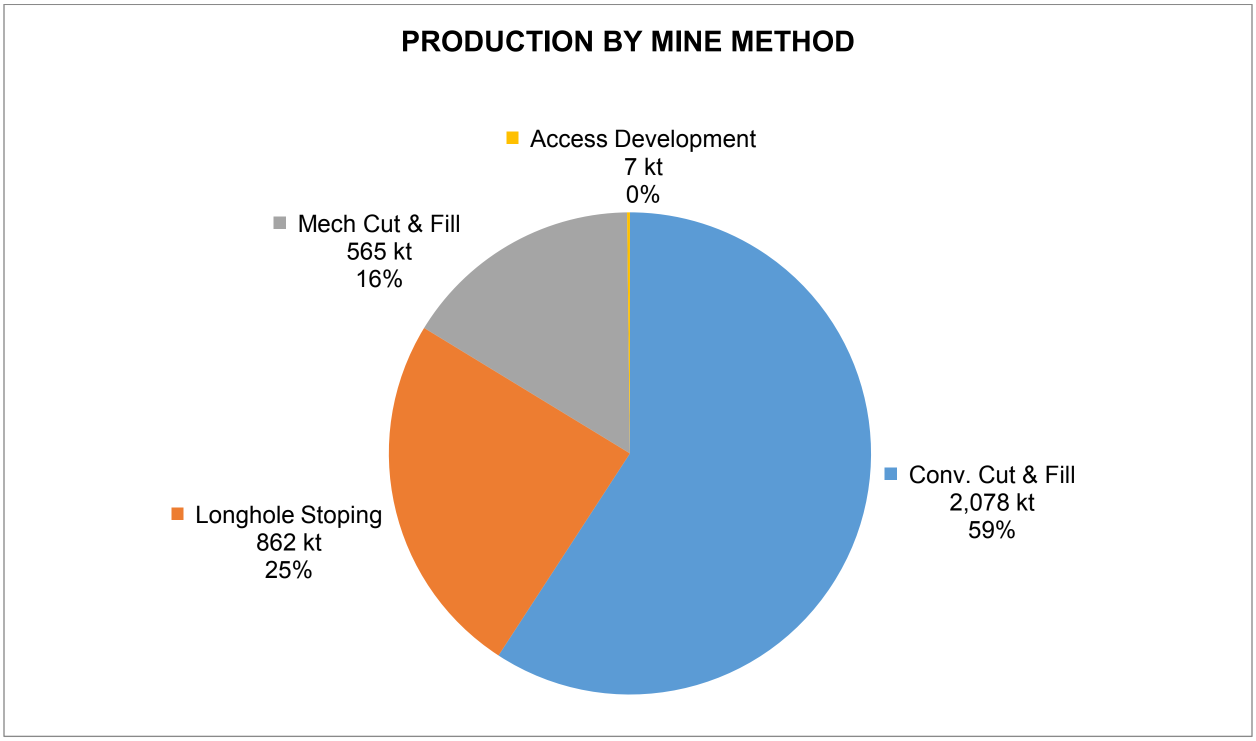
16.5 Mining Methods

Three mining methods are proposed for the Madsen deposit, LH, CCF and MCF. LH stoping is the preferred mining method at Madsen and will generally be used in areas of good ground, where the dip is greater than 60°, veins are continuous along strike for 30 m or greater, and continuous vertically for at least 15 m. MCF will be utilized where greater ore selectivity may be required, areas of shallow dips and poor ground. CCF mining will be utilized where accessing the stope by vertical development proves more efficient than by lateral access.

The mine generally consists of blocks of stopes separated by regions of lower grade material and/or historical workings. Each block regardless of mining method will be extracted in a bottom up sequence. With the current geological modelling it’s not envisaged that any sill pillars will be required.

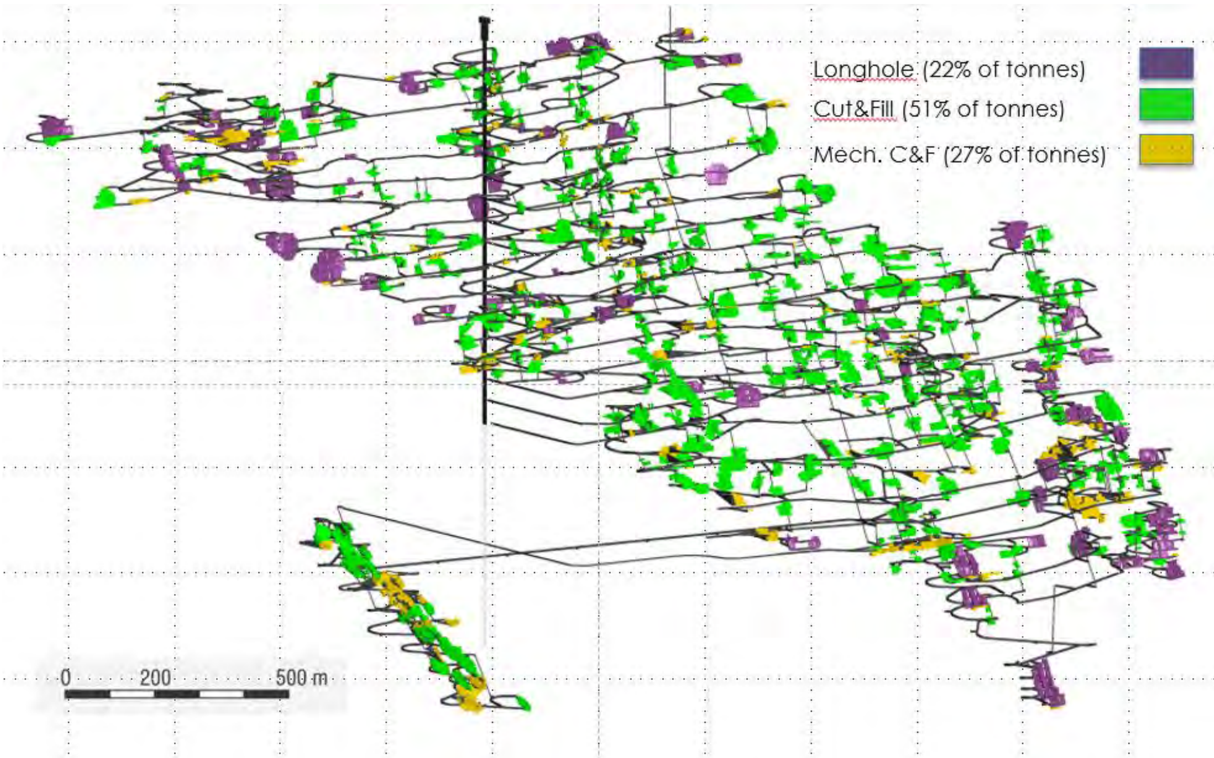
A breakdown of the mining production by method is shown in Figure 16-12. Mining method by location is shown in Figure 16-13.

Figure 16-12: Production by Mining Method



Source: JDS (2019)

Figure 16-13: Mining Method by Location. Oblique Long Section Looking 030 (Mine Grid)



Source: Pure Gold (2019)

16.5.1 Longhole Mining

LH will be used where the rock quality and ground conditions allow and where vein geometry is amenable to economic stope design. LH is the highest productivity method selected for the production phase of mining.

LH is the least selective of the mining methods when applied over long vertical distances due to potential drill hole deviation and vein geometry variations. To mitigate these effects, level spacing was limited to 25 m (floor to floor), sill drifts will be driven at 3.0 m W x 3.5 m H giving a maximum vertical hole length of 21.5 m. A minimum stope mining width of 1.7 m was used for design and planning purposes. Where the mineralized vein is less than the minimum mining width internal planned dilution is included to increase the width to the minimum 1.7 m.

Unconsolidated un-mineralized development rock will be used for backfill. In order to minimize dilution, rib pillars have been designed between stopes. Rib pillars were designed with a length of 1.5 times the width of the adjacent stopes, with a spacing that varies with depth as shown in Table 16-8.

Table 16-8: Rib Pillar Spacing Design

Depth from Surface	Maximum Rib Pillar Spacing
Surface to ~400m	45
400m to ~800m	30
Deeper than 800m	15

Source: JDS (2019)

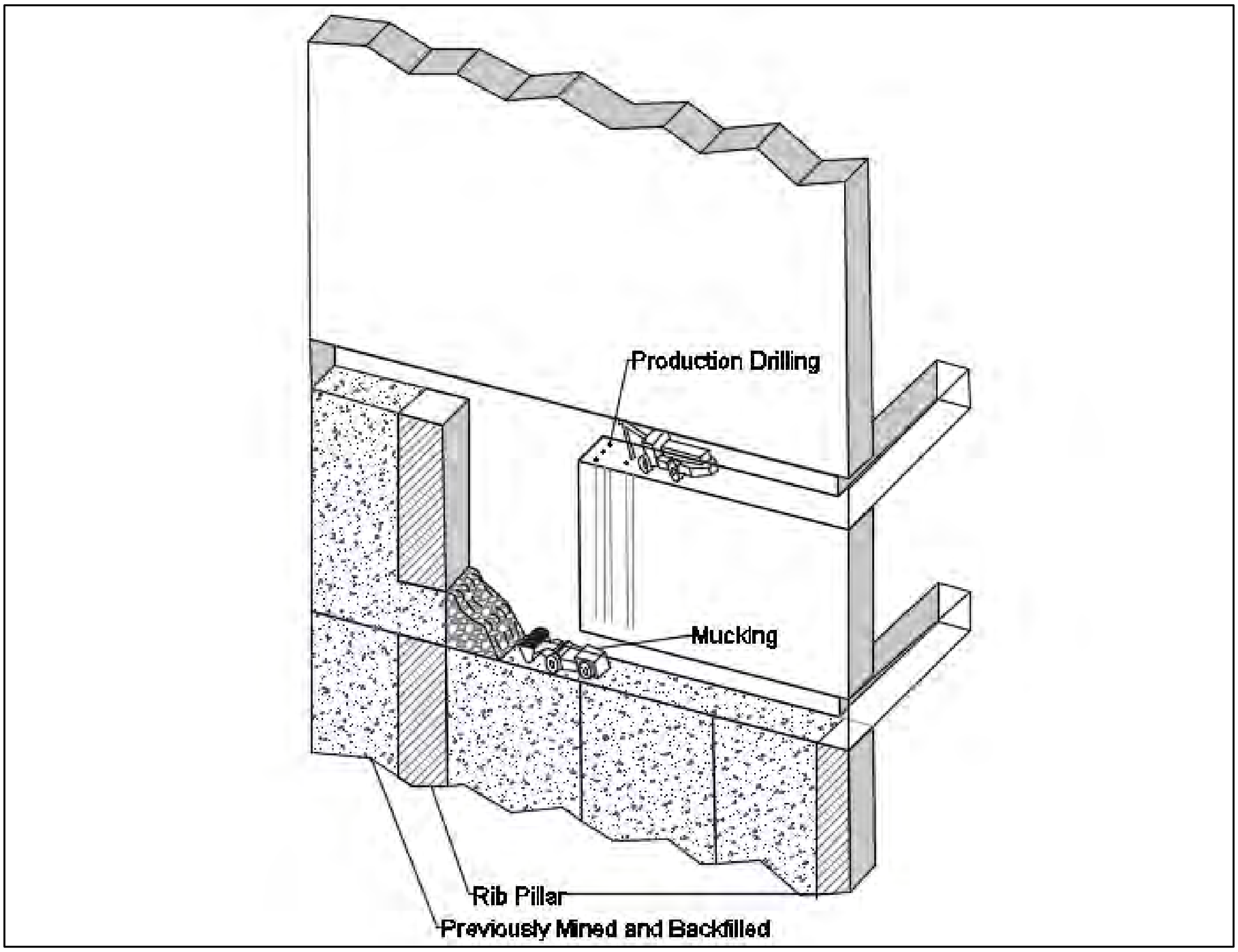
During top and bottom sub-level sill development, mine geologists will inspect and map headings to identify structural controls and collect channel samples as part of the mine grade control protocol. Infill drilling will occur along the sill drifts using a bazooka drill. Information from sampling and mapping in the top and bottom sills will be used, in conjunction with exploration and definition drilling results, to identify the economic limits and complete final stope designs. Mining engineers will use all collected grade control information to prepare drill hole designs and blasting sequences to minimize extraction dilution and maximize ore recovery. Mine surveyors will then locate and mark drill hole locations and provide drill hole lengths and orientations for drilling. Blast holes will be drilled from the upper drift to the lower drift and drilled to break-through into the lower drift where possible to verify drilling accuracy. If the hole deviates significantly from the design a new hole will be drilled. A five percent re-drill allowance has been assumed in the cost model.

The LH mining cycle will begin with blasting the slot raise to provide a free face for the first LH round as well as adequate void for swollen muck. Production blasting will begin at the stope ends and retreat to the cross-cut and/or access. Blasted material will report to the bottom of the stope where load haul dump (LHD) machines will muck the ore via remote control. LHD remote operators will be stationed in either safety cut-outs, around the corner, or on pedestals in order to maintain a safe line of site with the LHD. Specific safety aspects of remote operator location will be assessed for each stope prior to mucking.

Top sills will be utilized for drifting, loading, and blasting of the stope below, as well as providing access for LHDs for backfilling. Once the stope below is completed and backfilled, the drift will then be used as a bottom sill for mucking ore from the stope above. In the event a top sill is unavailable, the lower extraction sill may be used to drill up holes. Uphole stopes are planned to be left open, and make up approximately 3% of the total mining reserve by tonnes.

A typical LH layout at the Madsen deposit is shown in Figure 16-14.

Figure 16-14: Madsen Deposit Typical Longhole Layout



Source: JDS (2019)

16.5.2 Mechanized Cut and Fill

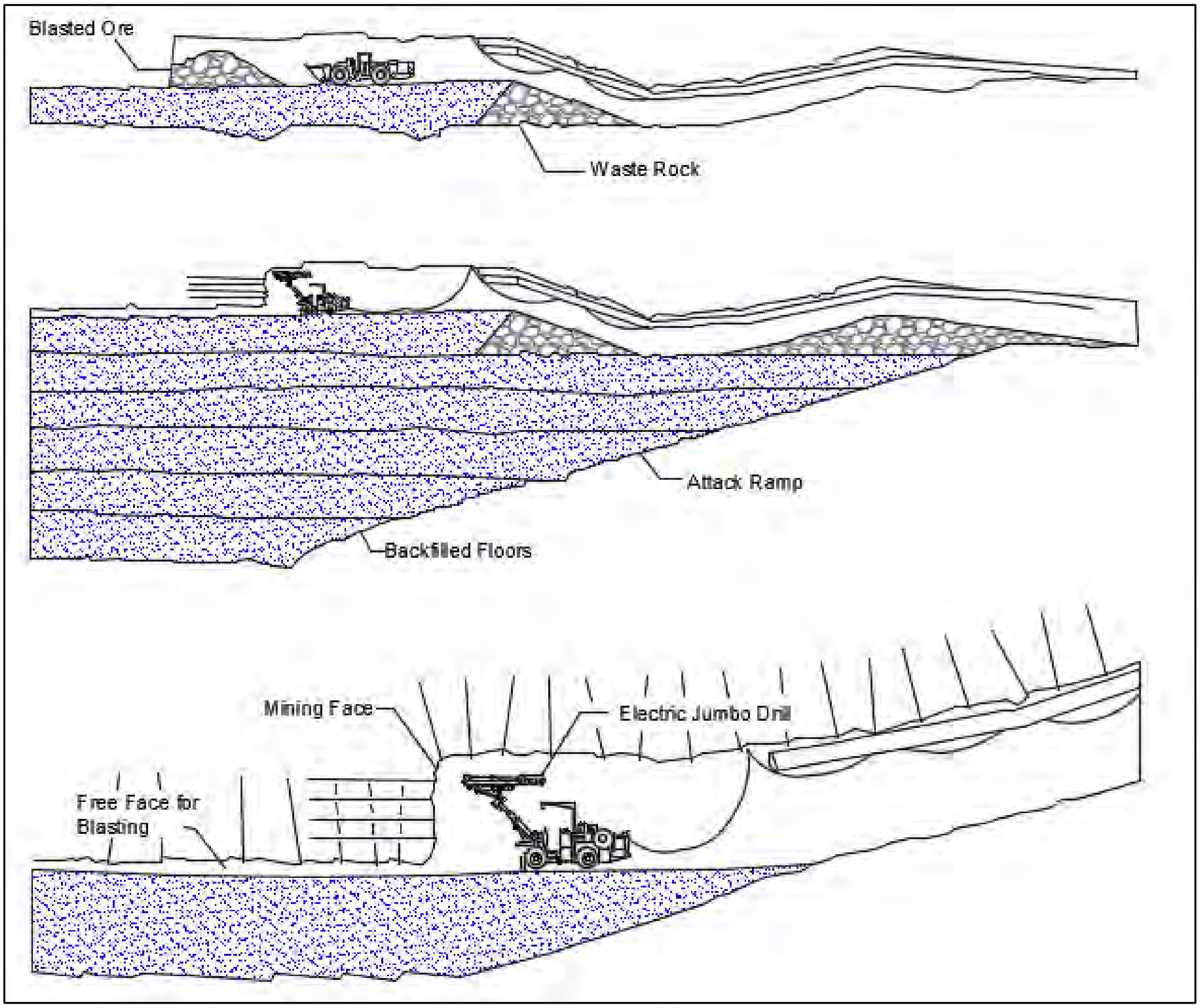
Overhand MCF mining methods were selected for areas that have lower quality rock and/or where the resource geometry is not amenable to LH. In this method, mining begins at the bottom of the ore block and progresses upward. During the mining sequence, the back of the excavation is temporarily supported using rock bolts before the stope is back filled to form the floor of the next level of mining. Backfill is designed to provide sufficient excavation support and a strong working floor for personnel and equipment. MCF stopes at Madsen will be filled with hydraulic fill, however there is flexibility in this method to allow un-mineralized rock fill when practical.

Stopes are accessed via attack ramps driven at a maximum 15% gradient from a working level or access ramp. Minimum drift size for MCF is 2.0 m W x 2.4 m H. In stopes where the width of the economic mineralization swells wider than the primary drift, the wall would be slashed up to a maximum of 4 m. In areas that are at 4 m W for extended sections, a primary / secondary mining sequence will take place whereby the primary cuts are taken first and filled with a structural cemented backfill. This provides structural wall support to permit mining directly adjacent to the primary cut. 15% of hydraulic fill placed will be structural fill. Wall slashing makes up approximately 16% of the MCF tonnes, and the average slashing width is 0.6 m where it occurs.

On the top cuts of stoping blocks where no mining occurs directly above, the neighboring cut below will be used as an extraction drift and the final top cut will be blasted with upholes in a retreat mining sequence. Only the extraction drift of these MCF uppers would be backfilled, the upper stope would remain open. Using uppers in this way reduces access development.

Specialized mechanized narrow vein mining equipment has been selected to operate in these narrow drifts. A typical mechanized cut and fill layout is shown in Figure 16-15.

Figure 16-15: Mechanized Cut and Fill

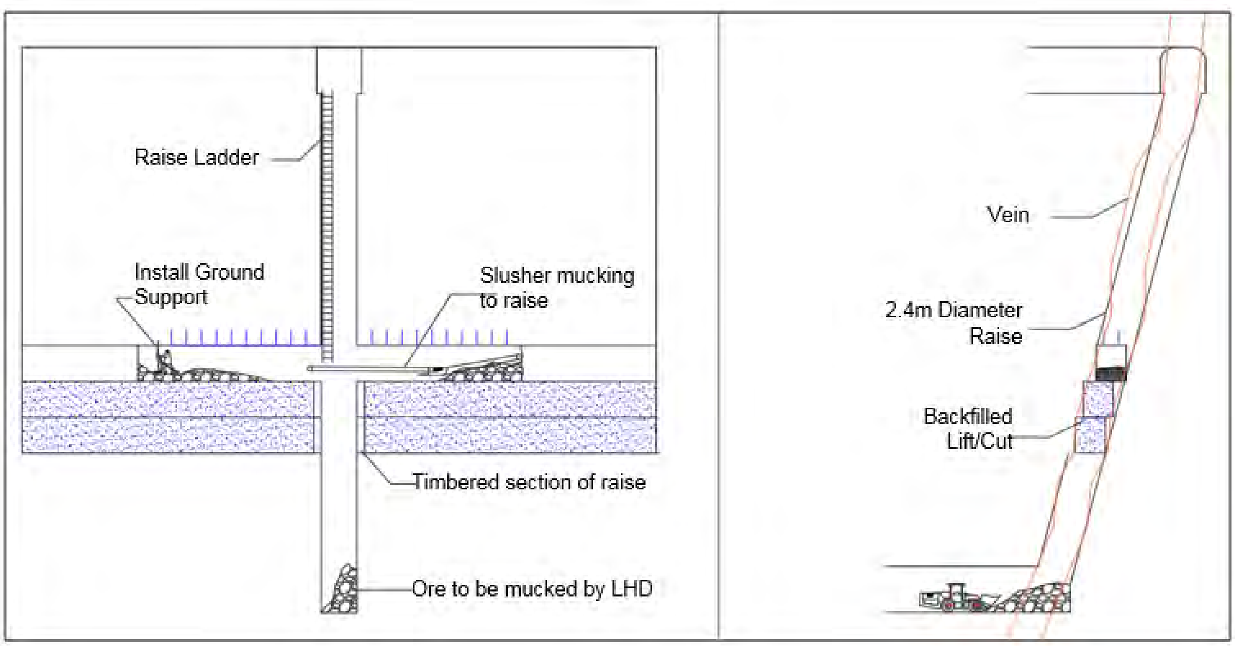


Source: JDS (2019)

16.5.3 Conventional Cut and Fill

Overhand CCF mining was selected for areas not amenable to LH mining or where a significant reduction in development was achieved by vertical development as opposed to the attack ramps used for MCF. Attack ramps used for MCF. In this method two near vertical raises will be driven with a raisebore machine for each stope. For stoping blocks under 5,000 t ore only one raise is planned to be used. A tugger will be installed at the top of the raises to transport supplies up and down the raise. Access ladders will be installed from the top of the raise and any ground remediation / support required can be completed from the top down. The first cut is driven off the raise and advanced along the vein, with ground support installed off the muck pile using jacklegs and stopers. Muck is removed from the stope using slusher machines scraping the muck back to the raise and down the pass. Once the cut is complete fill barricades will be constructed and will form the timbered section of the raise as the cuts advance upwards. Each cut is filled with hydraulic fill. An example of a small CCF stope with one raise is shown in Figure 16-16.

Figure 16-16: Long Section and Cross Section of a Conventional Cut and Fill Stope



Source: JDS (2019)

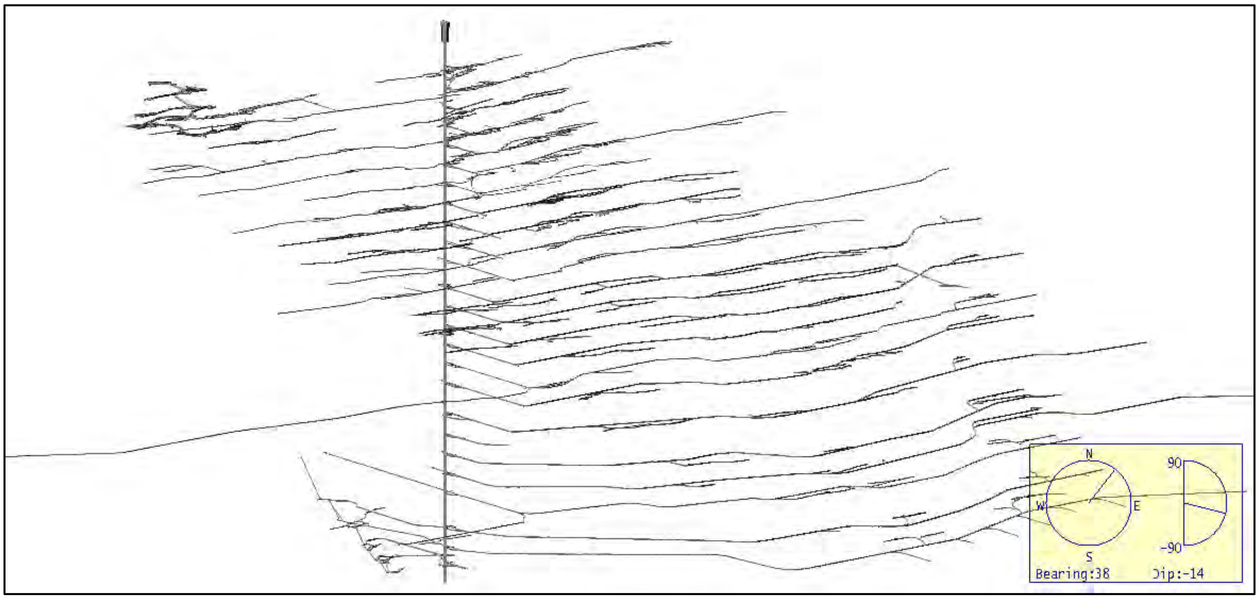
16.6 Mine Design

16.6.1 Existing Workings

The Madsen deposit currently has 27 levels that were initially developed from the existing shaft as track drifts. Drifts are on average 2.8 m W x 2.8 m H, level spacing is generally ~50 m (150 ft). A decline ramp extends 130 m below surface in the McVeigh area of the deposit. The current ramp has been driven at a 15% gradient with a curvature radius of 15-16 m. The mine plan will use where possible existing development to access new stopes. Where mobile equipment require access, levels will be slashed to 3.7

m W x 3.7 m H; there are 12,590 m equivalent of level slashing in the mine plan. Areas that are being utilized for ventilation or services will be rehabbed; there are 1,480 m of rehab in the mine plan. Only sections of the levels that are required will be either rehabbed (ventilation) or slashed out (equipment access). Figure 16-17 shows a 3D view of the existing mine development.

Figure 16-17: Perspective View of the Madsen Deposit existing Development Workings (Mine Grid)



Source: JDS (2019)

16.6.2 Stope Optimization

Mine planning for the Madsen project was completed by using Maptek™ Vulcan™ (Vulcan) software.

Vulcan's stope optimization software was utilized to identify mineable stope blocks and generate optimum stope shapes within the resource block model. These shapes were used to guide the final stope design, Table 16-9 shows the optimization parameters used for the shape optimization software.

Table 16-9: Stope Optimization Parameters

Parameter	Unit	Value (All Zones excl. 8 Zone)	Value (8 Zone)
Block Model		Various	madsubfin_z8_feb19_2018.bmf
Cut-off Variable		Au_indicated*	Au_indicated
Stope Orientation Plane		XZ	XZ
Framework Bearing	degrees	90 (Mine grid)	80 (Mine Grid)
Step X	m	6	6
Step Z	m	6	2.4
Cut-Off	Au g/t	4.75 (**4.00 McVeigh)	4.75
Minimum Stope width	m	2	2
Top to Bottom Max Ratio	#	2.25	3.0
Max Strike Deviation	degrees	-45 to +45 w/ max change 20.0	-45 to +45 w/ max change 20.0
Minimum Dip Footwall	degrees	55	85
Minimum Dip Hangingwall	degrees	125	35
Max Dip Change between stopes	degrees	20	10

* Au_indicated is the combination Auhg_final_dyk and Aulg_final_dyk gold variables where only the indicated blocks carry grade.
** Lower Cut-off Grade used for McVeigh
Source: JDS (2019)

Optimizations were run separately for each zone. The resulting stope optimizer shapes were reviewed and grouped into likely economic stope blocks and assessed by mining method. Ultimately every stope was designed manually with the stope optimization shapes used as a guide and improved upon where possible.

16.6.3 Production Rate

The following factors were considered in the estimation of the underground mine production rate:

- Mining inventory tonnage and grade;
- Geometry of the mineralized zones;
- Amount of required development;
- Stope productivities;
- Sequence of mining and stope availability; and
- Mine layout and location of stopes.

Tonnes per vertical metre for the mine reserve is approximately 2,490 t / vert. m.

An underground mine production rate of 800 t/d was selected for this FS and is considered appropriate due to the productivities of the selected stoping methods and available working faces and/or stopes. The selected mine rate of 800 t/d requires that approximately 115 vertical metres are mined annually, or approximately two mine levels.

16.6.4 Mine Design Criteria

Development profiles and gradients were selected based on the equipment specifications, ventilation requirements and stope sizes as shown in Table 16-10.

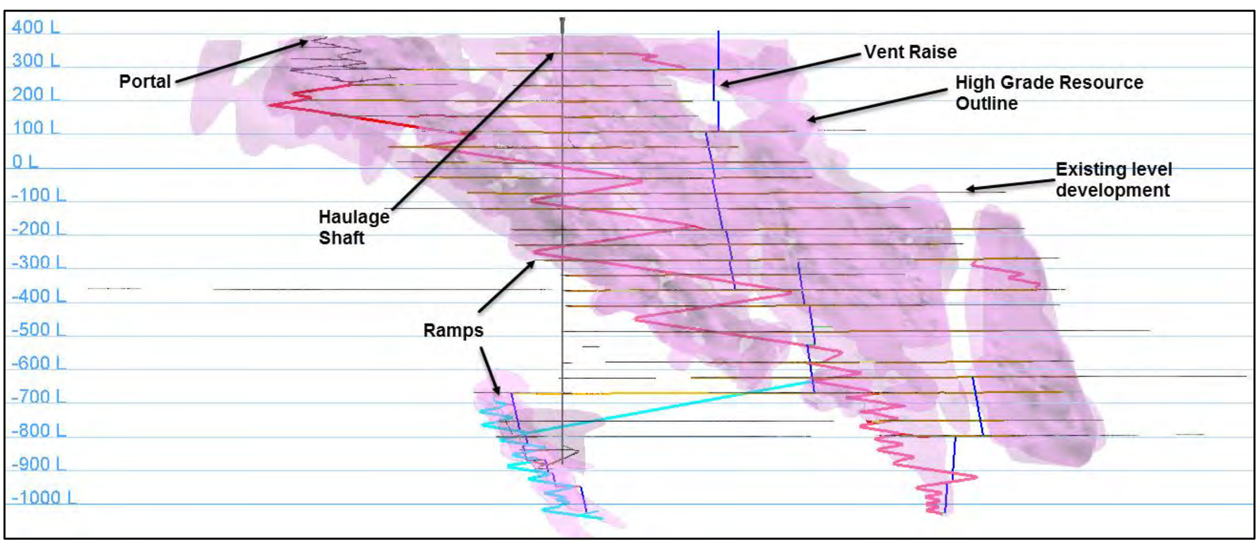
Table 16-10: Mine Design Criteria, Development Dimensions

Development Heading Parameters	Width (m)	Height (m)	Maximum Gradient (%)
Ramps	4.2	4.2	15
Level Access and Remucks	4.2	4.2	1
Access Ramps and Pump Stations	3.7	3.7	15
Level, Vent Drifts, Slashing, and Remucks	3.7	3.7	1
Attack Ramps	3.0	2.4	15
Cut and Fill Drives	2.0	2.4	1
Longhole Development	3.0	3.5	1
Ventilation (small) and Stope Access Raises	Ø 2.4		90º
Ventilation Raises (large)	Ø 3.7		90º

Source: JDS (2018)

Figure 16-18 shows the main ramps, portal location and vent raises relative to the deposit and existing workings.

Figure 16-18 : Existing and Planned Development and Ventilation



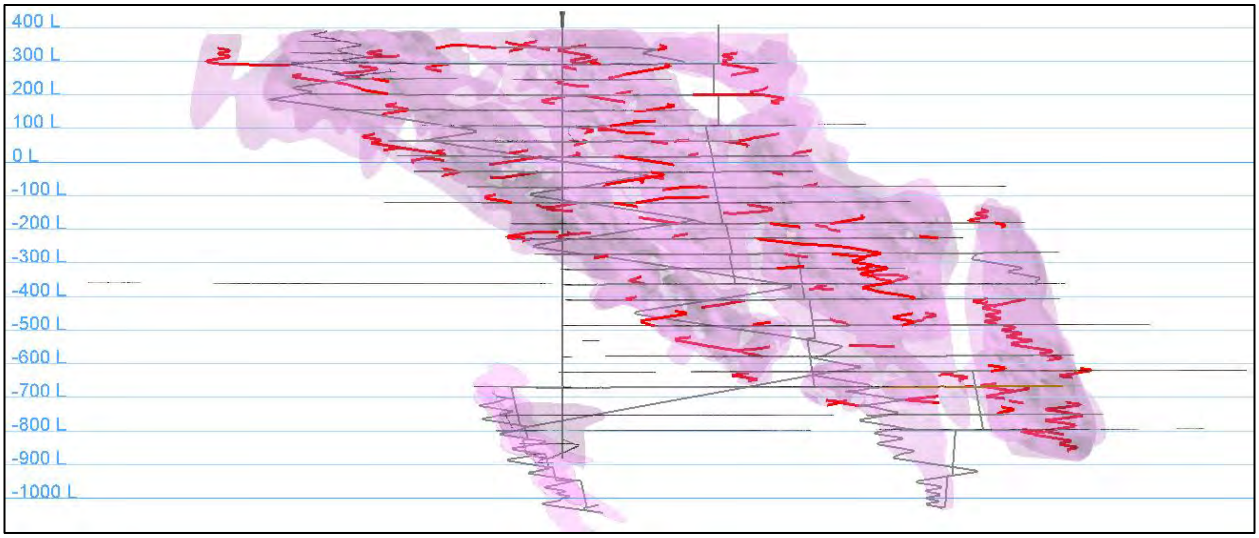
Source: JDS (2019)

16.6.5 Development Types

The main ramp will provide access to each of the existing mine levels and new stoping areas. The ramps are driven at -15% grade and 4.2 m W x 4.2 m H, with a curvature radius between 20 m to 30 m. The ramp has been designed with a minimum offset of 20 m from stopes to limit potential blast damage.

Access ramps are designed at 15% gradient and 3.7 m W x 3.7 m H, with a minimum curvature radius of 15 m to improve access to stoping blocks not serviced by the main ramp or current mine levels. Figure 16-19 shows where access ramps (in red) have been utilized.

Figure 16-19: Access Ramping



Source: JDS (2019)

Levels where equipment access is required will be slashed out to 3.7 m W x 3.7 m H. This will allow for truck access, ventilation and services. Sections of levels only to be used for services distribution and ventilation will only require rehab. Lateral access drives will be mined from slashed levels, haulage ramps and access ramps at 3.7 m W x 3.7 m H to stopes and stope access development.

LH stope sub level development will be mined at 3.0 m W x 3.5 m H to allow for LH production drills and scoop access.

MCF zones are accessed by attack ramps from access drives, levels or the haulage ramp directly. Attack ramps are driven at a maximum 15% grade and will stack vertically to access multiple cuts from a single access point. MCF cuts are driven at a minimum 2.0 m W x 2.4 m H; other un-mineralized development drifts are driven at 2.0 m W x 2.4 m H to connect isolated ore cuts on the same elevation.

CCF stopes are accessed via 2.4 m diameter raise bores. The raises are multi-purpose, and provide man access, ventilation and act as a muck passes. Where the ore portion of the cut does not meet the raise a 2.0 m W x 2.4 m H stope access is driven off the raise to the ore boundary.

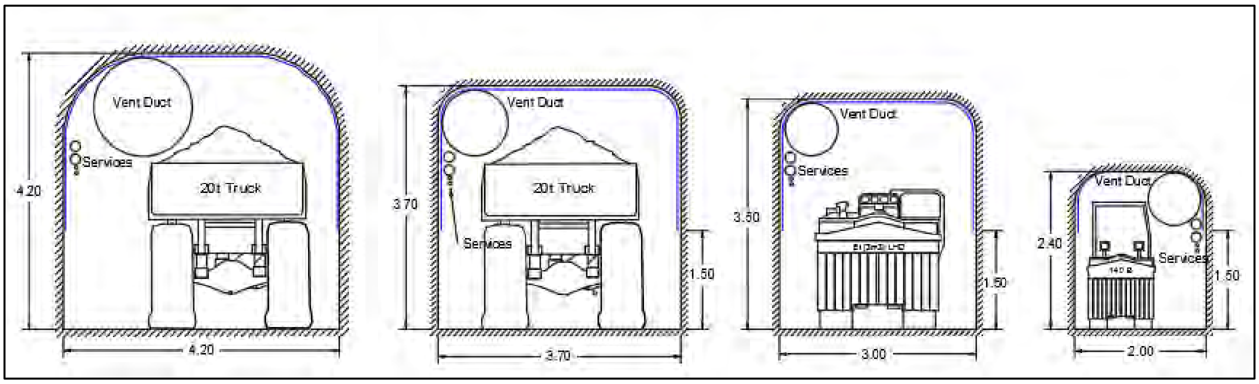
Remucks are excavated on the main ramp to help speed up the development mucking cycle. A maximum of 150 m separates the remucks, which are designed at 4.2 m W x 4.2 m H x 15 m L. Where a level access or lateral access drift has been designed off the ramp, these can also be utilized as temporary remucks.

Existing development on the levels will be utilized as water collection sumps, refuge station locations, and electrical bays. Safety bay cut-outs will be installed along the ramps at no more than 30 m apart.

Fresh air raises will be raise bores in two different diameters depending on location. From surface to 16 Level, 3.7 m diameter raises will be used. All other vent raises will be excavated at 2.4 m diameter.

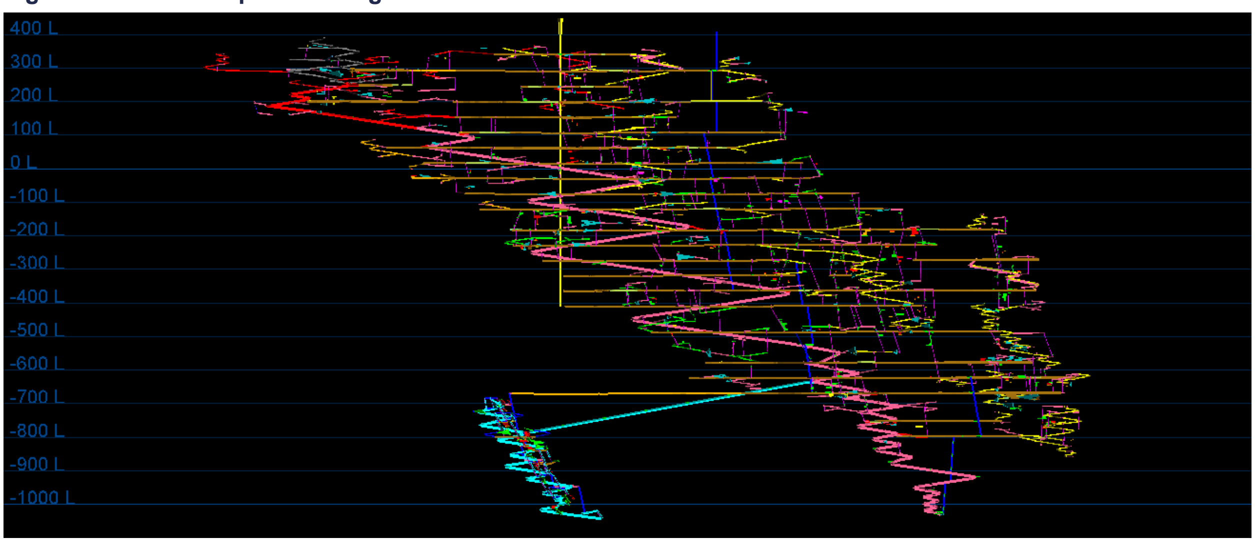
Design drift profiles and the 3D mine design is shown in Figures 16-20 and 16-21, respectively.

Figure 16-20: Drift Profiles



Source: JDS (2019)

Figure 16-21: Development Design at Madsen



Source: JDS (2019)

16.7 Mining Dilution and Recoveries

See Sections 15.2 and 15.3 for details on mining dilution and recovery parameters that were developed and utilized for stope design.

16.8 Mine Hydraulic Backfill

16.8.1 Tailings Characterization

The Madsen tailings are typical of cyanide leached gold mine tailings and are generally very fine with a P_{80} of about 75 microns, a P_{50} of 40 microns, and a P_{20} of 10 microns. The lack of sand sizes and the high proportion of slimes preclude the use of whole mill tailings for use as hydraulic fill.

Pilot scale testing has shown that cyclone classification of the tailings can be used to remove most of the slime fraction. The resultant underflow has about half the slimes fraction of the whole mill tailings. Although the pilot scale tests resulted in an 80/20 mass split between the underflow and overflow, cyclone simulations show that a more favorable mass split of 67/33 is achievable. The latter figure has been used for the designs herein.

Settled density testing shows that the Madsen tailings settle to a clear liquid in about 30 minutes. This testing shows that about 20 percent of the slurry volume can be decanted off immediately, and another 15 percent of the slurry volume will be removed during drain-down of the fill mass. The final settled density of the fill is estimated to be 1.86 t/m^3.

A single percolation test yielded a value of 70 mm/hr.

16.8.2 Backfill Strengths

The following backfill strength targets were provided to MineFill by JDS:

- For the overhand cut and fill stopes with widths up to 5 m, the UCS should exceed 150 kPa;
- For the overhand cut and fill stopes with widths over 5 m, the UCS should exceed 250 kPa; and
- For the longhole stopes the UCS should exceed 450 kPa.

It's assumed that all the mining will be overhand and that no undercutting of backfills will be required. No hydraulic backfill is used in LH stopes.

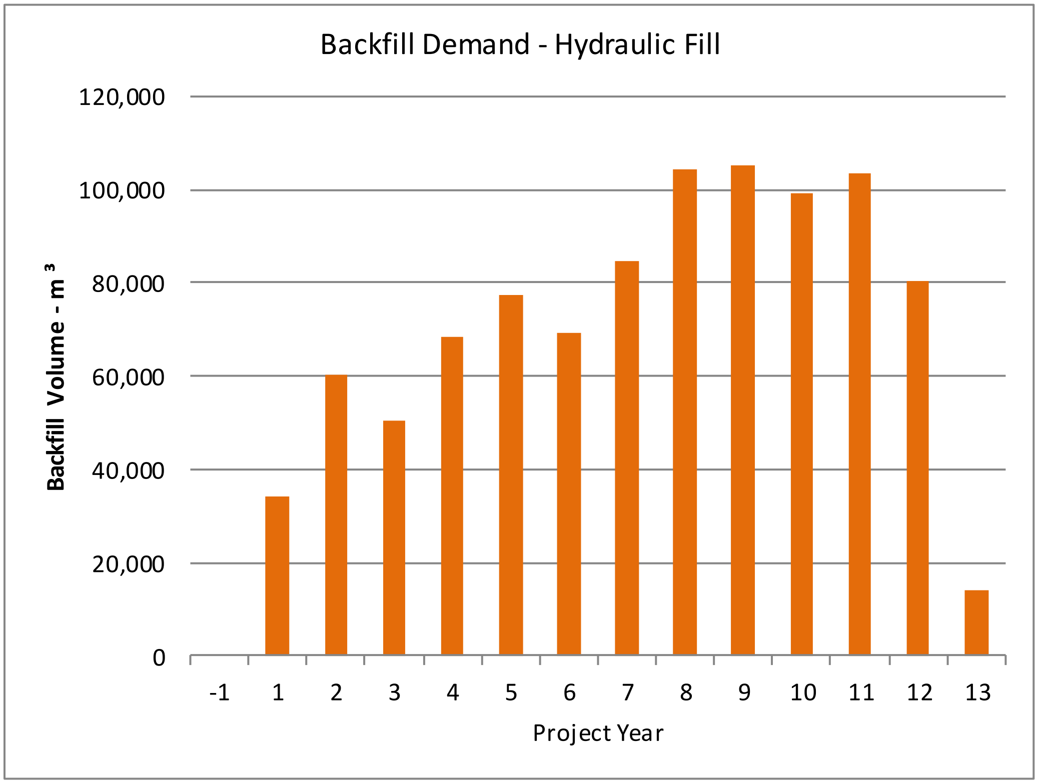
Uniaxial compression testing shows that the 150 kPa strength can be achieved with 5% Ordinary Portland Cement (OPC) binder, the 250 kPa strength can be met with 6% OPC binder, and the 450 kPa strength can be met with 7% OPC binder.

16.8.3 Backfill Demand

The Madsen backfill plant is sized to meet the demand for backfill in the mine. Due to the need to pre-cyclone all of the Madsen mill tailings, the plant must run in a batch mode based on the continuous storage of mill tailings. This results in a large amount of standby time for the backfill plant. The actual plant throughput is based on practical minimums for pipe sizing and filling rates in the stopes.

At the 800 t/d mining rate, the Madsen mine will create roughly 100,300 m^3 of void space per year for all stoping methods. The mine production schedule shows roughly 1 Mm^3 of hydraulic fill placed over the life of the mine, and 0.44 Mm^3 of rockfill. This equates to an average utilization of 70% for hydraulic fill and 30% for rockfill. The backfill demand for hydraulic fill over the life of the mine is shown in Figure 16-22.

Figure 16-22: Life of Mine Hydraulic Backfill Demand



Source: MineFill (2019)

In the early years of the mine the demand for hydraulic fill is around 60,000 to 80,000 m^3/year. In the last five years of the mine life the demand increases to 100,000 to 110,000 m^3/year. The increased demand in the latter years of the mine results from the backfilling of development drifts in zones that are completely mined out (e.g. 8 Zone).

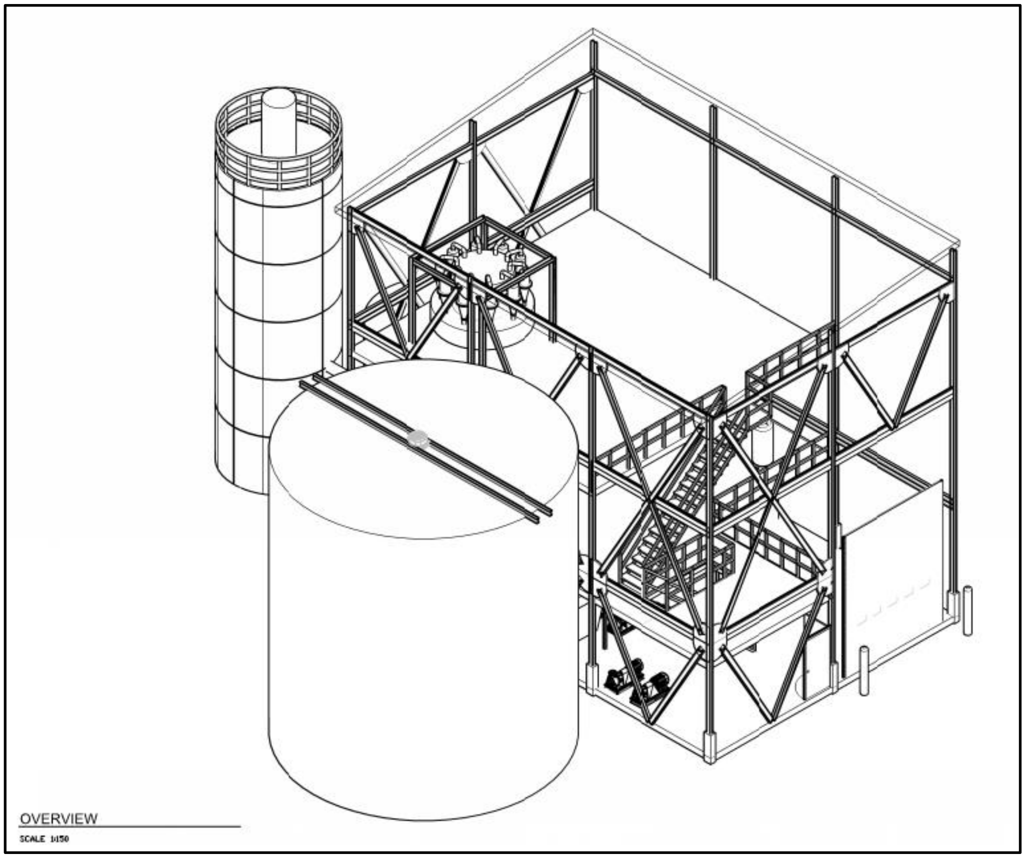
The proposed operating capacity of the backfill plant is 46 m^3/hr, which equates to 54 days of continuous filling or 15% plant utilization at the 60,000 m^3/year rate, increasing to 90 days or 26% at the 100,000 m^3/year rate. Hence, on average, the backfill plant is expected to operate for 5 to 6 hr/d. The actual plant utilization will be dependent on the actual filling factor as not all mining voids may be filled with backfill. For the purposes of this study we have assumed a filling factor of 100% for stopes that receive hydraulic fill.

16.8.4 Backfill Plant Design

The backfill scheme adopted in this study is simple in order to keep capital and operating costs at a minimum. Whole mill tailings will be delivered to a cluster of hydro-cyclones to classify the tailings sands and remove a portion of the slimes fraction. From there the classified tailings are delivered to a backfill mix tank where the tailings slurry is mixed with cement binder. The slurry is then pumped to the mine by centrifugal pumps.

The backfill plant will be located immediately adjacent the Madsen mill building in order to share infrastructure and services, and to simplify operations. The plant consists of four main sections: an external mill tailings receiving tank (tailings surge tank), the hydro-cyclones, the cement handling system, and the mixing tank. The total plant footprint covers 15 m by 20 m not including the cement silo or tailings surge tank (Figure 16-23).

Figure 16-23: Backfill Plant Isometric View



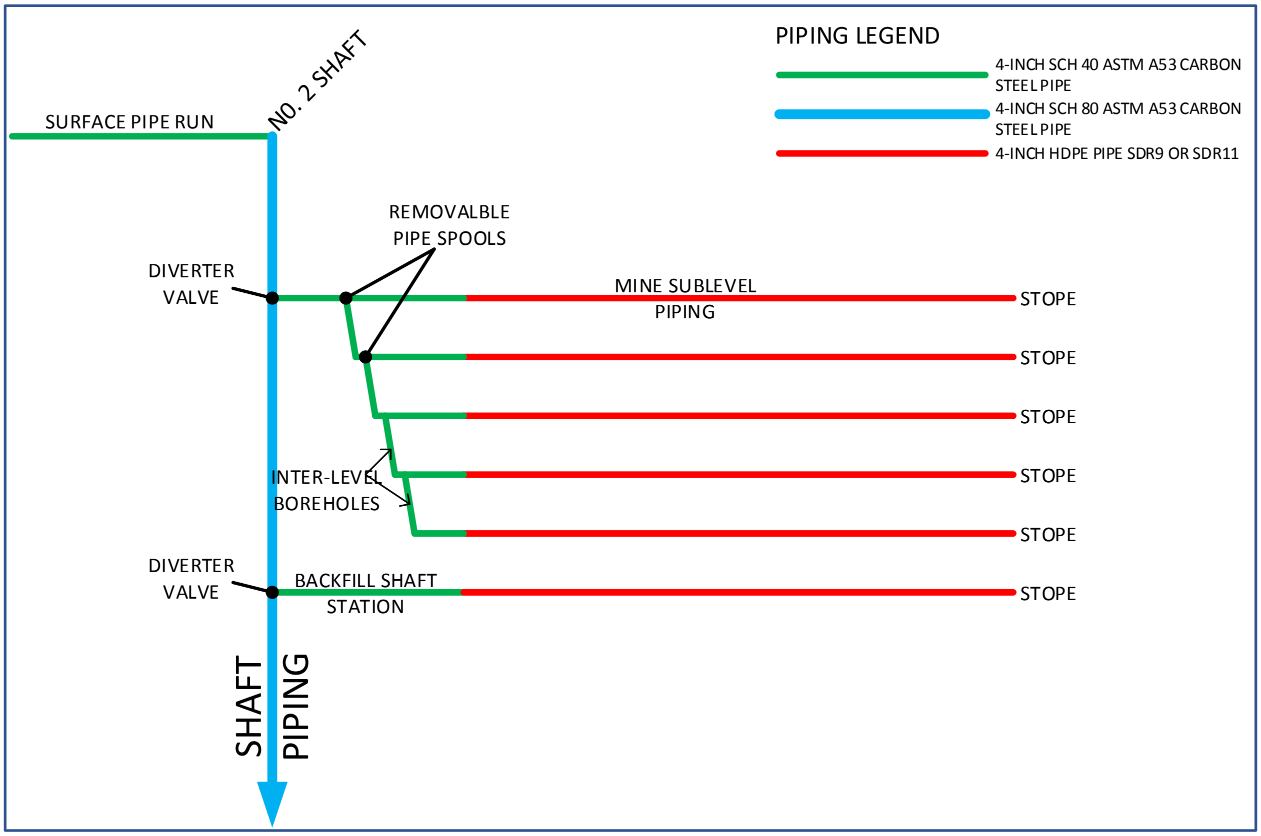
Source: MineFill (2019)

The backfill preparation building will have two main floors. The lower floor will house the slurry pumps, mixing tank, services, electrical controls, and flush water tank. The second floor will house the hydro-cyclones. An 80-tonne cement silo will be located adjacent the backfill building.

16.8.5 Backfill Delivery

The underground reticulation design is largely based on the use of gravity to deliver the hydraulic fill to stopes. The system makes use of the existing No.2 Shaft for delivery of slurry to the mining levels. Most, if not all, of the mine sublevel piping will be HDPE, whereas the shaft piping and surface piping runs will be carbon steel as shown in Figure 16-24.

Figure 16-24: Backfill Distribution System - Simplified View



Source: MineFill (2019)

Stopes at Madsen are relatively small and individual cuts from MCF and CCF can be filled in about 4 hours. The backfill production includes roughly one new cut each day.

16.8.6 Backfill Operations

Mining at Madsen will produce roughly 3.5 Mt of ore from just under 6,500 stope cuts. Of this, some 6,000 cuts, representing 75% of the ore, will come from cut and fill stopes. These stope cuts will be filled with hydraulic fill. The remaining 25% of the ore will come from longhole stopes and will be filled with unconsolidated rockfill.

The typical stope volumes are estimated to be in the order of 150 m^3 to 300 m^3. Most of these stopes are narrow vein with mining widths in the order of 2 m to 3 m, and strike lengths of over 100 m. Over the mine life, Madsen will require backfill in over 400 cuts annually. This equates to just over one stope cut per day. Hence, on average, the stope preparation crews need to prepare one new stope cut for filling every day.

The most common choice for fill fences in hydraulic fills is fabric-lined timber fences. A timber fence is very simple to construct with a small crew. The fences are typically located at the end of the access ramp at the stope entrance, or even in the stope itself on strike with the mining. The fences would be pinned to the rock walls of the stope with rebar or grouted bolts. Once the timber framing is in place, the fence is lined on the filling side with a non-woven geo-fabric to retain the solids in the stope. If installed correctly the discharge water should be clear.

Depending on the mining method the fill fences do not necessarily have to be tight to the back. However, if this is required, then breather tubes need to be inserted at the top of the fill fence to prevent over-pressuring the fence with compressed air. The filling is stopped when slurry or water appears in the breather tube.

The fences do not have to be particularly robust as they are only needed to support the fluid weight of a single 4 cu. m pour. A single timber bulkhead can be constructed by a crew of three in one shift.

16.9 Mine Services

16.9.1 Mine Ventilation

16.9.1.1 Key Design Considerations

The ventilation network and fresh air supply quantities were designed to comply with Ontario ventilation standards. Key design criteria include velocity requirements, airflow allocation criteria for fixed facilities and other underground infrastructure, k-factors for different airways, climate modelling criteria (mine air heating), auxiliary ventilation criteria, and others.

Airflow requirements for mobile equipment:

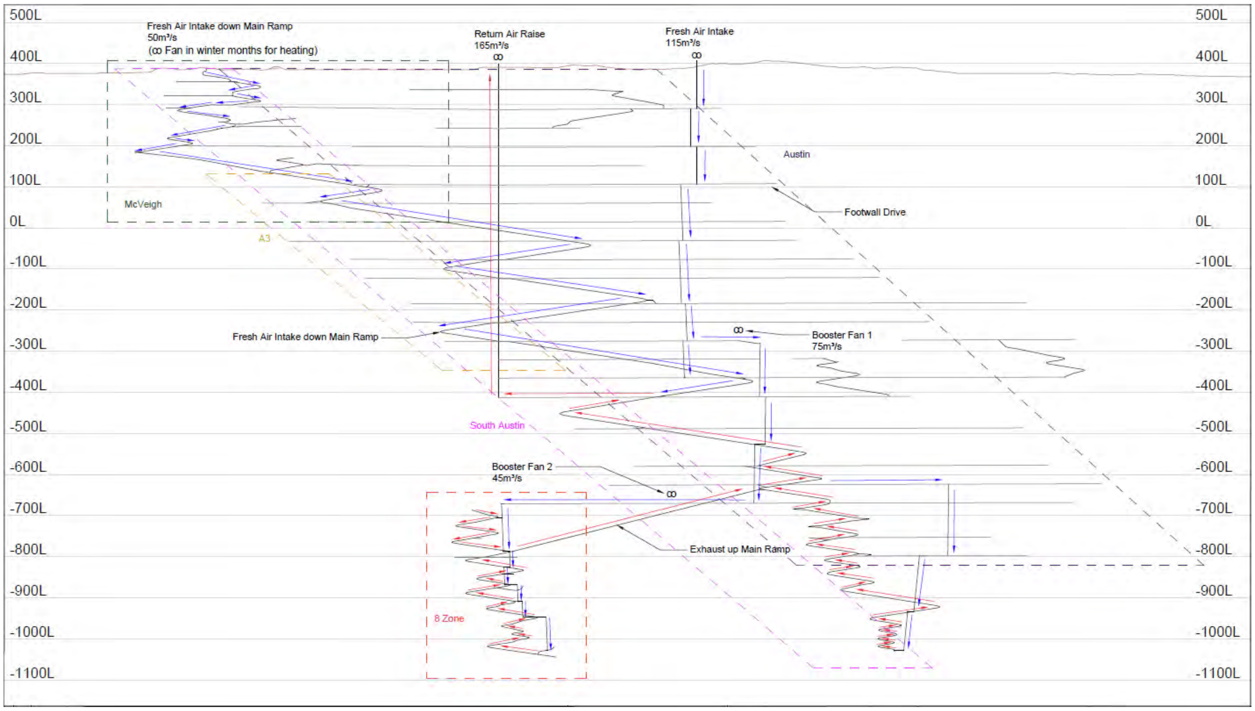
- 0.063 m^3/s/kW (100 cfm/bhp) for diesel powered equipment; and
- 0.025 m^3/s/kW (40 cfm/bhp) for battery/electric powered equipment.

Primary haulage equipment such as trucks and loaders assume an engine utilization of 85%, while auxiliary and drilling equipment assume an engine utilization of 15%.

16.9.1.2 General Arrangement

The proposed ventilation system consists of a push pull system whereby the main fan mounted on the surface ventilation raise will push fresh air into the mine, and an exhaust fan will be installed at surface on the ventilation segment of the haulage shaft pulling air out of the mine. The decline will also act as fresh air intake, with a fan utilized in the colder months to mix heated air required to achieve a +1.5°C intake air temperature. The main ramp will serve as a mixed air way and for primary egress of equipment and labour. Fresh air raises will extend down from surface as a series of raise-bored excavations and will be equipped with ladder ways to provide secondary egress. Figure 16-25 illustrates the proposed ventilation network at Madsen.

Figure 16-25: Mine Ventilation Process Flow Diagram



Source: JDS (2019)

The ventilation controls including regulators, man-doors and control doors that are required at each level will be installed on the fresh air raise access and shaft access, to control the airflow drawn from intake raise and the ramp access.

16.9.1.3 Airflow and Fan Selection

The calculation of ventilation requirements for the mine was based on:

- Mobile equipment fleet and mining activities in the mine;
- Underground fixed facilities such as service bays, pump stations, etc.;
- Inactive areas that need nominal airflow;
- Haulage routes with production equipment including trucks;
- Personnel working underground; and
- An estimated airflow leakage factor.

For sizing the main underground infrastructure, peak ventilation demand is first calculated, followed by the airflow requirements at individual ventilation phase or milestone. The peak ventilation demand for the Madsen mine is 160 m^3/s (340 kcfm) based on the equipment fleet, and 161 m^3/s (341 kcfm) based on work place activity (bottom-up approach). The bottom-up ventilation demand is taken as the final driving factor for the ventilation demand, due to the higher airflow requirement.

The main fan duty points respective to ventilation requirements were determined using Ventsim™ modeling software. The mine requires an intake fan and exhaust fan on surface, and two underground booster fans at peak ventilation demand phase. In addition to the estimated losses used in the Ventsim model, a 10% contingency factor for airflow (Q) and 21% contingency factor for pressure were also incorporated to size the main fan motors. To allow for different flow requirements at the different ventilation phases, the main intake and exhaust fans will be installed with Variable Frequency Drives (VFD's).

Table 16-11: Fan Duty Points at Various Stages of Mine Life

Location	Mine Phase	Mine Head (kPa)	With Pressure Losses (kPa)	With Pressure Losses & Contingency (kPa)	Airflow Quantity per Fan (m^3/s)
Main FAR	Early Production	0.57	0.94	1.14	105
	Steady State Production	0.66	1.03	1.24	110
	Maximum Mining Extents	0.87	1.24	1.50	115
Main RAR	Early Production	0.86	1.35	1.47	155
	Steady State Production	1.30	1.80	2.18	165
	Maximum Mining Extents	1.45	1.95	2.36	165
Booster Fan - 320L	Early Production	N/A	-	-	-
	Steady State Production	0.80	1.17	1.42	58
	Maximum Mining Extents	1.10	1.47	1.78	75
Booster Fan - 810L	Early Production	N/A	-	-	-
	Steady State Production	N/A	-	-	-
	Maximum Mining Extents	1.65	2.00	2.45	45

Source: JDS (2019)

The auxiliary fans for ramp development, level production and fixed facility ventilation were sized and the number of such fans, with critical spares, for each year during the life of mine was determined. 110 kW fans were sized for ramp ventilation. There are currently six 75 kW fans onsite which will be utilized for level development, and 11 kW fans were sized for stope and miscellaneous local ventilation.

16.9.1.4 Ventilation Phases

Based on the development and production sequence, four ventilation phases are identified in the life of mine plan:

- Pre-Production Phase (Year -1);
- Early Production Phase (Year 1);
- Steady State Production Phase (Year 3); and
- Maximum Mining Extents Phase (Year 6).

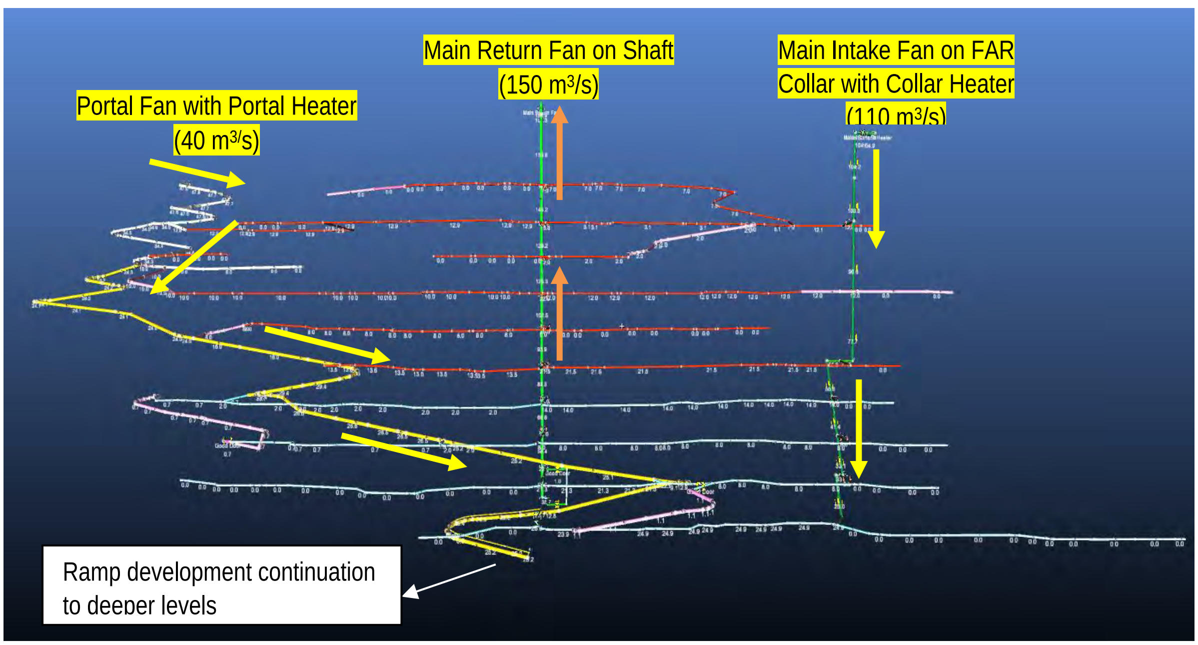
Pre-Production Phase

During the pre-production phase (Year -1), development crews are operating on the top levels. Decline development fans will operate until the first breakthrough of the main intake raise is made. Once the top levels are connected to the main intake raise, the main intake fan will be installed with main air heater at the raise collar. The main return fan and portal intake fan will be installed at the end of pre-production phase.

Early Production Phase

Existing upper levels are fully slashed to their final dimensions, ore production is in progress and the primary ventilation network is established. Figure 16-26 depicts the approximate development extents and the modelled airflows. The period assumed is Year 1. All the main surface fans are installed and operating from this time frame through the end of the mine life. Ramp development crew advance to the bottom levels, and the level development crews start slashing each level as the ramp reaches those levels. The airflow requirement for this phase is estimated to be 148 m^3/s.

Figure 16-26: Early Production Phase Ventilation Model



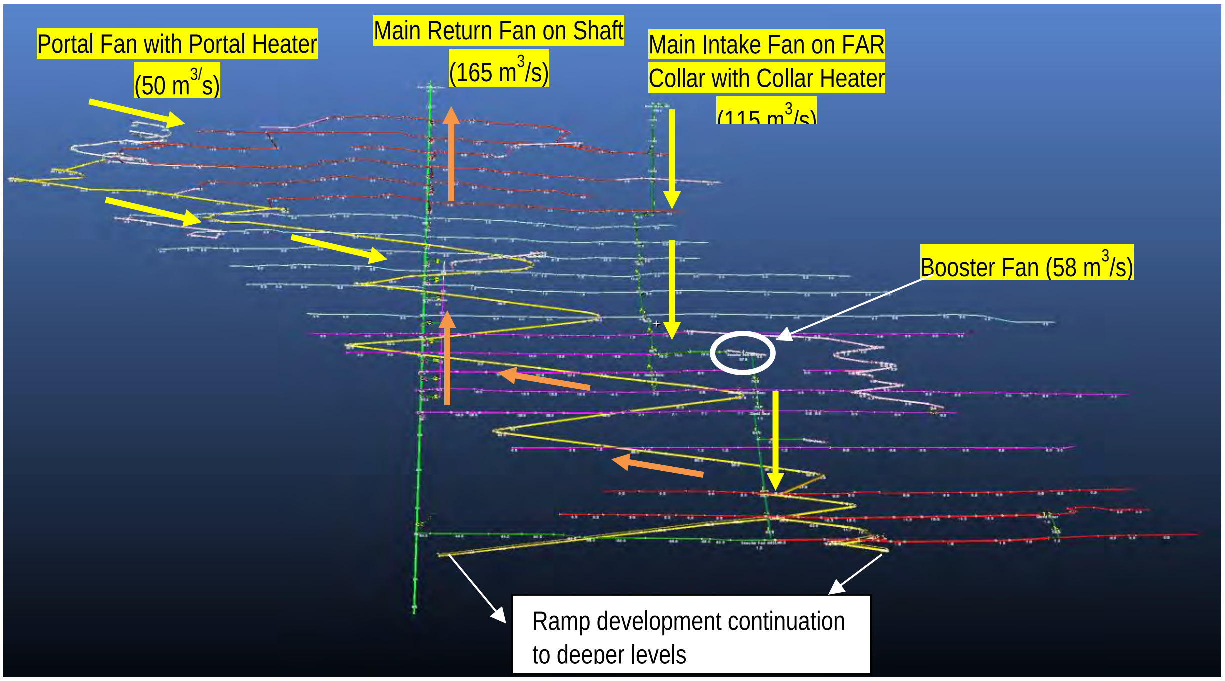
Source: JDS (2019)

Steady State Production Phase

Steady state production capacity is achieved, but the location of the respective mining activities varies according to the development and production sequence. This phase is modeled at a timeframe when the mine reaches steady state production capacity (Year 3) of 800 t/d, but with prior to development of deeper levels of the mine. To provide airflow for production and ramp development to the bottom levels, a booster fan will be installed on 14 Level.

The mine during this phase is shown in Figure 16-27. The mine development activities continue to reach the lower levels of the mine. The airflow requirement for the mine at this stage is estimated at 160 m^3/s. To account for auto-compression and density changes, some additional airflow will be pushed by main fans (165 m^3/s).

Figure 16-27: Steady State Production Phase Ventilation Model - Years 3-4

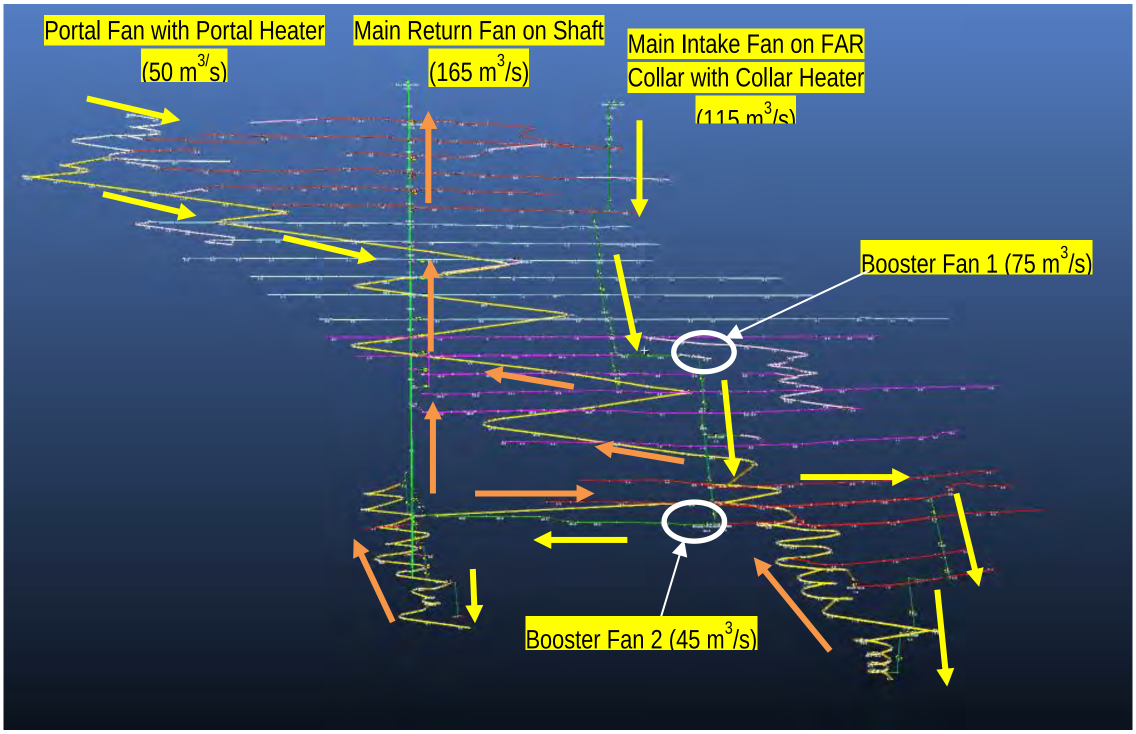


Source: JDS (2019)

Maximum Mining Extents Phase

During this phase more airflow needs to be provided to the bottom most levels of the mine to support the mining activities in 8 Zone during and after Year 6 and 7. The second booster fan will be installed underground at 21 Level, and the booster fan on 14 Level will operate at a maximum duty point. During this phase ramp development is complete, and nearly all the levels are developed and connected to the ventilation network. Depending on the production sequence, multiple levels may be in production at a given point of time. The total airflow requirement for the mine at this stage is modelled at 157 m^3/s, however the main fans will provide an airflow of 165 m^3/s to account for auto-compression. The model for this phase is shown in Figure 16-28.

Figure 16-28: Maximum Mining Extents Phase Ventilation Model - Years 6-7



Source: JDS (2019)

16.9.2 Mine Air Heating

The Madsen mine is located in a climatic zone where freezing temperatures are experienced for extended periods of time. It's generally advised that mine intake air be heated above the freezing point of water to:

- Prevent freezing of water in utility lines;
- Prevent buildup of ice on the surface of the shaft / raise which could constrict the airflow; and
- Prevent the freezing of water on any ramp/decline surfaces.

The effect of warming the air for the comfort of the workforce is secondary, since the air temperature rises as the air descends into the mine, due to friction, auto-compression, etc. Mine air is planned to be heated to a temperature of +1.5°C. The pressure, temperature, and humidity of the ambient air flowing into the mine will vary seasonally and diurnally. These variances typically result in the transfer of heat to or from the intake shaft/raise walls and are damped by a thermal flywheel effect. Thus, average intake conditions were taken as the basis for the estimation of heating requirement for Madsen. These are tabulated in Table 16-12 below. Based on the typical annual weather conditions, mine air heating will be required from November through March of every year, with a possible requirement during months of April and October.

Table 16-12: Historical Climate Data - Red Lake Region

Month	Temperature		
	Average High (°C)	Average Low (°C)	Average (°C)
January	-14.0	-25.1	-19.6
February	-9.0	-21.4	-15.2
March	-1.1	-14.2	-7.7
April	8.2	-4.5	1.9
May	16.6	3.4	10.0
June	21.2	9.2	15.2
July	23.8	12.3	18.1
August	22.4	11	16.7
September	15.3	5.4	10.3
October	7.7	-0.4	3.7
November	-2.7	-10.0	-6.4
December	-11.3	-21.1	-16.2

Source: https://www.eldoradoweather.com/canada/climate2/Red%20Lake.html (2019)

A minimum expected temperature is used as a basis to size the heater capacity. The minimum expected temperature is the absolute temperature expected. This reference temperature, based on the historical climate data, is set to -30 °C. Hence, the heaters are designed to provide sufficient heat to raise the full design airflow rate by approximately 32 °C. As an operational consideration, in the event that the outside ambient temperature falls below -30 °C, the main fans can be adjusted with their VFDs to reduce the total mass flow so that a control temperature of +1.5 °C is maintained.

Propane is the assumed heating fuel used in heating systems, both at the portal and the raise collar. Estimated average monthly propane consumption occurs in Year 4 and is shown in Table 16-13.

The main decline of the mine pushes fresh air at an average rate of 55 m^3/s through the mine life. The portal heater will need to have a capacity of 8.2 MMBtu/hr. The main intake raise fans will push an airflow of 115 m^3/s during the peak ventilation phase. Based on the absolute minimum intake conditions, the portal heater will need to have a capacity of 17.1 MMBtu/hr.

Table 16-13: Mine Air Heating Monthly Propane Consumption (Year 4)

Month	Average Ambient Temperature (°C)	Desired Temperature Rise (°C)	Unit Energy Required (Btu/hr)	Propane Consumption (L)
January	-19.6	21	13,456,888	437,769
February	-15.2	17	10,650,712	346,480
March	-7.7	9	5,867,712	190,875
April	1.9	-	-	-
May	10.0	-	-	-
June	15.2	-	-	-
July	18.1	-	-	-
August	16.7	-	-	-
September	10.3	-	-	-
October	3.7	-	-	-
November	-6.4	8	5,038,361	163,904
December	-16.2	18	11,288,479	367,228

Source: JDS (2019)

16.9.3 Water Supply

The active underground workings are currently supplied service water from the Process Water Pond. Mine plan equipment requirements were reviewed to determine peak service water requirements of 120 gpm. Water will be distributed via the mine ramp in 6” lines, which will feed the main distribution lines on the levels.

16.9.4 Dewatering

The mine is currently dewatered to 5 Level below the shaft collar (approximately 5 Level). Underground water volumes and inflow rates were determined by Lorax and three water sources were identified:

- Water associated with existing flooded open and backfilled voids (1.3 Mm^3);
- Operational water associated with mining activities such as, drilling, dust suppression and backfill decant water (1,108 m^3/day); and
- Groundwater recharge from surrounding bedrock (800 m^3/day).

A dewatering system has been designed so that it’s initially capable of pumping 3,900 m^3/day for the pre-production period, then as the system is extended and the existing flood water is removed the maximum required dewatering rate is reduced to 2,000 m^3/day.

Initial dewatering will be done via the shaft utilizing a Grundfos 250 HP deep well submersible pump. As dewatering progresses permanent pump stations will be setup on the 4, 8, 12 and 16 Levels. Permanent pump stations utilize a VFD equipped Technojet 200 HP pump. Once 17 level is reached the submersible pump will be extended as far down the shaft as possible. It has been assumed for this study that 19 level can be dewatered by the shaft submersible pump.

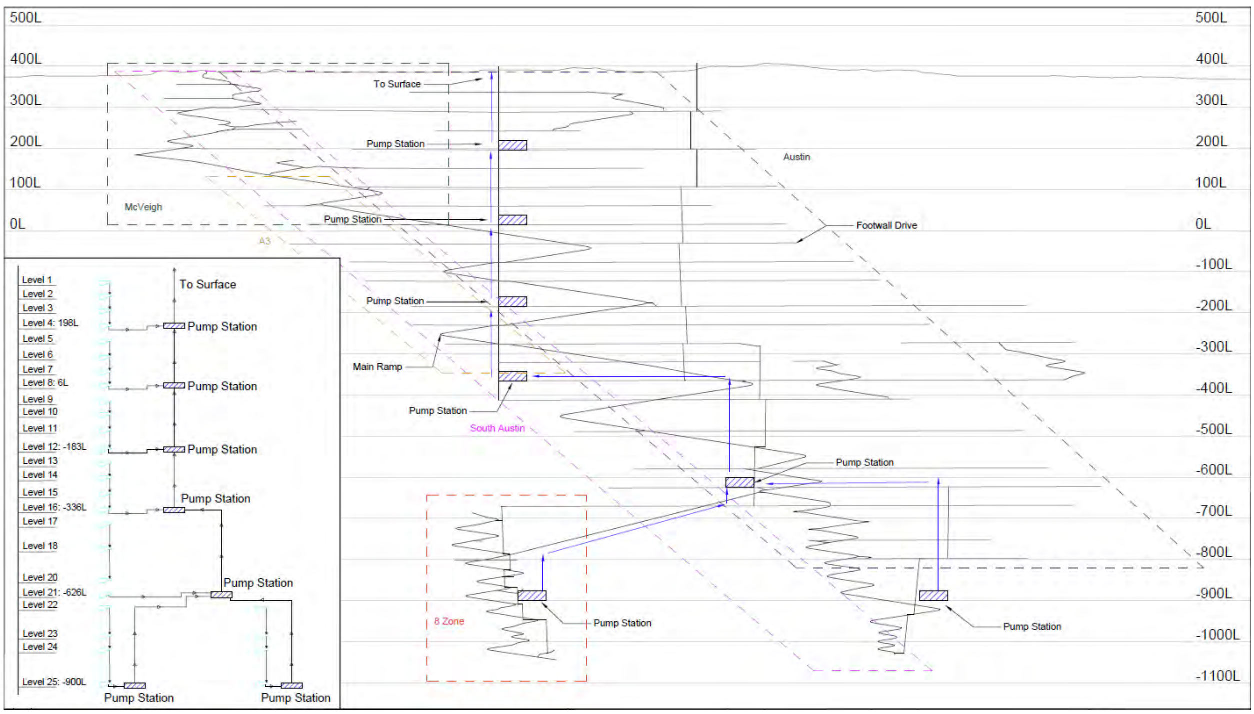
Existing Levels 20 through 27 will be dewatered with skid mounted pumping stations as they are developed and connected to the ramp system. Additional permanent pumping stations will be constructed on the 21 Level, and in proximity to the lower South Austin and 8 Zones.

On the levels, water will drain back to sumps where it's collected and pumped to a central sump on the level. Pumps on the level will be a combination of air operated Wilden pumps and electric 15 HP and 30 HP pumps. Water will be gravity fed via drain hole to the nearest permanent pump station.

In decline development, water will be initially picked up by a 15 hp pump and fed to a skid mounted mobile pumping station equipped with 2x 60 HP pumps which will feed back to either a level sump or main pump station.

The dewatering single line diagram is shown in Figure 16-29.

Figure 16-29: Dewatering Single Line Diagram



Source: JDS (2019)

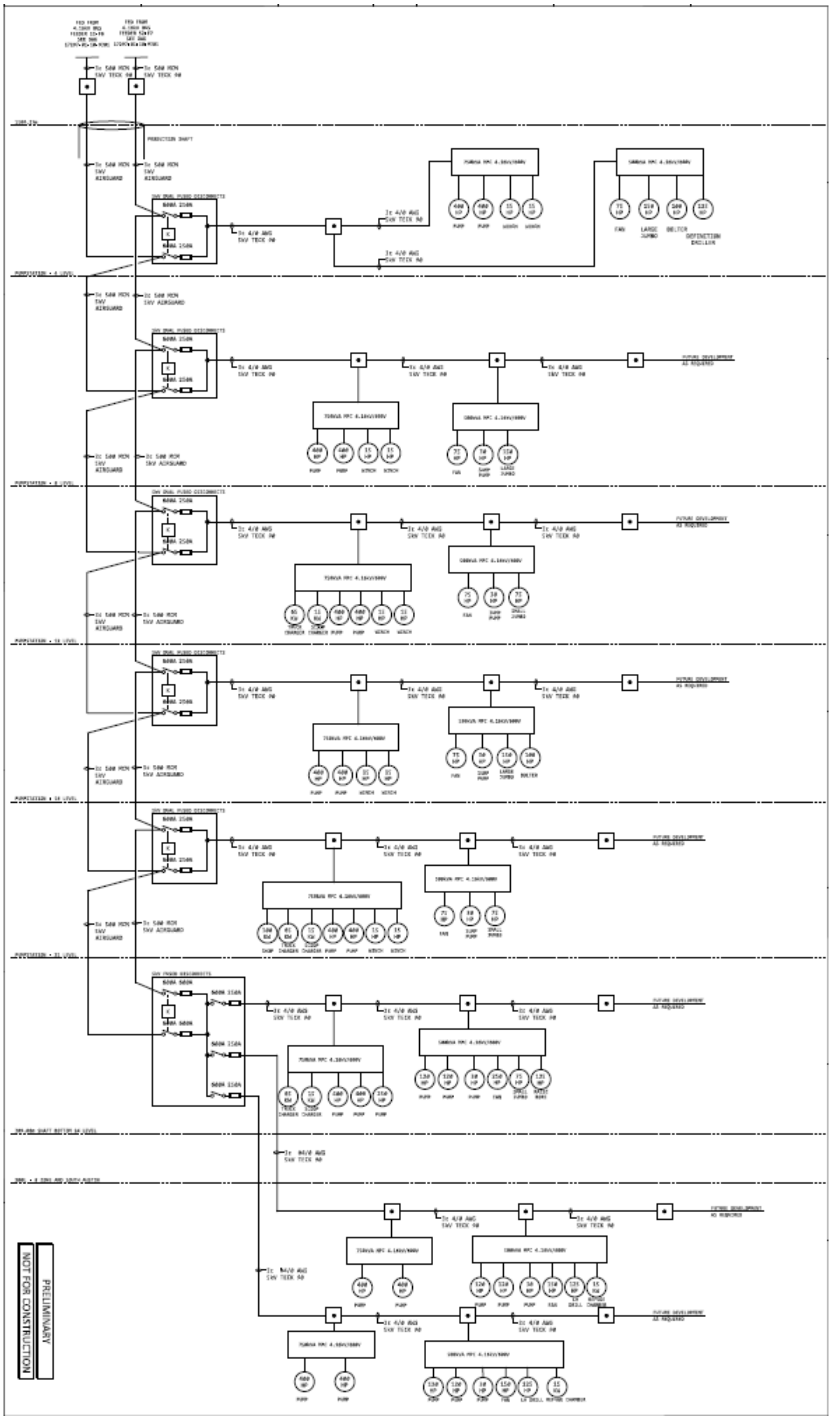
16.9.5 Electrical Distribution

Two existing 2300 V shaft feeders will be maintained to supply power for existing lighting and miscellaneous loads down as far as 16 Level (if required). The existing 600 Volt distribution entering the Madsen Portal will also be maintained.

A shown in Figure 16-30, two new 4160 V, 500 MCM feeder cables will be installed in the existing shaft to supply sufficient power for new mining development down to the South Austin and 8 Zone deposits as well as dewatering and ventilation equipment at various levels. These feeders will be connected to two feeder circuit breakers located at the new main substation switchgear building. 5 kV rated junction boxes will be installed at the sub-collar area to transition from standard Teck 90 armoured cable in tray to the vertical shaft cables.

Interlocked disconnect switches will be installed at 4 Level, 8 Level, 16 Level, and 21 Level to be able to select between one of the two shaft feeders to supply the loads at that level. 750 kVA and 500 kVA rated portable mine power centres will be connected and utilized as needed for dewatering, ventilation and mining loads as indicated on the single line drawing. Power cabling to adjacent levels will be accomplished via level to level boreholes.

Figure 16-30: Underground Electrical Distribution Single Line Diagram



Source: Nordmin (2019)

16.9.6 Mine Communications

An underground network of leaky feeder communications will be installed and expanded at depth. Mobile equipment operators, light vehicles, and supervisors will be equipped with hand-held radios to communicate with personnel on surface. Communication protocols will be used to ensure safe travel on the ramps and levels.

16.9.7 Compressed Air

Newer mobile mining equipment often has built-in air compressors and does not need to be connected to the mine compressed air system. However, compressed air will be required by certain mining equipment including jackleg drills and stopers, slushers, air winches, ANFO loaders and shotcrete sprayers. Peak compressed air requirements are estimated to be 6 kcfm.

Compressed air will be distributed via a 4” line in the ramp to the working levels where it will be carried in 2” lines.

16.9.8 Explosives Storage

Blasting products will be stored in underground explosive storage magazines with bulk explosive and detonators having their own magazines, as per regulations. The blasting crews will pick up the estimated quantities of explosives required for each shift using explosives transport vehicles and deliver those explosives to working faces and explosives-loading equipment underground. Excess explosives and accessories will be returned to the secure powder magazine every shift. All explosives and detonators in and out of the magazines will be documented as per Ontario explosives regulations.

16.9.9 Equipment Maintenance

Major maintenance will be performed on surface in the maintenance shop discussed in Section 18.6.

Small mobile equipment and trucks hauling will refuel as required on surface. A mobile fuel and lube truck will be servicing all mobile equipment remaining underground.

16.9.10 Mine Safety

Self-contained portable refuge stations will be provided in the main underground work areas. The refuge chambers are designed to be equipped with dedicated fresh air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers doors are sealed to prevent the entry of gases. The portable refuge chambers will be moved to the new locations as the working areas advance.

Primary mine access will be through the main portal and decline. Secondary emergency egress will be through the fresh air raises on each level and connecting to surface. The raises will be equipped with all the necessary equipment to provide secondary egress.

A fully equipped mine rescue team will be available 24 hours a day, 365 days per year to respond to emergencies. Currently this coverage is provided by a small Pure Gold team and supported through a mutual aid agreement with Goldcorp Red Lake Mines Ltd.

16.10 Unit Operations

16.10.1 Drilling

There are six principal drilling machines selected for Madsen. Each has their own primary use:

- Two-boom electric-hydraulic jumbos for large dimension development and production headings;
- Single-boom electric-hydraulic jumbos for level slashing and mechanized cut and fill;
- Mechanized rock-bolters for ground support installation;
- Electric-hydraulic longhole drills for production;
- Jackleg and stoper drills for bolting in smaller cut and fill headings and raises, drilling safety bays, service drilling, ground support installation; and
- Bazooka drilling for infill or definition drilling.

Drilling productivities (metre drilled / percussion hour) were built up from first principles and industry standards. Jumbo drilling rates vary from 32 m/hr in small headings (1-boom) to 54 m/hr in large headings (2-boom), and longhole drill machines average 12 m/hr. Drilling productivities include penetration rates, reaming rates, repositioning, set-up / mark-up, and a 5% factor for re-drilling.

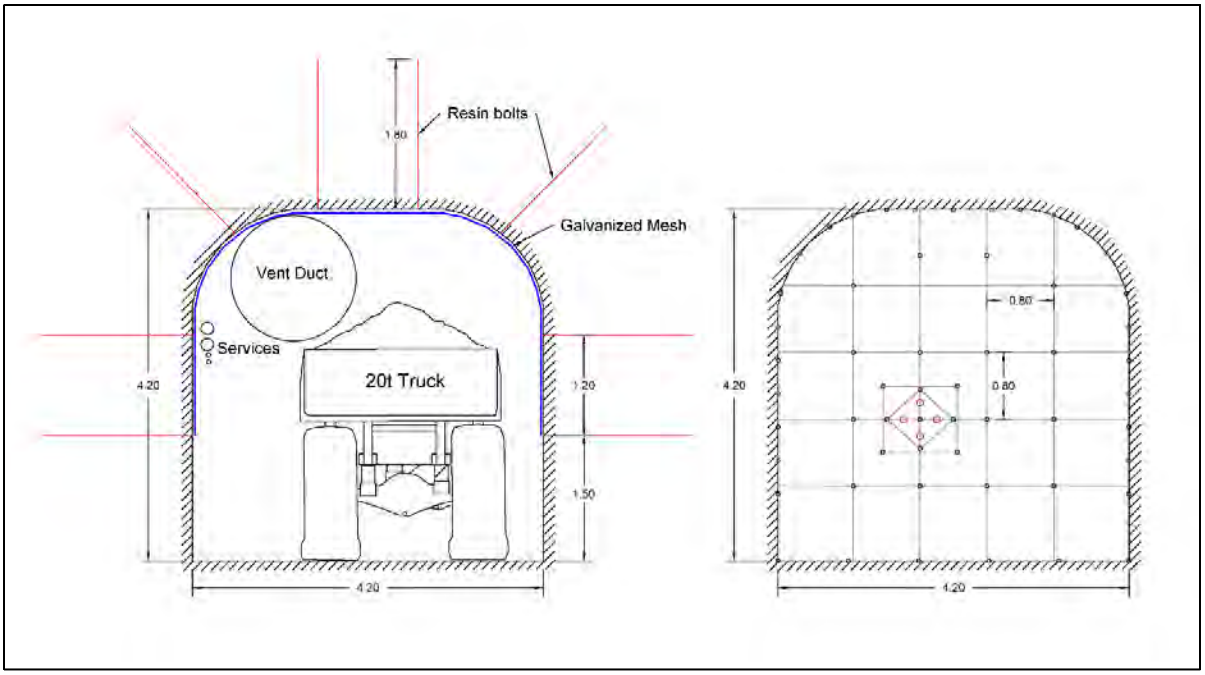
Smaller cut and fill headings, attack ramps and level slashing will be conducted by single-boom electric jumbo drills. The single-boom jumbos will be equipped with 4.3 m (14 ft) drill steel and will advance at an average rate of 3.8 m/d per machine.

Capital ramps, cross-cuts, footwall drives and other large headings will be developed by two-boom electric jumbo drills. Jumbos will be equipped with 4.88 m (16 ft) drill steel and will advance at an average rate of 4.0 m/d per machine.

Approximately 5% of all holes are anticipated to require re-drilling.

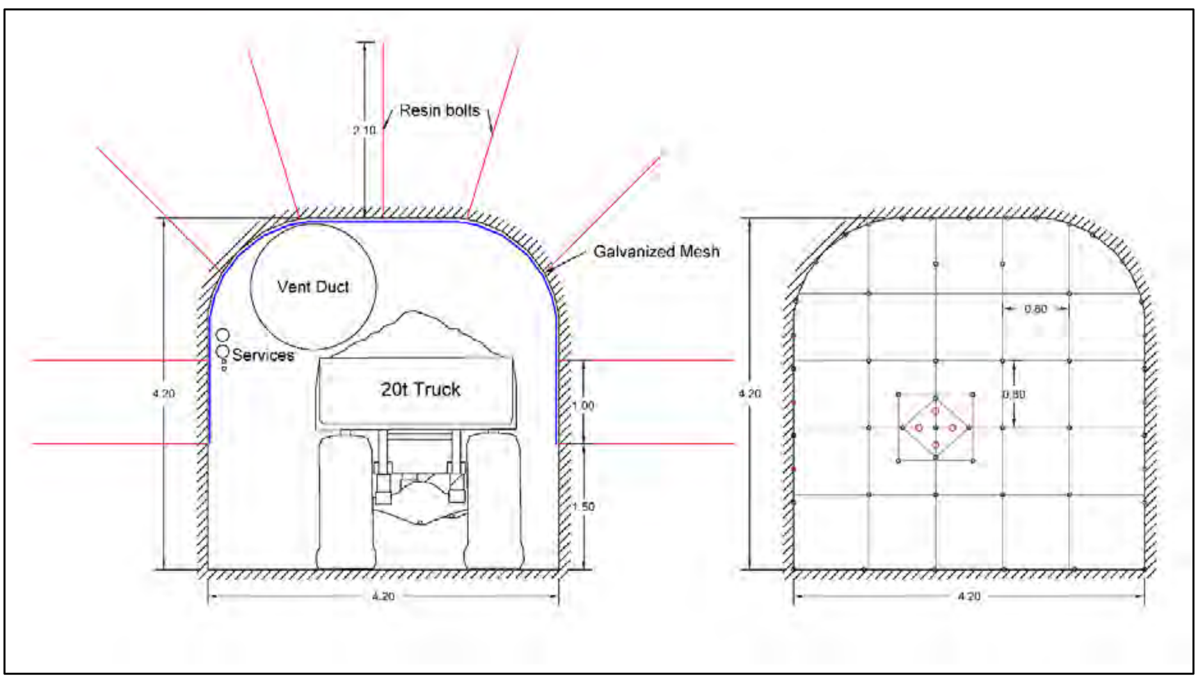
Typical heading profiles and jumbo drill patterns for Madsen development, LH sub-levels, and MCF mining are depicted in Figures 16-31 to 16-37.

Figure 16-31: 4.2 m W x 4.2 m H Development



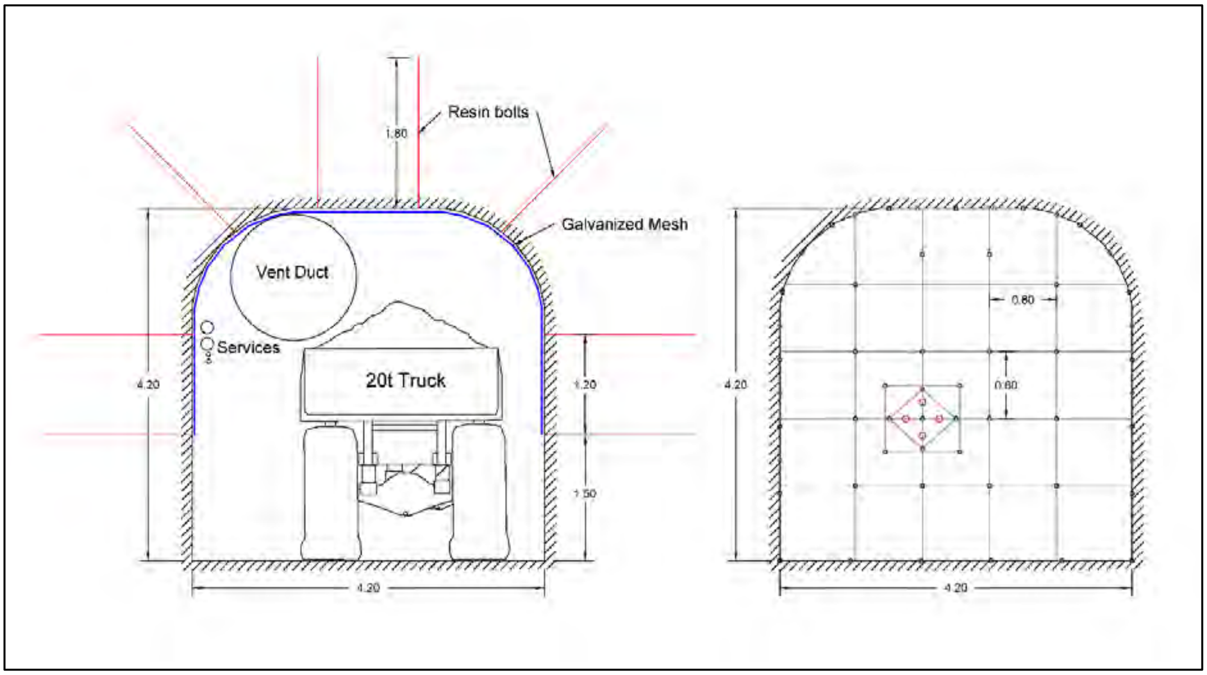
Source: JDS (2019)

Figure 16-32: 4.2 m W x 4.2 m H 8 Zone



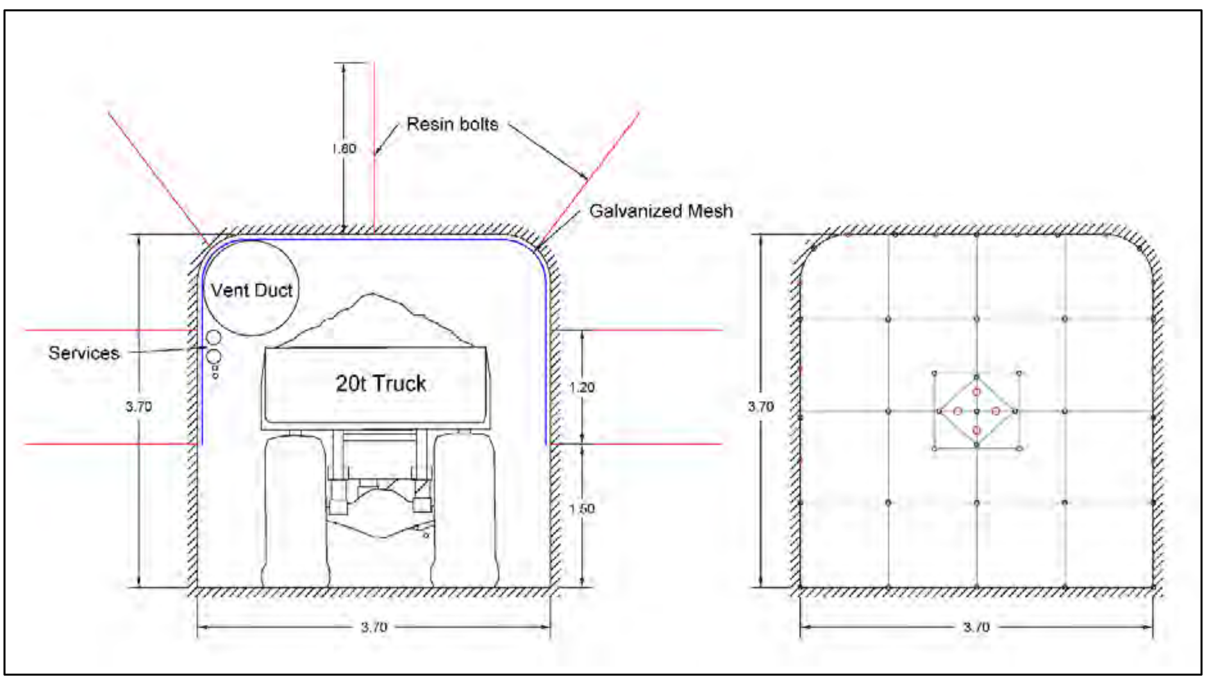
Source: JDS (2019)

Figure 16-33: 4.2 m W x 4.2 m H 8 Zone, Additional Ground Support Area



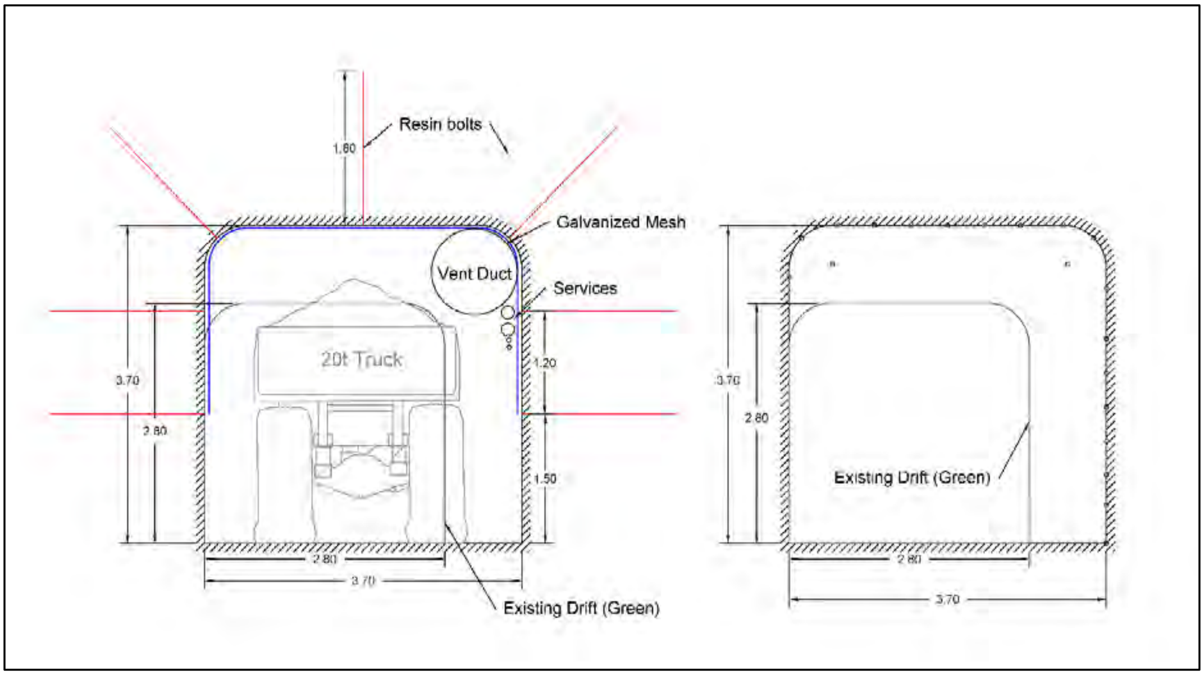
Source: JDS (2019)

Figure 16-34: 3.7 m W x 3.7 m H Development



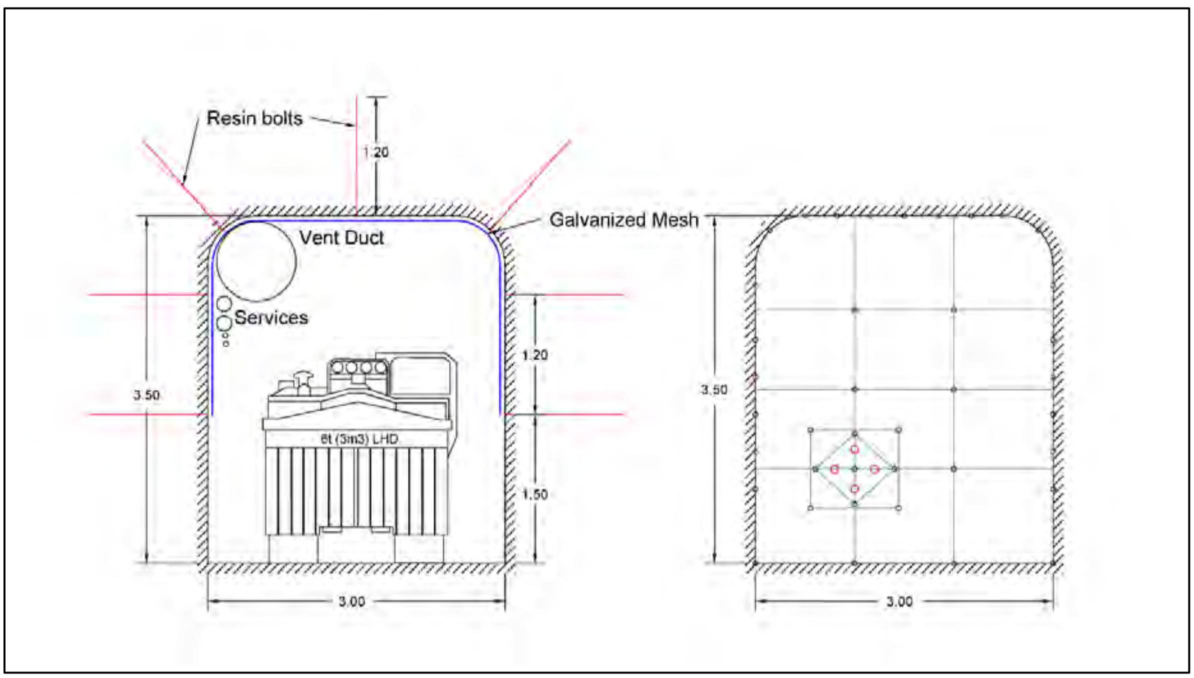
Source: JDS (2019)

Figure 16-35: 3.7 m W x 3.7 m H Slashing



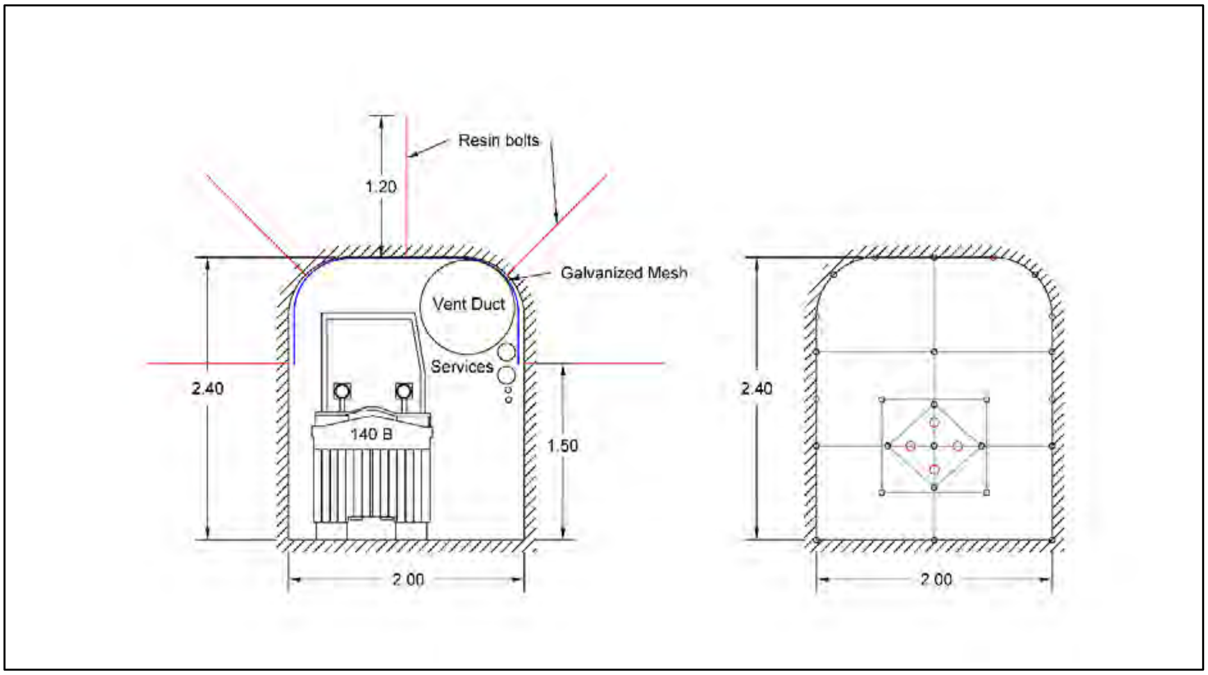
Source: JDS (2019)

Figure 16-36: 3.0 m W x 3.5 m H Development



Source: JDS (2019)

Figure 16-37: 2.0 m W x 2.4 m H Development



Source: JDS (2019)

16.10.2 Blasting

Blasting crews will be trained and certified for explosives use. Bulk or packaged emulsion will be used for production blasting and development rounds. Ammonium nitrate fuel oil (ANFO) explosives will be used where possible. Boosters, primers, detonators, detonation cord and other ancillary blasting supplies will also be utilized. Smooth blasting techniques may be used as required in headings, with the use of trim powder / bulk emulsion for loading the perimeter holes.

All blasting will be done at shift change. All personnel not clear from the underground will be required to be in a designated Safe Work Area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes.

Blasts will be initiated with Nonel LP detonators and paired with Pentex SB-8 boosters for lateral development rounds and Pentex AP-340 for LH stopes. To insure full detonation of explosives, loaded holes will be traced with Det Cord.

16.10.3 Ground Support

Ground support will be installed in accordance with specifications based on geotechnical analysis provided in Section 16.3 for the various rock qualities expected. Electric-hydraulic bolters, longhole drills, jackleg drills, and shotcrete spraying machines will be used.

Ground support will be installed post-mucking of the blasted drift. No additional development will be commenced in the heading prior to the installation of ground support. At no time will mine workers be under unsupported ground.

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with depth and active use of the heading. Discretion will be made by the development lead with support from the geotechnical engineer as to which ground support is required.

Regular pull tests will be conducted on-site to ensure adequate installation of rock bolts. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy.

Rock bolts and screens will be installed with a bolter machine. In smaller cut and fill drifts, jackleg drills will be used to install the required ground support.

Cable bolts will be installed in 3 and 4 way intersections and along sub-level LH sill drifts when required. Cable bolts may be installed shortly after the development behind the development crew as to maintain the advance rate of the drift. When intersections are located in poor ground conditions, shotcrete and cable bolts will be installed prior to development of the intersection. Cable bolts will be drilled with a longhole drill and installed from a scissor deck manually.

Shotcrete application is not expected to require quantities to justify an on-site shotcrete batch plant, and pre-mixed bags of shotcrete have been budgeted for the project. Mobile shotcrete machines will apply shotcrete to the underground workings as needed.

16.10.4 Mucking

Two LHD sizes are proposed for the Madsen project. Generally, the large LHD will be used to perform the task when possible to maximize efficiencies. Smaller units smaller units will be used in areas the larger units do not fit.

The large LHDs have a nominal 3.0 m^3 (6 t) bucket capacity. The large LHD will be will be used for primary ramp development and excavation of other larger development headings, longhole stope mucking, loading trucks from re-mucks, and tramming material from the ore pass to the hoisting system. The small LHDs have a 0.7 m^3 (1.3 t) bucket capacity. The small LHD will be used for material handling in smaller headings such as those for cut and fill mining.

LHDs will typically muck a blasted round to a nearby re-muck bay in order to clear the working face prior to ground support installation. Rock temporarily stored in the re-muck is then loaded into a haul truck. When available LHDs will direct load trucks from the face. Where un-mineralized development headings are proximal to stope voids requiring rock fill, the LHD will tram the material to the stope void. LHDs will tram material to a maximum of 230 m to either a re-muck, directly into a haul truck or a stope void.

In LH stopes, an LHD equipped with remote control will be utilized in order to keep personnel away from unsupported ground. LHDs will tram the material to a nearby re-muck bay.

16.10.5 Hauling

Muck will be hauled using 20 t capacity underground haul trucks. In most cases, trucks will be restricted to loading at re-muck stations due to the increased back height requirements for LHDs to load over the side of the truck box. Trucks will haul muck either to surface or a skip loading pocket, depending on the mining production schedule. Muck transported to surface will be re-handled by surface equipment for transport to the crusher or rock storage.

16.10.6 Backfill

Backfill strategies will consist of rock fill and hydraulic fill, both structural and non-structural hydraulic fill. Rock fill will be used in longhole stope voids and hydraulic fill will be used in cut and fill.

The purpose of the use of non-structural hydraulic fill is to fill stope voids that cannot be practically filled with rock, to utilize tailings (thereby decreasing tailings storage requirements), as well as to provide an adequate working floor in a relatively shorter time frame.

Life of mine backfill distribution by tonnage is estimated as follows:

- Rock Fill 45%;
- Hydraulic Fill 43%; and
- Structural Hydraulic Fill 12%.

16.10.6.1 Rock Fill

Rock fill will be used to backfill longhole stopes and in some cases MCF stopes. Rock fill consists of blasted un-mineralized rock from the underground workings and is not subjected to any crushing, screening, or cement addition. Rock fill will be delivered to stopes by either truck or LHD depending on location, back height, and activity. Whenever possible un-mineralized rock will be directly hauled from the blasted heading or remuck to an empty stope, this will minimize re-handling of material brought to surface for temporary storage. It’s anticipated that rock fill requirements will be fulfilled by daily development in un-mineralized rock. Any additional rock fill requirements that are not fulfilled by underground sources will be back-hauled from the MRMF.

Hydraulic backfill is discussed in Section 16.8.

16.11 Shaft and Hoisting

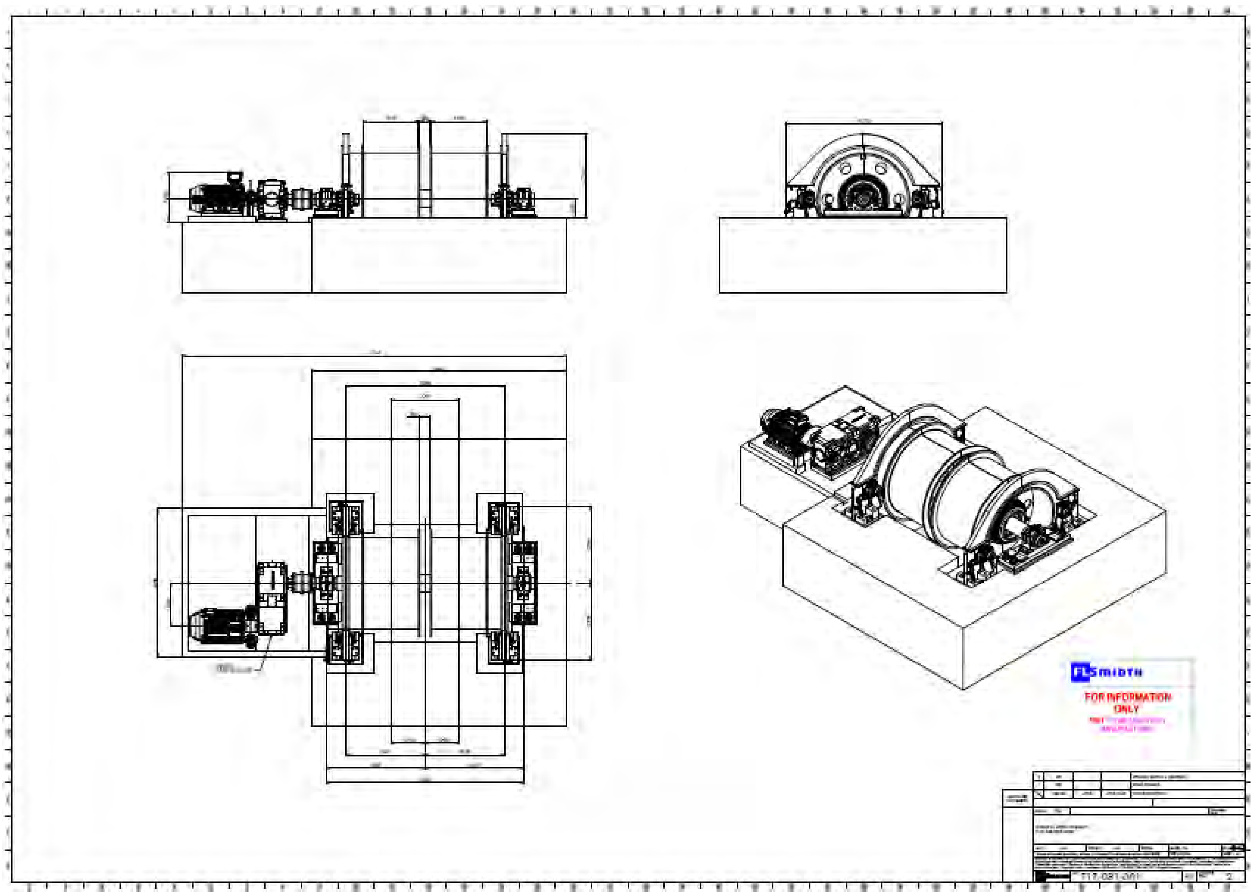
The existing shaft, headframe and hoisting system last operated in a production capacity in 1999 and last operated in an exploration support capacity in 2013. Due to periods of inactivity, some components will be replaced, and some components updated with modern technologies. The hoisting of ore and un-mineralized rock from depth to surface will be done using skips filled at the loading pocket at 17 Level. The 17 Level infrastructure was selected due to the unknown condition of the shaft below. An estimated combined 1,200 t/d of ore and un-mineralized rock is the design basis of the infrastructure repair and upgrade.

16.11.1 Hoist and Hoist House

Due to the lack of suitability of the hoist and mechanical systems, a new hoist is recommended and subsequently costed. Opportunity does exist to refurbish the existing equipment upon further inspection and testing. The recommended hoist is a 3 m diameter double drum, single clutch hoist supplied by FL Schmidt. This hoist is able to handle the hoisting demands required for the proposed production and development rates. The proposed hoist is shown in Figure 16-38.

Due to the new hoist type and configuration, the current hoist house is not fit for purpose. A new hoist house has been detailed and included in the capital estimate. The new hoist house will contain all of the equipment necessary for the hoist operation.

Figure 16-38: Hoist General Arrangement



Source: Nordmin (2019)

16.11.2 Headframe

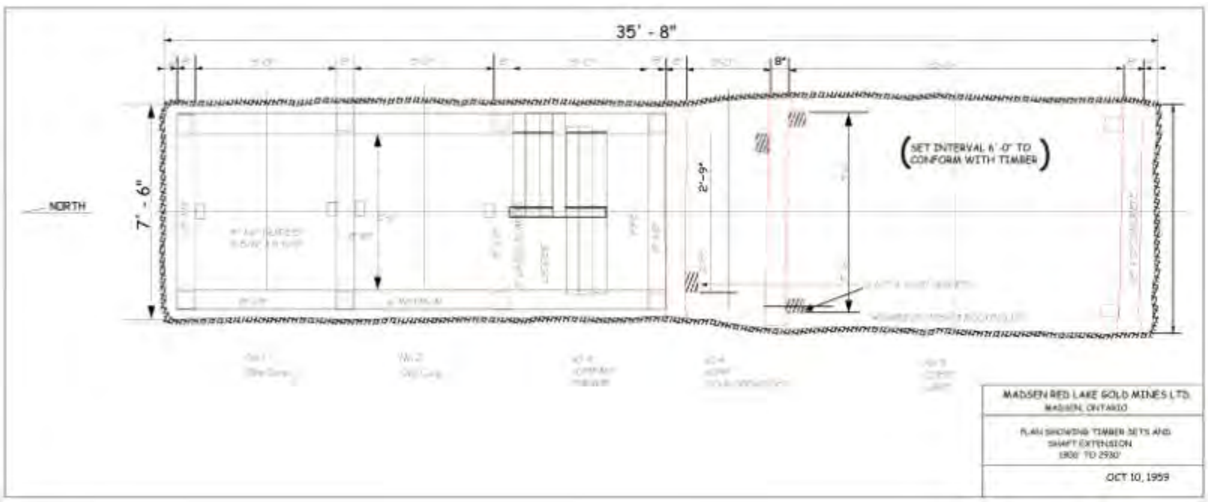
With some modifications, the current headframe is deemed structurally adequate to support hoisting and mining activities. Back legs will need to be installed on the hoist side of the headframe in order to manage the new force direction of the new hoist setup. Also, new head sheaves will need to be installed at the top of the headframe for the hoist rope to run through. Minor modifications to the dump system on surface will also be required. This is to allow dumping of the 2 m^3 Kimberley type skip units.

16.11.3 Shaft

The Madsen Shaft is a timbered shaft arrangement that consists of five separate compartments – two skip compartments, one manway compartment, one counterweight compartment, and one cage compartment. The shaft is approximately 35.5 ft. by 7.5 ft. in dimension.

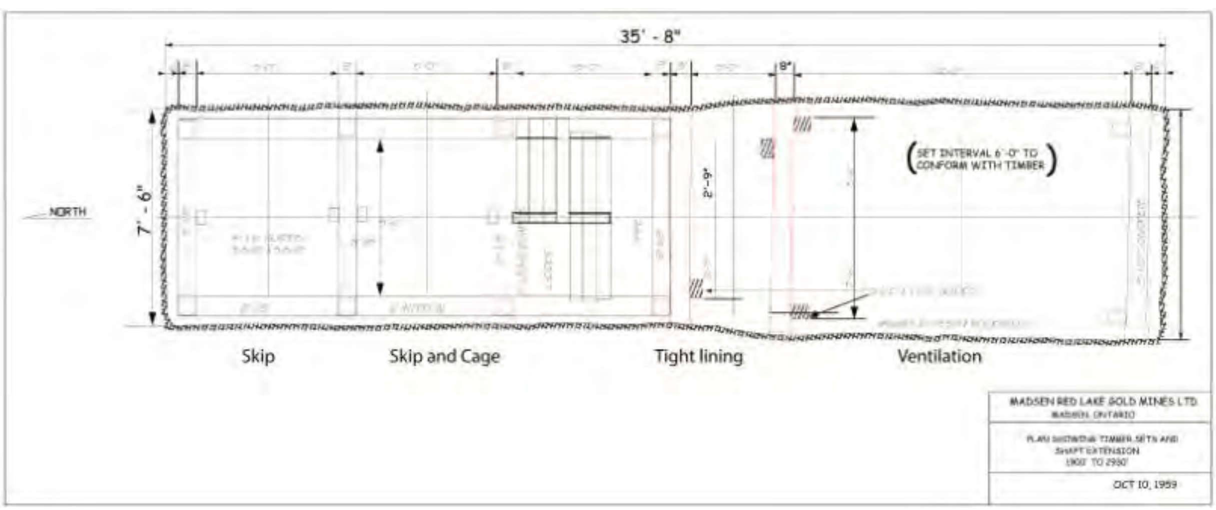
The proposed setup for the shaft will have one skip compartment, one cage over skip compartment, one manway compartment, and the other two compartments will be used for ventilation purposes. This will be accomplished by installing tight lining between the manway and previous counterweight compartment. The current and proposed shaft configurations are shown in Figures 16-39 and 16-40.

Figure 16-39: Existing Shaft Plan



Source: Nordmin (2019)

Figure 16-40: Proposed Shaft Plan



Source: Nordmin (2019)

As the shaft rehabilitation progresses downwards, the following tasks will be performed:

- Check ground and shaft conditions and remediate as required;
- Install tight lining;
- Check shaft guides and replace as necessary;
- Install new compressed air line;
- Install new process water line;
- Install new electrical cable; and
- Install new backfill line.

Once the shaft work has been completed, the hoist will be commissioned, and tested as required.

16.12 Mine Equipment

All underground mine equipment required to meet the life of mine plan is summarized in Table 16-14. Battery powered load and haul equipment has been utilized where possible to reduce ventilation requirements in the mine. For flexibility the first four haul trucks purchased are conventional diesel trucks, and the next five trucks required are battery powered. The small 1.3 tonne LHDs are also battery powered, and the 6.0 tonne tethered electric loader will be used at the load station of the hoisting system transferring muck from the pass to the load pocket.

Table 16-14: LOM Underground Equipment Fleet

Equipment	Avg	Peak	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Truck (20t/9.7m^3)	2	4	3	4	4	4	4	4	3	1	1	1	1	1	1	1
Truck (20t/9.7m^3) Battery	3	5	-	1	3	5	5	5	5	3	3	3	2	2	2	1
LHD (1.3t/0.7 m^3) Battery	7	9	4	8	9	9	8	8	8	5	5	5	5	5	5	3
LHD (6.0t/3.0 m^3)	2	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2
LHD (6.0t/3.0 m^3) Tethered Electric	1	1	0	0	0	0	1	1	1	1	1	1	1	1	1	1
Jumbo - 1 Boom	3	3	1	3	3	3	3	3	3	3	3	3	3	3	3	3
Jumbo - 2 Boom	4	5	3	4	5	5	5	5	5	3	3	3	3	3	2	1
Bolter	3	3	2	3	3	3	3	3	3	3	3	3	3	3	2	1
Longhole Drill	2	2	0	2	2	2	2	2	2	2	2	2	1	1	1	1
Small Explosives Truck	2	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Shotcrete + Transmixer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Jackleg / Stoper	29	36	0	13	25	26	34	36	36	36	36	36	36	36	36	36
Grout Pump	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Personnel Carrier	2	2	1	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel / Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	2	2	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Electrician Truck	2	2	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Utility Vehicle	4	5	0	4	5	5	5	5	5	5	5	5	5	5	5	5
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Telehandler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	2	2	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Supervisor Truck	4	4	2	4	4	4	4	4	4	4	4	4	4	4	4	4
Bazooka Drill (Infill Drilling)	1	1	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Slusher	12	18	0	6	10	10	15	15	15	15	15	16	18	18	18	18

Source: JDS (2019)

16.13 Mine Personnel

Mine operations personnel are scheduled to work 12 hour shifts on a 4 days on 4 days off roster (4&4). Other staff will work 10 hour shifts, 5 days a week. Tables 16-15 to 16-21 show personnel estimated. Quantities represent total personnel across all crews on and off site.

Table 16-15: Mine Management

Position	Rotation	Peak	Average	Pre-Production
Mining Manager	5x2	1*		
Mining Superintendent	5x2	1	1	1
Training Officer	5x2	2	2	2
Chief Geologist	5x2	1	1	1
Technical Services Manager	5x2	1	1	1
Maintenance Superintendent	5x2	1	1	1
Mine Shift Foreman/Shift Boss	4x4	4	3	4
Maintenance Supervisor/Shift Boss	4x4	4	3	4
Mine Clerk	5x2	1	1	1
Subtotal		**16**	**14**	**16**

Source: JDS (2019)
* Labour cost for Mine Manager carried in G&A

Table 16-16: Mine Operations - Drill and Blast

Position	Rotation	Peak	Average	Pre-Production
Jumbo Operator	4x4	32	26	17
Conventional Stope Miner	4x4	52	37	-
Production Drill Operator	4x4	12	11	9
Blaster	4x4	8	8	4
Subtotal		**104**	**82**	**30**

Source: JDS (2019)

Table 16-17: Mine Operations - Load and Haul

Position	Rotation	Peak	Average	Pre-Production
LHD Operator	4x4	44	37	22
UG Truck Driver	4x4	36	21	10
Subtotal		**80**	**58**	**32**

Source: JDS (2019)

Table 16-18: Mine Operations - Support Services

Position	Rotation	Peak	Average	Pre-Production
Electrician	4x4	8	8	8
Development Service	4x4	8	8	7
Ground Support/Bolter/Shotcrete	4x4	16	15	13
Tier 2 Utility Vehicle Operator	4x4	16	15	8
Construction Miner	4x4	8	8	8
Mine Helper	4x4	8	8	4
Subtotal		**64**	**62**	**48**

Source: JDS (2019)

Table 16-19: Mine Operations – Hoisting

Position	Rotation	Peak	Average	Pre-Production
Skip/cage Tender	4x4	8	6	-
Hoist Operator	4x4	4	3	-
Shaft Maintenance	4x4	2	1	-
LHD Operator (Remote from Surface)	4x4	4	3	-
Subtotal		**18**	**13**	**-**

Source: JDS (2019)

Table 16-20: Mine Maintenance

Position	Rotation	Peak	Average	Pre-Production
Maintenance Planner	5x2	1	1	1
HD Mechanic	4x4	16	16	11
Drill Mechanic	4x4	6	4	4
Electric/Hydraulic Mechanic	4x4	6	3	4
Lube / PM Mechanic / Light Duty Mechanic	4x4	4	4	4
Welder	4x4	4	4	4
Apprentice	4x4	4	4	4
Dry / Lapman / Bitman	4x4	2	2	2
Subtotal		**43**	**38**	**34**

Source: JDS (2019)

Table 16-21: Technical Services

Position	Rotation	Peak	Average	Pre-Production
Senior Mine Engineer	5x2	1	1	1
Mine Planning Engineer	4x4	2	2	2
Mine Ventilation / Project Engineer	4x4	1	1	1
Geotechnical Engineer	4x4	1	1	1
Sr. Mine Technician	4x4	2	2	2
Surveyor / Mine Technician	4x4	2	2	2
Surveyor Helper	4x4	2	2	2
Production Geologist	4x4	2	2	2
Geotechnical Technician / Sampler	4x4	4	4	4
Bazooka Driller (Infill Drilling)	4x4	4	4	-
Drillers Helper	4x4	4	4	-
Subtotal		**25**	**25**	**17**

Source: JDS (2019)

16.14 Mine Development Plan

Mine development is divided into two periods: pre-production development (prior to commercial production) and ongoing development (during commercial production). The objective of pre-production development is to provide equipment access to high grade areas and prepare enough stoping areas to support the mine production rate once commercial production commences.

Pre-production development is scheduled to:

- Continue development of the decline / ramp;
- Slash 2 Level for access;
- Develop access and establish initial stoping areas; and
- Begin mining ventilation raise.

During the pre-production period some ore will be mined through the development of the longhole sub level drifts. This ore will be stockpiled on surface and will serve as a buffer aiding a smooth ramp up of the mill production. The ore stockpile will peak at 13,400 t at the end of the pre-production period. Where possible blasted un-mineralized development rock will be hauled directly to a stope in the fill cycle otherwise will be hauled to surface to be stored in the MRMF. A summary of the annual surface stockpile size is in Table 16-22.

Table 16-22: Cumulative Annual Surface Stockpile Size

Stockpile	Unit	Peak	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Temporary Ore Stockpile	kt	13	13	5	5	5	2	2	2	2	2	2	2	2	-	-
MRMF	kt	1335	69	239	412	601	780	984	1130	1171	1193	1233	1269	1315	1331	1335

Source: JDS (2019)

The mine was sequenced so that as the ramp accesses each level only activities related to establishing the ventilation network occur on the level until the ventilation circuit is established. Once the ventilation circuit is established other development and stoping activities can begin on the level.

Once production begins capital and access development remains a priority, in order to access lower levels and the 8 Zone as early in the mine life as possible.

The Madsen hoisting system is planned to be commissioned for the start of year 4 in the mine plan. The No. 2 shaft will be rehabbed and refurbished in year 3 from surface to 17 Level where the loading station will be located. Haulage routing will change once the hoist is commissioned so that material hauled from 7 Level and above will still be hauled to surface. Material hauled from the 8 to 16 levels will be transferred to an ore or un-mineralized rock pass reporting to 17 Level to be hoisted to surface. Material hauled from below 17 Level will be moved using the ramp system to the loading station on 17 Level.

16.15 Mine Production Schedule

The criteria used for scheduling underground mine production were as follows:

- An average annual mill feed production rate of 292 kt/year (800 t/d) was scheduled;
- Provide enough production faces to support a daily mine production rate of an average of 800 t/d;
- Target the mining blocks with higher profitability in the early stages of mine life to improve project economics including prioritising access to the high grade 8 zone; and
- Minimize mobile equipment requirements by smoothing ore production and development.

Minemax iGantt™ scheduling software was used to optimize the mine production schedule by maximizing the Net Present Value (NPV), subject to constraints including maximum lateral development rates, maximum production rates, and extraction sequence. Scheduling constraints utilized in the software are listed below:

- 2,000 lateral metres per month including cut and fill production;
- 24,400 ore tonnes per month; and
- Ramp development at 5 m/day.

Other scheduling rates and productivities are listed in Tables 16-23 and 16-24.

Table 16-23: Development Productivity Rates

Development Type	Unit	Rate
Ramp / Decline	m/day	5.0
General Development	m/day	2.5
Rehabilitation	m/day	6.0
Slashing	m/day	10.0
Raisebore	m/day	2.5

Source: JDS (2019)

Drilling, blasting, mucking and backfill cycle times were added together to realize a total cycle time for each mine object, the result of which was imported to iGantt software for application to the mine schedule.

Table 16-24: Stoping Productivity Rates

Mining Method	Unit	Rate*
Longhole Stoping	t/day	70
Mechanized Cut and Fill	t/day	31
Conventional Cut and Fill	t/day	31

* Productivity includes backfill pour and curing if applicable
Source: JDS (2019)

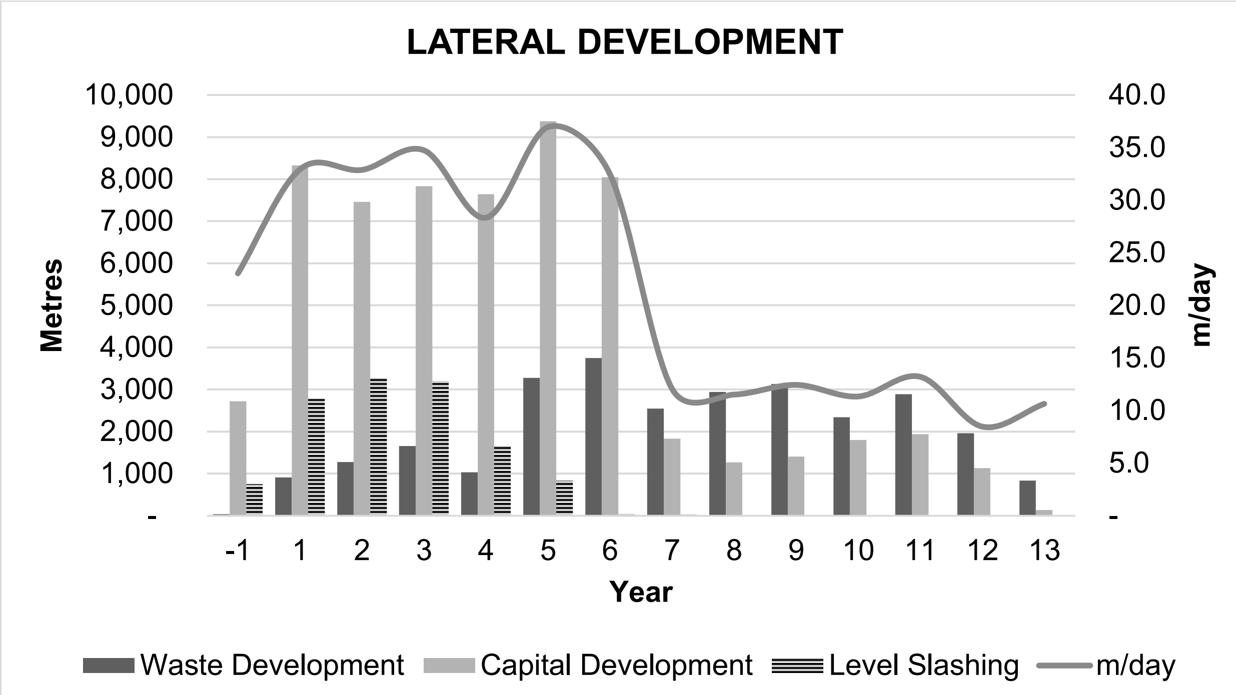
Final results of the iGantt schedule were organized such that physical metres, tonnes and ounces were broken down into different categories for direct use in the cost model. From the final schedule, cost model requirements including items such as the mining fleet, manpower, consumables, ventilation, pumping, and power were determined and used to develop costs from first principles.

Detailed mine planning and scheduling was completed on a monthly basis for the life of mine and is summarized annually in this report.

16.15.1 Mine Development

Total underground capital and sustaining lateral development are 60,887 m and 28,567 m, respectively. Underground capital development averages 7,343 m/year or 21.9 m/day from Year -1 to Year 6 when a majority of the underground capital development is completed. Lateral development averages 2,040 m/year or 6.2 m/day over the life of mine and is shown in Figure 16-41.

Figure 16-41: Overall Lateral Development Schedule

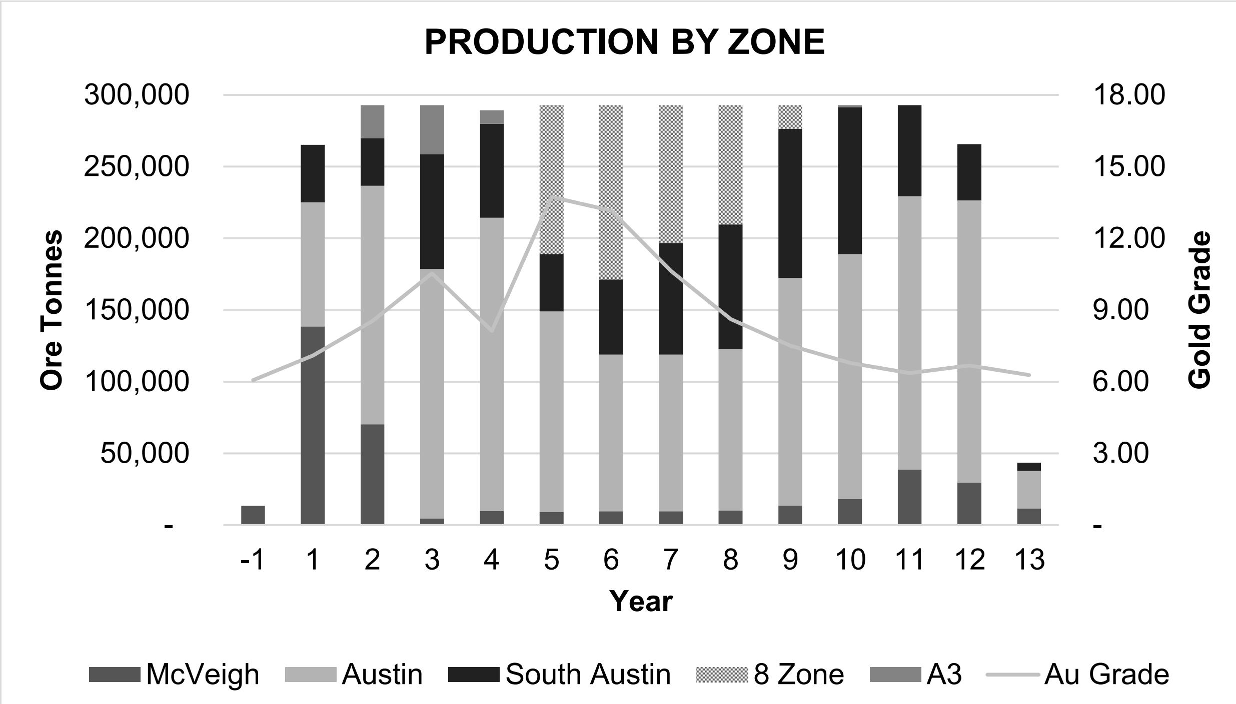


Source: JDS (2019)

16.15.2 Mine Production

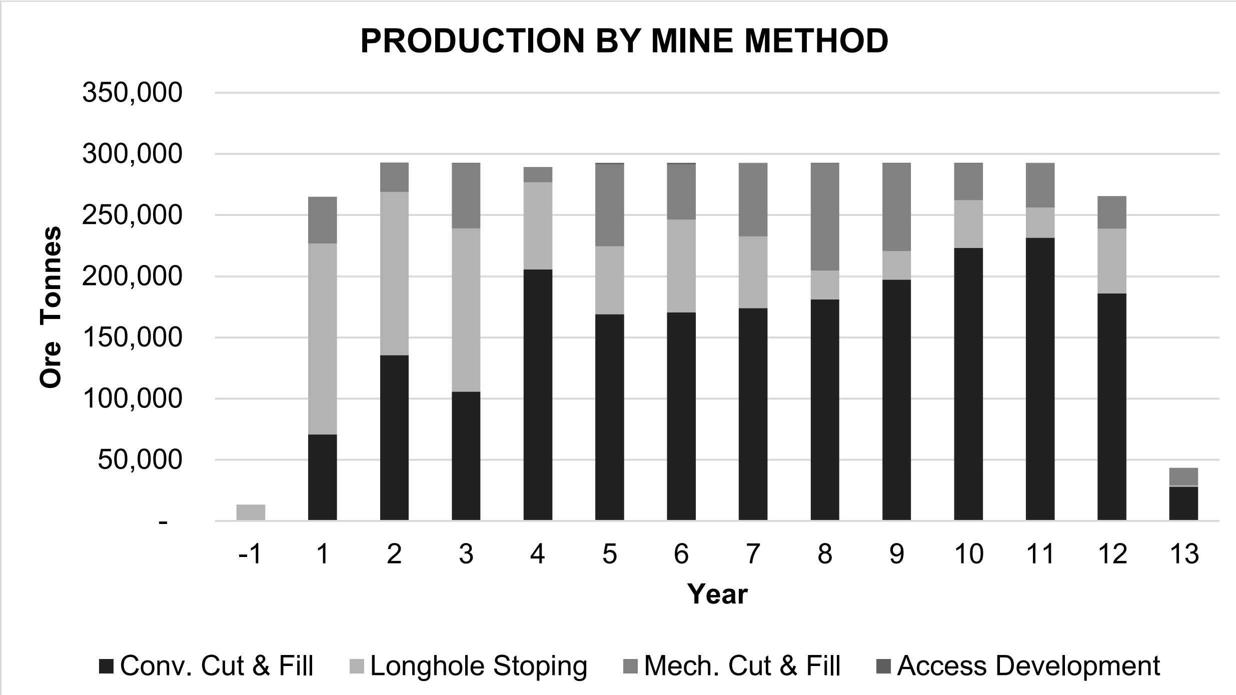
Mine production of 800 t/d will be produced by LH, MCF, and CCF mining with a small amount of ore tonnes also coming from access development. The 12.3 year mine life averages 8.97 g/t gold with production of 1,013 koz gold. Annual mine production by zone and mining method is shown in Figures 16-42 and 16-43.

Figure 16-42: Annual Mine Production by Zone



Source: JDS (2019)

Figure 16-43: Annual Production by Mining Method

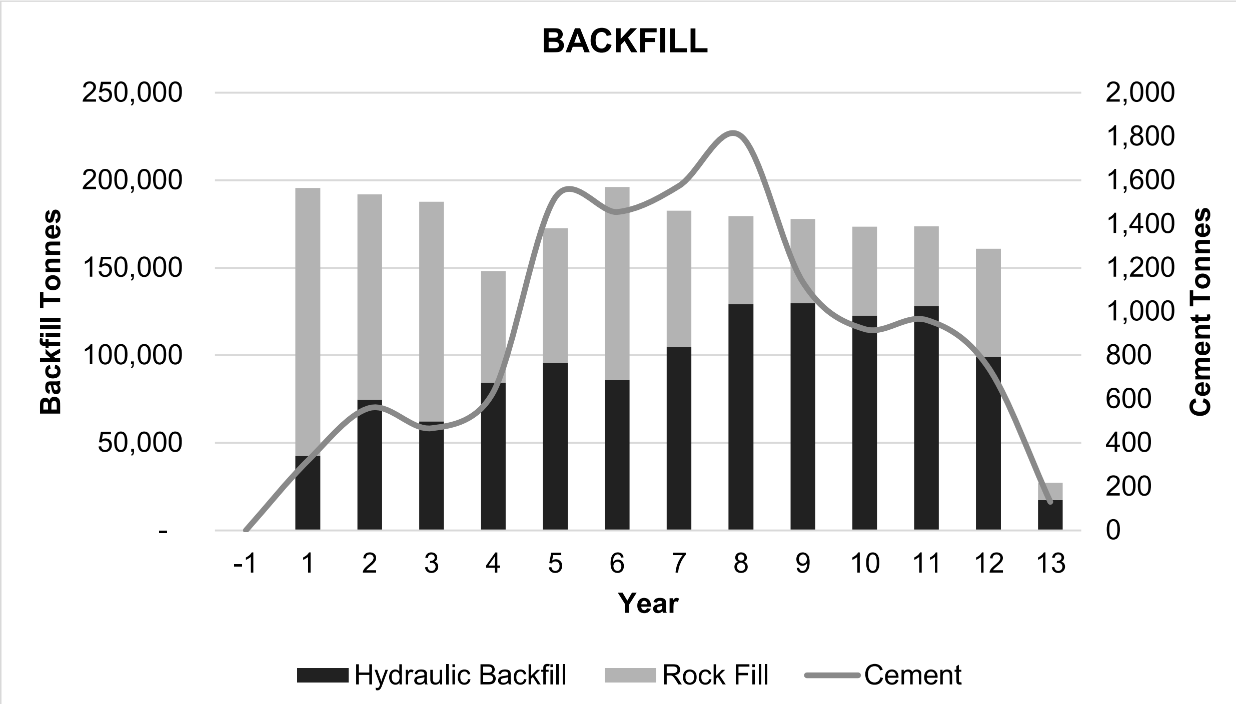


Source: JDS (2019)

16.15.3 Backfill

Mine backfill totals required a LOM placement of 1.39 M m³ of fill material. The backfill is 46% unconsolidated rock fill, and 54% hydraulic fill. Cemented hydraulic fill is used where structural fill is required. Figure 16-44 depicts the mine backfill schedule by fill type and the cement requirements.

Figure 16-44: Annual Backfill Placement including Cement Demand



Source: JDS (2019)

16.15.4 Schedule Summary

The annual mine production schedule is provided in Tables 16-25 to 16-27 and shows annual summaries of tonnage mined, ore grades and backfill quantities. Ore, un-mineralized rock, and backfill tonnages have been rounded to the nearest thousand.

Table 16-25: Annual Production Schedule

Production	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Mined Un-mineralized Rock	kt	4,011	156	537	508	553	469	539	440	148	123	139	131	150	97	20
Mined Ore	kt	3,512	13	265	293	293	289	293	293	293	293	293	293	293	266	43
Mined Grade	Au g/t	8.97	6.06	7.09	8.54	10.54	8.12	13.74	13.15	10.64	8.62	7.52	6.78	6.35	6.68	6.28
Mined Gold	Au koz.	1,013	3	60	80	99	76	129	124	100	81	71	64	60	57	9
Mill Feed Ore	kt	3,512	273	293	293	293	289	293	293	293	293	293	293	293	266	43
Mill Feed Grade	Au g/t	8.97	7.03	8.52	10.51	8.15	13.71	13.15	10.66	8.62	7.53	6.79	6.36	6.67	6.28	7.03

Source: JDS (2019)

Table 16-26: Annual Mine Production by Stoping Type Summary

Mining Method	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Longhole Stoping	kt	861.7	13	156	134	133	71	56	76	59	24	23	39	25	53	1
Mechanized Cut and Fill	kt	565.3	-	38	24	53	12	67	45	60	88	72	30	37	26	14
Conventional Cut and Fill	kt	2,078.5	-	71	135	106	206	169	171	174	181	197	223	232	186	28
Access Development	kt	6.7	-	-	0.1	0.6	0.3	1.6	1.6	0.3	0.5	0.9	0.5	0.3	0.2	-

Source: JDS (2019)

Table 16-27: Annual Backfill Summary

Backfill	Unit	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Rock Fill	kt	991.3	-	153	117	125	64	77	110	78	50	48	51	46	62	108
Hydraulic Fill	kt	1,176.0	-	42	75	62	84	96	86	105	129	130	123	128	99	17

Source: JDS (2019)

17 Process Description / Recovery Methods

The Pure Gold Mining's Madsen Project focuses on developing five different lenses – Austin, South Austin, A3, McVeigh and 8 Zone. The recent metallurgical test program completed at Base Metallurgical Labs in Kamloops, BC (BL0288), summarized in Section 13, has demonstrated that gravity concentration followed by pre-oxidation, cyanide leach, carbon adsorption/desorption and electrowinning can yield an average overall gold extraction of 96.6%. Results from this test program were used to develop the corresponding process design criteria, mechanical equipment list, flowsheets and operating costs.

The process plant will include:

- One stage of crushing;
- Two stages of grinding with hydro-cyclone size classification;
- Gravity concentration and Intensive Leach;
- Cyanide leaching and carbon adsorption using carbon-in-pulp (CIP);
- Cyanide destruction, dewatering for hydraulic fill and storage of slurry tailings;
- Carbon acid wash, elution and regeneration; and
- Electrowinning and refining.

The primary crushing plant will have a throughput of 800 t/d with an average life of mine (LOM) head grade of 8.97 g/t Au. The circuit will operate at an availability of 50%, resulting in an hourly throughput of 66.7 t/h. The milling, gravity, leach and CIP circuits will operate 24 hours per day, 365 days per year at an availability of 95%, resulting in an hourly throughput of 35.1 t/h. The carbon plant is expected to transfer 1 tonne of loaded carbon daily to the elution circuit to recover the gold to doré bars.

The primary crushing circuit will reduce the material down to a product size of 80% passing (P_{80}) 103 mm and two stage of grinding will target a final P_{80} grind size of 75 µm. Two batch gravity concentrators will be installed in the ball mill circuit to recover any gravity recoverable gold. Ground slurry will then be leached with cyanide and the gold-cyanide complexes will be adsorbed onto the pores of activated carbon in a Carbon in Pulp (CIP) circuit. The loaded carbon will then be washed with hydrochloric acid and stripped under pressure using a hot 1% sodium hydroxide solution. The resulting pregnant gold solution will be subjected to electrowinning and the recovered gold sludge will be refined in an induction furnace to produce gold doré bars. Leached slurry from the CIP circuit will be treated with the SO_2/Air cyanide destruction process, and the final tailings will be processed for hydraulic fill or pumped to the tailings facility.

17.1 Introduction

The recovery method will consist of the following unit operations:

- **Primary Crushing** – A vibrating grizzly feeder and 25" x 40" jaw crusher in open circuit, producing a final product P_{80} of 103 mm;

- **Crushed Material Bin and Reclaim** – A 600 t live capacity bin with chain gate at the discharge chute feeding onto the SAG mill feed conveyor.
- **Primary Grinding** – A 4.9 m diameter x 1.5 m long semi-autogenous grinding (SAG) mill in closed circuit with a screen, producing a T_{80} transfer size of 750 µm;
- **Secondary Grinding** – A 2.9 m diameter x 4.9 m long ball mill in reverse closed circuit with a cluster of hydro-cyclones, producing a final product P_{80} 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** – Intensive leach reactor to leach gold from the gravity concentrator concentrate for further processing in the refinery to produce gold doré.
- **Pre-leach Thickening** – An 18 m diameter high-rate thickener to achieve an underflow solids density of 50% prior to leaching;
- **Pre-oxidation** – An agitated tank sparged with oxygen to oxidize the slurry prior to leaching;
- **Cyanide Leaching** – Five agitated leach tanks, giving 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 agitated tanks, one operating and one standby, to reduce the CN_{WAD} (weak acid dissociable) concentration in the CIP tailings to < 5.0 mg/L with sodium metabisulphite for SO_2, air, lime and copper sulphate; and
- **Final Tailings** – A plant will process the tailings for hydraulic fill for deposition underground or the slurry will be pumped to the TMF.

17.2 Plant Design Criteria

17.2.1 Process Design Criteria

The Process Design Criteria and Mass Balance detail the annual production capabilities, major mass flows and capacities, and availabilities for the crushing, process and carbon handling plants. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 22. Key process design criteria from Section 13 are summarized in Table 17-1.

Table 17-1: Key Process Design Criteria

Criteria	Unit	Nominal Value	Source
General			
Daily Throughput	t/d	**800**	JDS FS 2018 mine plan
Process Plant Availability	%	**95**	Based on previous operating data
Process Plant Throughput	t/h	**35.1**	Engineering Calculation
LOM Average Au Head Grade	g/t	**8.97**	JDS FS 2018 mine plan
Overall Au Recovery	%	**95.8**	Base Met (2018): BL0288 average of 30 variability composites with gravity concentration and 24 h leach
Crushing			
Bond Crushing Work Index	kWh/t	**13.2**	Base Met (2018): BL0288
Availability/Utilization	%	**50**	Selected to utilize jaw crusher capacity
Crushing Plant Throughput	t/h	**66.7**	Engineering Calculation
Number of Crushing Stages	-	**1**	Industry Standard
Primary Crusher Type	-	**Jaw**	Industry Standard for throughput requirement
Primary Crusher Product Size (P_{80})	mm	**103**	Vendor Simulation at a closed side setting of 90 mm
Crushed Material Storage Bin			
Bin Capacity (live)	t	**600**	Design Consideration
Bin Capacity (live)	h	**18**	Engineering Calculation
Grinding			
SMC – Mia	kWh/t	**26.2**	Base Met (2018): BL0288 average of 12 samples
SMC – Mib	kWh/t	**18.8**	Base Met (2018): BL0288 average of 32 Bond ball mill work index samples
SMC – Mic	kWh/t	**11.0**	Base Met (2018): BL0288 average of 12 samples
Axb	-	**29.4**	Base Met (2018): BL0288 average of 12 samples
Bond Ball Mill Work Index (Average)	kWh/t	**14.1**	Base Met (2018): BL0288 75th percentile of 32 samples
Bond Abrasion Index	g	**0.239**	Base Met (2018): BL0288
Primary Grinding Mill Type	-	**SAG Mill**	Industry Standard for primary grinding to target transfer size
Mill Diameter	m	**4.9**	Existing
Mill Length	m	**1.5**	Existing
Installed Power	kW	**336**	Design Consideration to replace 336 kW motor
Circuit Configuration	-	**Closed**	Design Consideration
Size Classification Type	-	**Screen**	Design Consideration
Circulating Load	%	**15**	Design Consideration
Primary Grinding Transfer Size (T_{80})	µm	**750**	Design Consideration
Secondary Grinding Mill Type	-	**Ball Mill**	Industry Standard
Mill Diameter	m	**2.9**	Existing
Mill Length	m	**4.9**	Existing
Installed Power	kW	**597**	Design Consideration to replace 261 kW motor
Circuit Configuration	-	**Reverse Closed**	Industry Standard
Size Classification Type	-	**Hydro-cyclones**	Industry Standard
Circulating Load	%	**300**	Industry Standard

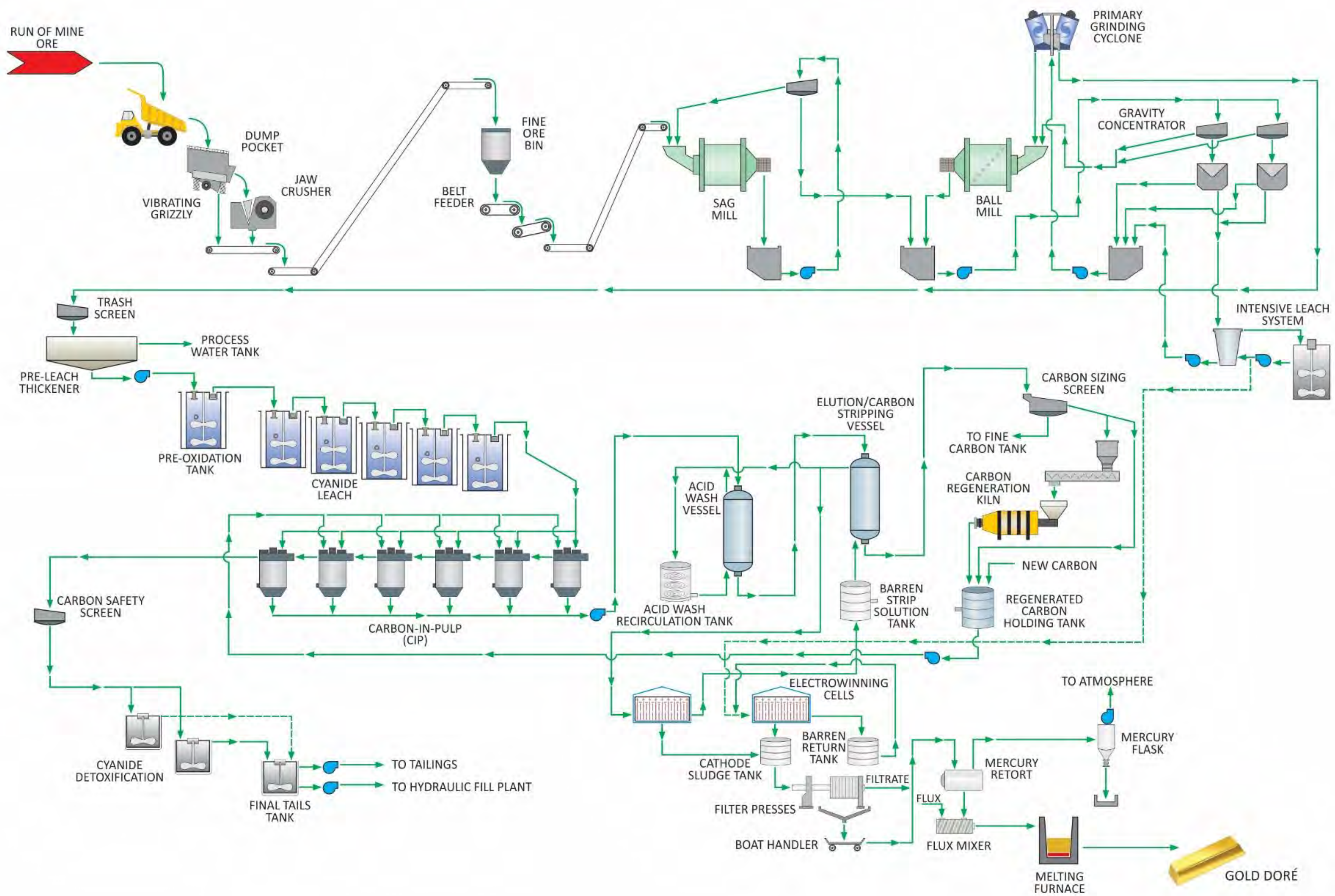
Criteria	Unit	Nominal Value	Source
Final Product Size (P_{80})	µm	**75**	Base Met (2018): BL0288
Gravity Concentration			
Concentrator Type	-	**Batch Centrifugal**	Industry Standard
Number of Units	#	**2**	Vendor Recommended based on high gravity recoverable gold projections from Base Met (2018): BL0288 test work
Feed Source	-	**SAG Screen Undersize / Ball Mill Discharge**	Design Consideration based grind size and water balance
Recovery Method	-	**Intensive Leach Reactor**	Industry Standard
Pre-Leach Thickening			
Thickener Loading Rate	t/hr/m²	**0.57**	Historical data
Thickener Underflow Density	% w/w	**50**	Design Consideration
Leaching			
Pre-Oxidation	Y / N	**Yes**	Base Met (2018): BL0288
Pre-Oxidation Retention Time	h	**2**	Base Met (2018): BL0288
Leach Retention Time	h	**24**	Base Met (2018): BL0288
Number of Leach Tanks	#	**5**	Design Consideration to ensure adequate leach time without short-circuiting
Sodium Cyanide Concentration	ppm	**500**	Base Met (2018): BL0288
CIP			
CIP Retention Time	h	**5**	Base Met (2018): BL0288
Number of CIP Tanks	#	**6**	Design Consideration to ensure adequate adsorption time without short-circuiting
Carbon Loading	g / t carbon	**4,500**	Base Met (2018): BL0288, Design Consideration and Vendor Recommendation
Carbon Processing			
Carbon Handling Capacity	t/d	**1.0**	Engineering Calculation
Acid Wash Type	#	**Hydrochloric Acid**	Industry Standard
Elution Operating Temperature	⁰C	**140**	Vendor Recommendation
Elution Operating Pressure	kPa	**350 to 500**	Vendor Recommendation
Smelting Furnace Type	-	**Elec. Induction Furnace**	Industry Standard / Vendor Recommendation
Carbon Consumption Rate	kg / t Carbon Stripped	**30**	Based on operating data
Cyanide Destruction			
Discharge Solution, CN_{WAD}	mg/L	**< 5.0**	Base Met (2018): BL0288
Design Retention Time	h	**1.0**	Base Met (2018): BL0288
Number of Tanks	#	**2**	Selected to allow one operating and one standby as per International Cyanide Management Code
SO_2 Consumption	SO_2:CN_{WAD}	**5:1**	Base Met (2018): BL0288
Lime Consumption	CaOH:CN_{WAD}	**2:1**	Base Met (2018): BL0288
$CuSO_4$-$5H_2O$ Consumption	mg/L	**30**	Base Met (2018): BL0288
Tailings Management			
Disposal Type	-	**Slurry Tailings and Hydraulic Fill**	Design Consideration

Source: JDS (2019)

17.3 Plant Design

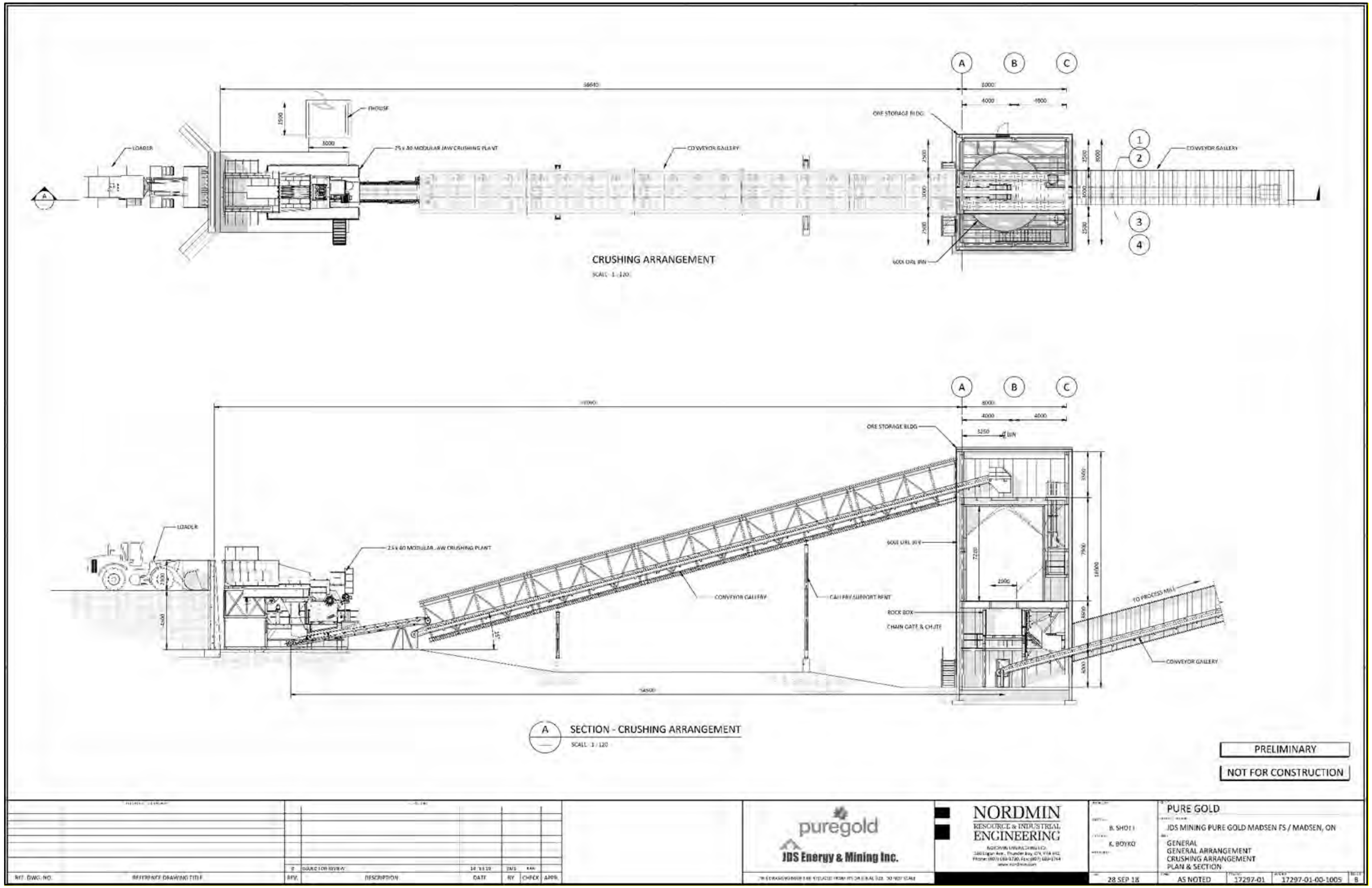
A summary of the process flowsheet is presented in Figure 17-1. Models of the crushing and process facilities are displayed in Figure 17-2 and Figure 17-3, respectively.

Figure 17-1: Overall Process Flowsheet



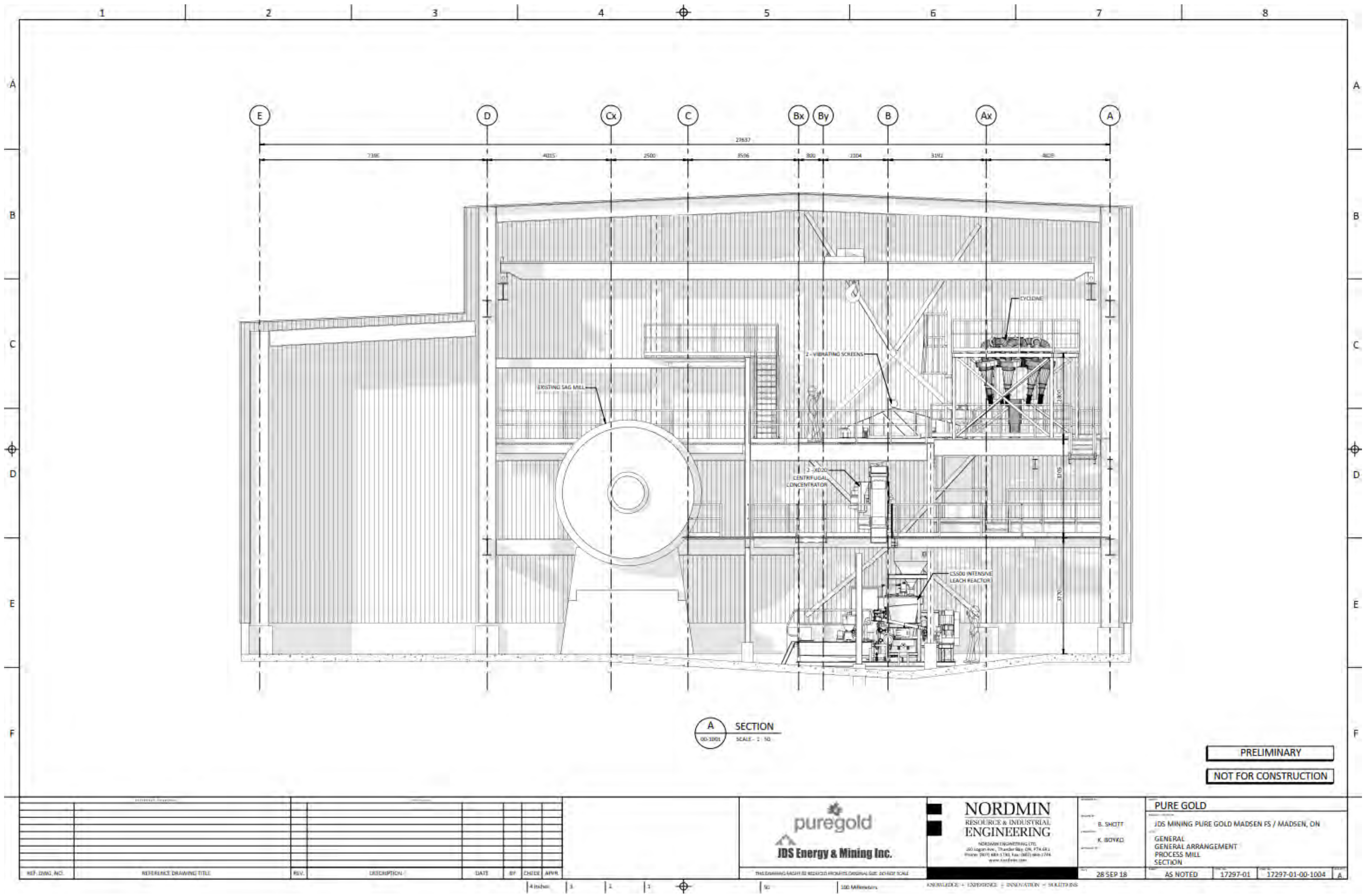
Source: Pure Gold (2019), after JDS (2019)

Figure 17-2: Crushing and Mineralized Material Storage Layout



Source: JDS (2019)

Figure 17-3: Process Plant Layout



Source: JDS (2019)

17.4 Process Plant Description

17.4.1 Crushing

Material from the underground mining operations will feed a crushing plant that consists of one stage of crushing. The crusher will process 66.7 t/h of material, operate 12 hours per day and produce a final product P_{80} of 103 mm.

17.4.1.1 Primary Crushing

Material will be dumped onto a 500 mm static grizzly above a dump pocket. The material will discharge through the static grizzly into the crusher feed hopper. Oversize material from the static grizzly will be removed for later size reduction using a rock breaker.

A vibrating grizzly feeder will draw material from the feed hopper at a rate of 66.7 t/h. The vibrating grizzly oversized material will feed directly into a 635 mm x 1,016 mm (25" x 40") jaw crusher with an installed power of 90 kW. The undersized -75 mm material will bypass the crusher and feed directly onto the bin feed conveyor. The primary crushing stage will produce a product P_{80} of approximately 103 mm at a crusher closed side setting (CSS) of 90 mm.

17.4.2 Crushed Material Storage Bin

Primary crusher product, with a P_{80} of 103 mm, will be conveyed to the crushed material storage bin. The bin will provide 600 t of live storage capacity. Chain gate on the discharge chute located underneath the bin will feed the primary grinding circuit. The SAG feed conveyor installed with a VFD and scale will be capable of providing a constant feed to the plant of 35.1 t/h.

17.4.3 Grinding

The grinding circuit will consist of a primary SAG mill followed by a secondary ball mill. A gravity concentration circuit will be installed in the ball mill circuit to recover any gravity recoverable gold. The primary SAG mill will operate in closed circuit with a screen, while the secondary ball mill will operate in reverse closed circuit with a cluster of hydro-cyclones. SAG screen undersize and ball mill discharge will be processed through the gravity circuit. The grinding circuit will be able to process a nominal throughput of 35.1 t/h (fresh feed), and produce a final product P_{80} of 75 µm.

17.4.3.1 Primary Grinding

Reclaimed material from the crushed material storage bin will feed a 4.9 m diameter x 1.5 m long grate discharge SAG mill via the SAG mill feed conveyor. The mill will be installed with a 373 kW induction motor. A 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%. Slurry will flow through the discharge grate into the SAG mill pump box. The ground slurry will then be pumped to a 0.9 m x 2.4 m vibrating screen with an aperture size of 2 mm for wet screening. The SAG screen oversize will discharge back into the SAG mill for further grinding, while the SAG screen undersize will be sent to the gravity concentration circuit. The primary grinding circuit will produce a T_{80} transfer size of 750 µm.

17.4.3.2 Gravity Concentration

Undersize material from the SAG discharge screen will flow into the gravity feed pump box and combine with the ball mill discharge before being pumped to the gravity concentrator feed screen. With an aperture size of 1 mm, the feed screen will remove any oversize particles prior to gravity concentration. The screen undersize will be split and fed into one of two semi-continuous batch 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 ten minute cycles. During a cycle, gravity recoverable gold will collect in the concentrate cone. At the end of the cycle, the gravity concentrator feed will be 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.

17.4.3.3 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 six (4 operating / 2 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 P_{80} of 75 µm, will be pumped to the pre-leach thickener. The hydro-cyclones have been designed for a 300% circulating load.

Cyclone underflow will feed a new 2.9 m diameter x 4.9 m long overflow ball mill with an installed power of 597 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.

17.4.4 Pre-Leach Thickening

Cyclone overflow, at a P_{80} of 75 µm, will flow onto a vibrating trash screen for removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to an 18 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 tank, while the thickener overflow will flow by gravity into the process water tank to be used as make-up water in the grinding circuit.

17.4.5 Leaching

Pre-leach thickener underflow will be pumped to a 5 m diameter x 6 m high pre-oxidation tank prior to leaching. Oxygen will be sparged into the bottom of the agitated tank and slurry will be conditioned for 2 hours to oxidize sulphide minerals.

Based on metallurgical testing, 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 consumptions in the cyanide destruction circuit.

The oxidized slurry will then flow to the first of five 6.9 m diameter x 8.2 m high agitated leach tanks. The leach circuit is designed to provide 24 hours of retention time and will be operated at a sodium cyanide concentration of 500 ppm. Lime slurry will be added to the first and second leach tanks to maintain protective alkalinity at a design pH of 11.0, preventing the creation of hydrogen cyanide gas (HCN). Oxygen will be sparged in from the bottom of each tank. As the slurry progresses through the circuit, gold will be leached into solution according to the following chemical formula:

$$4\ Au\ (s) + 8\ NaCN\ (aq) + O2\ (g) + 2\ H2O\ (l) \rightarrow 4\ Na[Au(CN)2]\ (aq) + 4\ NaOH\ (aq)$$

Slurry from the leach circuit will then flow to the CIP circuit for carbon adsorption.

17.4.6 Carbon in Pulp (CIP)

Leached slurry will flow into the first of six 3.5 m diameter x 4.8 m high CIP tanks. Each tank will be installed with an agitator and an inter-stage screen pumpcell for retaining activated carbon. As the slurry flows through the six CIP tanks, gold-cyanide complexes will be adsorbed onto the pores of activated carbon. The average carbon concentration in the CIP circuit is expected to be approximately 25 g/L, with the concentration higher at 50 g/L in the first tank to maximize adsorption.

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 located in the first CIP tank, interacting with the highest concentration of gold. Once 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 regeneration circuit will be pumped into the first CIP tank, becoming the last tank in the CIP train.

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 disposed of. Safety screen undersize will then be pumped to the cyanide destruction circuit.

17.4.7 Carbon Processing

The carbon processing plant has been designed to process 1 t/d of loaded carbon, producing gold doré. A batch of carbon will be processed through the acid wash, stripping and regeneration circuits on a daily basis.

17.4.7.1 Acid Wash

Loaded carbon from the CIP circuit will flow by gravity into a one tonne capacity acid wash vessel constructed of fibre-reinforced plastic. The carbon will be 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 regeneration circuit using a horizontal electric kiln.

During the acid wash cycle, the carbon will first be rinsed with fresh water. HCl solution will then be pumped from the dilute acid tank upward through the acid wash vessel, overflowing back into the dilute acid tank. The carbon will then be rinsed with fresh water to remove the remaining acid and any mineral impurities.

A recessed impeller pump will transfer acid washed carbon from the acid wash vessel into the elution vessel using transport water. Carbon slurry will discharge directly into the top of the elution vessel. Under normal operation, only one acid wash and elution cycle will take place per day.

17.4.7.2 Carbon Stripping (Elution)

The carbon stripping (elution) process will utilize barren strip solution to strip the loaded carbon, creating a pregnant gold solution which 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 for reuse.

The strip vessel will be a carbon steel tank with a capacity to hold approximately one tonne of carbon. During the strip cycle, 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 below its boiling point 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.

17.4.7.3 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 elution 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 250 kW horizontal electric kiln with residual heat dryer will be utilized to treat 1 t 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 then drain into the carbon quench tank and combine with the regenerated carbon discharging from the kiln.

17.4.8 Electrowinning and Refining

Pregnant solution from the strip circuit and intensive leach reactor will be pumped to the refinery for electrowinning in separate circuits, producing a gold sludge. The sludge will then be filtered, dried and refined in an electric induction furnace, producing gold doré bars.

Pregnant strip solution will be pumped through two 3.54 m^3 electrowinning cells operating in parallel. Gold will plate on the pair of 33 stainless steel cathodes, while the barren solution will flow into the barren return tank and be pumped back to the barren solution tank for reuse. To prevent a build-up of impurities, a 15% daily bleed of barrens solution will be pumped to the leach circuit. A separate electrowinning circuit will be installed to process the pregnant solution from the intensive leach circuit.

Gold rich sludge will periodically be washed off the stainless steel 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 a 125 kW induction furnace, producing gold doré. This process will take place within a secure and supervised area, and the precious metal product will be stored in a vault until shipping off site.

17.4.9 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 1 hour. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the circuit will then be pumped to the final tailings stock tank.

The cyanide destruction circuit will treat CIP tailings slurry, process spills from various contained areas, and process bleed streams.

Process air will be sparged from near the bottom of the tanks, under the agitator impeller. Lime slurry will be added, if necessary, to maintain the optimum pH of 8.0 – 8.5 and copper sulphate (CuSO4) will be added as a catalyst, maintaining a 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.

Approximately 50% of the treated slurry from the cyanide destruction circuit will be pumped to the final tailings management facility and the remaining 50% to the hydraulic fill plant.

17.4.10 Reagents Handling and Storage

Reagents consumed within the plant will be 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 will be bermed with sump pumps which will transfer spills to the final tailings stock tank, except for flocculant which will be returned back to the flocculant storage tank. The reagents will be mixed, stored and then delivered to the pre-leach thickener, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow metres and manual control valves. The capacity of the storage tanks will be sized to handle one day of production. 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 daily consumption rates. The table also includes other major process consumables.

Table 17-2: Process Design Criteria

Description	Delivered Form	Daily Usage (kg/d)
NaCN	1 tonne bags (dry)	300
Lime	2 tonne bags (dry)	559
Pb_2NO_3	50 kg bags (dry)	200
HCl	208 L drums (36% liquid)	104
NaOH	50 kg bags (dry)	46
$CuSO_4$	50 kg bags (dry)	98
SMBS	500 kg bags (dry)	826
Antiscalant	50 kg barrels	41
Flocculant	25 kg bags (dry)	12
Activated Carbon	50 kg bags (dry)	30
SAG Mill Grinding Media – 100 mm chrome steel	1 tonne bags	421
Ball Mill Grinding Media – 50 mm chrome steel	1 tonne bags	649

Source: JDS (2019)

17.4.11 Air Supply

An instrument and plant air system with two compressors and associated dryers, filters, and receivers will be provided and located in a compressor room inside the plant building.

Oxygen will be used in the pre-oxidation, leach and cyanide destruction circuits. Oxygen will be supplied by an oxygen generation system that is fed by two compressors.

17.4.12 Water Supply and Consumption

The following water types will be used in the process plant:

Plant Water – Plant water will be supplied from the TMF via the fire/plant water tank. The plant water will be used to supply gland, reagent and make-up water. Overflow water from the pre-leach thickener and plant water will supply water to the grinding water. The grinding water will have a low gold concentration and will be used predominantly in the primary and secondary grinding circuits to dilute slurry to the required densities.

Fresh Water – Fresh water for the process plant will be pumped from the polishing pond. Fresh water will be used as reagent make-up water, gland water, process make-up water and cooling water services in the strip circuit boiler. The estimated make-up water consumption in the process plant will be approximately 35 m^3/h.

17.5 Electrical and Controls Description

17.5.1 Control Systems – Mill Equipment

The current control system utilizes local control panels to operate individual pieces of equipment including the SAG mill, ball mill, kiln, induction furnace, electrowinning, emergency fire water pump and the main alarm annunciator. To allow consistent, automatic control of the key systems in the concentrator and therefore, the plant as a whole, a new concentrator-wide programmable logic controller (PLC) based control system will be commissioned. The new control system will consist of:

- A master PLC that can monitor and/or control any of the systems installed in the concentrator. The PLC will be fed by UPS power and by redundant chassis power-supplies to maximize the system up-time;
- Remote I/O control panels located at:
    - Area 100 for Eductor Water Supply & Distribution;
    - Area 300 for Primary Grinding, Secondary Grinding and Thickening;
    - Area 400 for Leaching, C.I.P. and Tailing / Reclaim Water;
    - Area 500 for Carbon Stripping, Acid Wash, Electrowinning / Refinery, Carbon Regen; and
    - Area 600 for Reagents, Compressed Air Generation and Process and Plant Distribution.
- A human-machine operator interface (HMI) that is distributed to all PLCs in the plant. The graphical interface will be designed to newer “situational awareness” standards which leverages the effective use of colour in the graphical animations to quickly draw an operator’s attention to abnormal operating conditions. The graphics will also be designed to minimize “clicks” such as page changes, frivolous popups, etc.; and
- A control system network that connects the main concentrator PLC via Ethernet to the I/O controlled by the PLC, to other PLCs in the concentrator (so process systems that are supplied complete with a PLC can be monitored/controlled), to the HMI servers, to the engineering workstation.

18 Project Infrastructure and Services

18.1 General

The Project infrastructure is designed to support the operation of an 800 t/d mine and processing plant, operating on a 24 hour per day, 7 day per week basis. It's designed for the local conditions and topography.

The main infrastructure items include:

- High voltage electrical substation, site electrical power distribution, and emergency power generators;
- On-site access roads development and improvements;
- Primary crusher, discharge conveyor, and crushed ore bin;
- Process plant improvements with new equipment and refurbishments;
- Assay laboratory;
- Backfill plant for mixing underground hydraulic backfill;
- Tailings detoxification;
- Temporary mine rock stockpile;
- Tailings management facility;
- Mine rock management facility;
- Overburden stockpile;
- Soil stockpile; and
- Water treatment plant (WTP).

18.2 General Site Layout

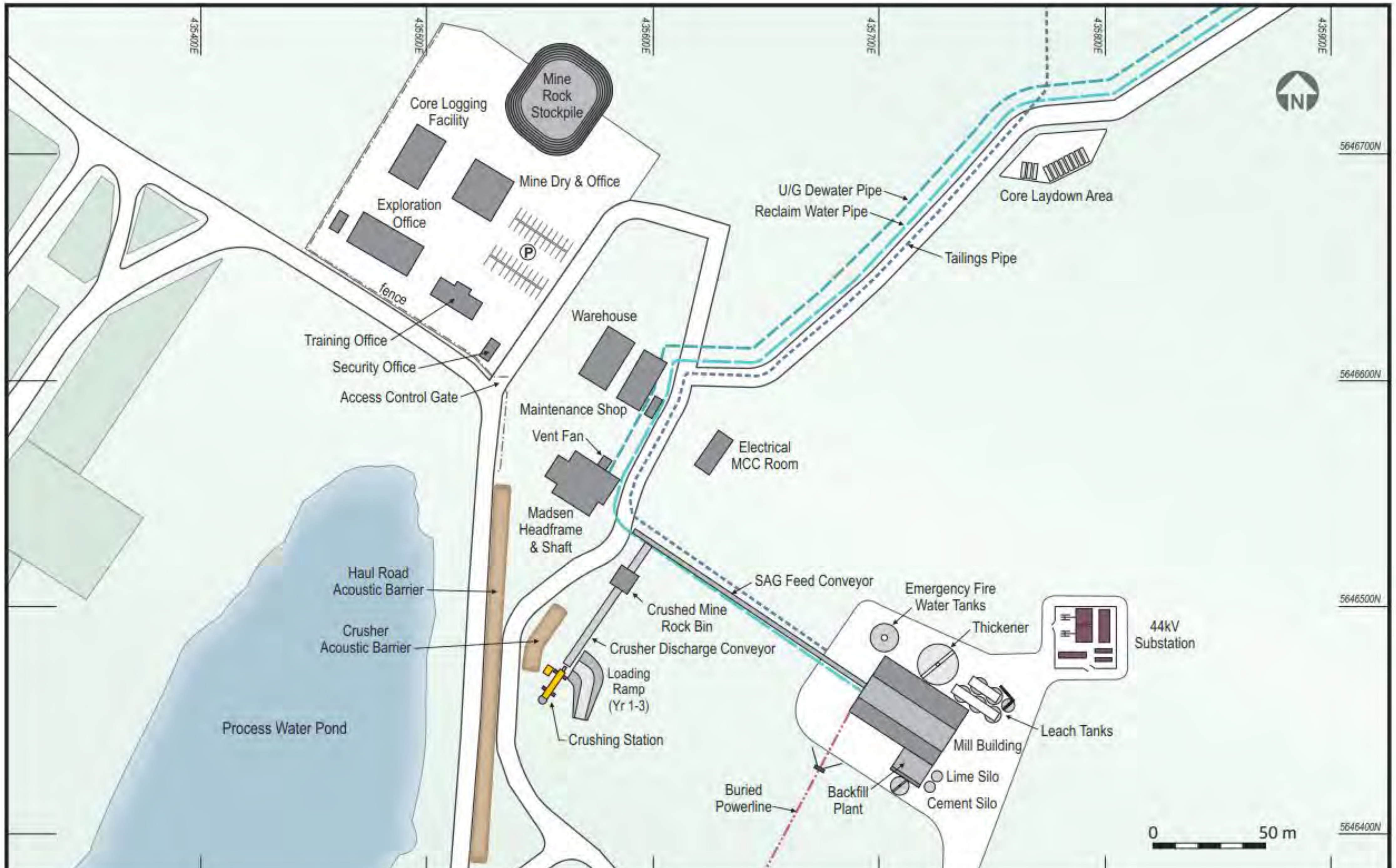
The site layout has been designed to minimize environmental and community impacts, provide security-controlled site access, minimize construction costs and optimize operational efficiency. The existing infrastructure will be utilized to the maximum extent possible. New facilities have been located to allow easy access for construction and utilize existing topography and existing development footprint to minimize earthworks requirement. The new primary crusher and ore bin have been located next to the head frame and existing mill feed conveyor.

The project site overall layout is provided in Figure 18-1. The plant site and the main infrastructure facilities arrangements are presented in Figure 18-2.

Figure 18-1: Overall Site Layout

Source: JDS (2019)

Figure 18-2: Main Plant and Infrastructure Site



Source: JDS (2019)

18.3 Electrical and Communications

18.3.1 Mine Site Power

Electrical grid power will provide the power to the project over the life of the mine. The existing site 2300 V power distribution switchgear is currently serviced by 2300 V power supplied by a Hydro One owned 44 kV / 2300 V transformer station. Increased operational power requirements are expected to be available from the 44 kV supply network grid, with 7 MW available until January 1, 2022, when the available power supply is anticipated to increase after the installation of a new 230 kV line to Pickle Lake. Site power distribution is to be provided by a new transformer station to be constructed on the mine site, and a refurbished power distribution system.

An electrical load list was developed for the Project operations, based on the process design and mechanical equipment list. Two cases are presented, representing the estimated loads prior to, and after the operation of the hoist. They are summarized, respectively, in Table 18-1, and Table 18-2.

Table 18-1: Summarized Electrical Load List Year 3

Service Area	Connected Load (kW)	Demand Load (kW)
Process Plant	4,961	2,484
Site Infrastructure	1,325	663
Underground Mining (No Hoist in Operation)	8,170	3,402
Totals	14,456	6,549

Source: JDS (2019)

Table 18-2: Summarized Electrical Load List Year 5 Month 2

Service Area	Connected Load (kW)	Demand Load (kW)
Process Plant	4,961	2,484
Site Infrastructure	1,325	663
Underground Mining (Hoist in Operation)	8,284	4,751
Totals	14,570	7,898

Source: JDS (2019)

18.3.2 Communications

Communication lines will be installed on the electrical pole line from the main road to the mine site. The existing mine site surface and underground radio system would be expanded. Landline telephones would be used for external contact.

18.3.3 Power System – Mine Main Feed

The main mine site 2300 V electrical substation is currently located near the headframe and is beside the Madsen Hydro One 44 kV / 2300 V transformer station (TS). The Hydro One TS, that is now obsolete, feeds the town as well as the mine. Hydro One is installing a new TS at a new location to feed the town of Madsen and the surrounding communities.

A new dedicated electrical substation servicing only the Madsen mine will be installed and commissioned for the mining and milling operations at the mine site. The dedicated substation will be located closer to the mill to take advantage of the closer proximity of the 44 kV power line. The new substation will be equipped with a 2 MW standby generator to feed the critical loads at the mine site, and provide additional capacity to the grid power if required.

The new substation will supply power to feed the concentrator building at 4160 V, the existing portal at 4160 V, the main underground distribution at 4160V, the wastewater treatment plant, tailings facility, and Fork, Russet South, and Wedge portals from a new overhead power line at 4160 V, and the main mine site 2300 V electrical distribution system will be fed at 4160 V through an existing step down transformer. A new main mine site 2300 V switchgear building / substation will replace the existing main mine site 2300 V substation.

A single line diagram of the main site distribution captures 4160 V, 2300 V and 600 V distribution systems at the mine site is shown in Figure 18-3.

18.3.4 Power System – Concentrator Electrical Systems

The concentrator 4160 V, 600 V, 240 V and 120 V electrical distribution systems including the 4160 V switchgear, 4160 V / 600 V transformers and 600 V MCC's and utilities including lighting will all be retained as they can be refurbished and upgraded to integrate into the new automation system. A new 4160 V / 600 V 5 MVA unit sub will be added at the concentrator to provide power to additional loads/MCC's added for concentrator optimization and for the new ball and SAG mill drives.

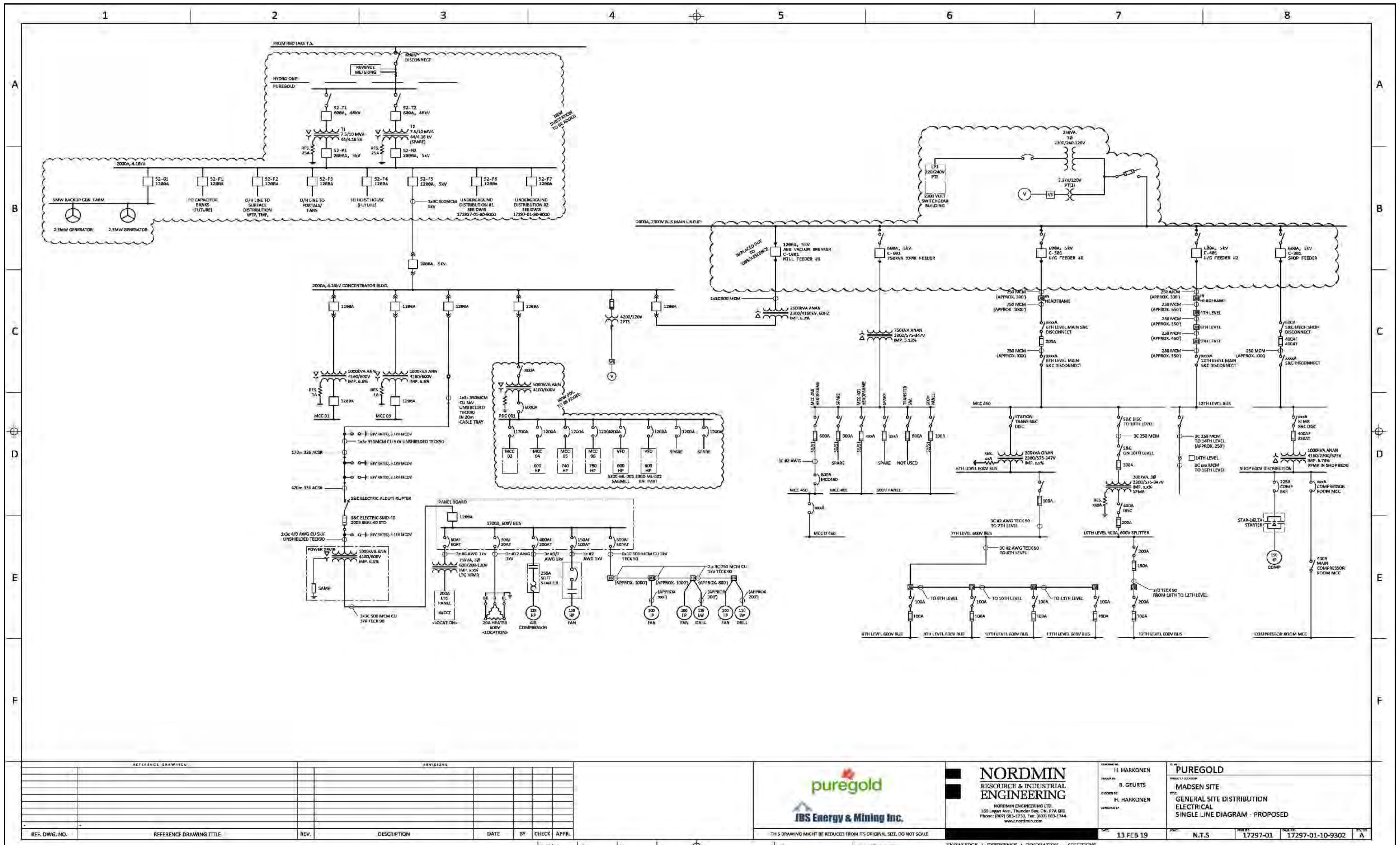
18.3.5 Power System – Surface Distribution

As shown on the main site single line diagram, Figure 18-3, power distribution to surface infrastructure including the wastewater treatment plant, tailings facility, and portals will be accomplished by installing a 4160 volt overhead power line system supplied from two 4160 volt feeder circuit breakers to be located at the new main substation switchgear building. The three phase, overhead, wood pole line will include provisions for installation of a fibre optic communications system as needed. Switches and fuses will be provided at each load location to allow for a cable connection to the various equipment.

18.3.6 Power System – Underground Mine

The underground power system has been described in Section 16.9.5.

Figure 18-3: Electrical Single Line Drawing



Source: JDS (2019)

18.4 Headframe

A continuous pour concrete headframe installed in the 1950's was used for prior underground mining operations at the site and remains in place. The headframe serviced a 1,275 metre deep, five compartment shaft and has an existing friction hoist mounted at the top. The shaft will initially be used as an exhaust air raise. New ventilation fans will be installed adjacent to the existing headframe. These fans will directly pull air from Levels 1 through 17.

The headframe will be upgraded in year 3 with the installation of a hoisting system to deliver mine rock to surface. The installation is described in more detail in Section 16.11. Once installed and in operation, the mill feed produced will be skipped to a surface stockpile and then to the crusher by FE loader. Un-mineralized rock will be dumped in a separate pocket, skipped to surface and loaded onto trucks to haul to the MRMF.

18.5 Processing Plant

The processing plant was purchased from Placer Dome (originally built at Dona Lake Mine), and was reassembled at Madsen in the late 1990's. The mill facility was in operation at Madsen from 1997 to 1999, at which time it was placed on care and maintenance. The mill consists of a two-stage grinding circuit, cyanide leaching circuit, and refinery.

The mill refurbishing plan is based on a site inspection and estimate from a site visit in early 2017 and supported by a subsequent site inspection in April 2018. The criteria used for the evaluation of the mill equipment was focused on achieving a 92% availability of the equipment processing 800 t/d through refurbishment and new investment. The process plant will require some mechanical equipment additions, replacements, and refurbishment of the existing process equipment. The major new equipment additions are listed as follows:

- A primary jaw crusher station, crusher ore storage bin, and transfer conveyors;
- A 9.5' x 16' ball mill;
- Primary cyclone for grinding circuit;
- Two gravity concentrators and intensive leach reactor circuit;
- Pre-aeration tank and agitator for the leach circuit;
- Carbon screens;
- Strip circuit heating system;
- Gold room equipment, including EW cell, sludge filter, melt furnace and ventilation system;
- Cyanide destruction circuit;
- Oxygen generator; and
- Pumps where increased flow or additional circuits require them.

The existing equipment to be used in this process will be refurbished as required to be operational for the designed throughput. To ensure 95% system availability, complete redundancy in pumping systems will be introduced, with installed spares available for service.

The current process plant building will be utilized, with upgrades for ventilation and make up air systems.

18.5.1 Crusher and Ore Bin Installation

A new modular primary crusher, ore bin, ore bin feed conveyor, and sag mill feed transfer conveyor will be installed on the surface near the existing headframe. The installation will feed crushed ore to the existing SAG mill feed conveyor installed at site.

The crusher will be supplied with structural steel modular frame, receiving dump hopper, grizzly feeder and discharge conveyor. It will be installed on a concrete foundation. Earthen ramps will be installed to the dump pocket for feeding by FE loader or dump truck.

The crusher discharge conveyor will transfer material to the ore bin feed conveyor. This conveyor will be enclosed in a gallery and feed the material into the top of the crushed ore bin. A transfer conveyor will feed from the ore bin to the existing SAG mill feed conveyor.

18.5.2 SAG Mill Feed Conveyor

An existing 36” conveyor from the headframe to the SAG mill, housed within a gallery, will be utilized to feed at a rate of 35.1 t/h. The existing conveyor will be completely refurbished, and outfitted with new impact receiving idlers and a new transfer chute from the crushed ore bin discharge conveyor. The gallery will be modified for the new transfer conveyor feed from the ore bin, otherwise will remain unchanged.

18.5.3 Grinding and Classification Circuit

The existing grinding circuit will be upgraded with a new ball mill installation and support equipment replacements. The support equipment to be newly installed includes the new cyclone cluster, slurry pumps, and SAG mill discharge screens.

The new ball mill is larger than the existing unit, however, a model review shows that the new mill will fit in the existing location with some foundation and structural platform modifications. The existing foundations will be used for anchoring and additional concrete and reinforcement will be installed as required for the new mill.

The existing SAG mill will be upgraded with a new motor and given a mechanical refurbishment, including:

- Alignment of drive train components;
- Rework of sole plates as required;
- Bearing and gear inspection; and
- Lubrication systems overhaul and refurbishment.

The existing medium voltage soft-start drives will be replaced with 600 V variable frequency drives (VFDs) for both mills. The existing bridge crane will be modified/replaced with the new mill installation.

18.5.4 Gravity Circuit

A new gravity circuit consisting of two concentrators, vibrating screens, and an intensive leach rector, will be installed. The leached solution will be pumped to the gold room and the leached solids returned to the milling circuit.

18.5.5 Leaching Area

The leaching area will use the existing pre-leach thickener and leach tanks, and have an additional pre-aeration tank and agitator installed beside the existing circuit. A new trash screen and new thickener feed and underflow pumps will be installed.

The pre-leach thickener and leach tank agitators will be inspected and refurbished to operational status. Pumps that aren't replaced will be rebuilt. To ensure 95% system availability, complete redundancy in pumping systems will be introduced, with installed spares available for service.

18.5.6 CIP Circuit Equipment

The existing CIP tanks, agitators and inter stage screens will be used, and the system pumps will be replaced with new pumps. New carbon transfer and safety screens will be installed.

18.5.7 Carbon Circuit

The main addition in the carbon circuit will be the carbon strip circuit heating system, planned to be an electric fired boiler. The existing carbon regeneration equipment will be used as is with minor refurbishments if required. New pumps and carbon screens are planned to be installed in this circuit.

18.5.8 Electrowinning and Refining

New gold electrowinning and refining equipment will be installed in the existing gold room at the process plant. This includes two EW cells and rectifiers, associated pumps and equipment, and complete set of equipment for producing doré from the EW sludge. The existing gold room will be secured and equipped with a monitored security system including cameras.

18.5.9 Reagents

The existing reagent area will be used for the handling, storage and mixing of the process reagents. The existing overhead crane, tanks, and piping will be used where possible. Existing reagent systems will be refurbished where necessary, and upgraded with modern dosing pumps. The lime system will be cleaned and refurbished to working order using the existing silo and mix system.

18.5.10 Cyanide Destruction Equipment

A new cyanide destruction circuit, consisting of two tanks, agitators, and discharge pumps, will be installed on a concrete foundation next to the plant.

18.5.11 Hydraulic Backfill Plant

The backfill mixing plant required for producing cemented hydraulic backfill, as described in Section 16.8, will be installed next to the process plant. An engineered structural lean-to will be added to the process plant building to enclose the backfill equipment. The tailings surge tank, and process water tank for backfill will be insulated and installed outside next to the building.

The backfill plant electrical controls will be installed and operated under the process plant control system.

18.5.12 Plant Controls

The plant control system, as described in Section 17.5 will be installed in the existing MCC rooms and control room of the processing plant. Existing wiring and cable trays are all in good shape, and will be used for the existing equipment and equipment replacements.

18.5.13 Plant Utilities

The plant utilities upgrades include an oxygen generator plant, and new process water pumps. The existing main compressor will be suitable for service with minor (filter) maintenance. The new pumps will be installed for process water distribution, gland water, and grinding water systems.

The existing water tank next to the plant and the firewater pumps and system will be utilized.

18.5.14 Assay Laboratory

A complete sample preparation and assay laboratory will serve as the main site metallurgical testing and analytical facility for the process plant and geology department. It will be located in the existing process plant building laboratory rooms. Pre-fabricated containers will be used for additional working room and storage.

18.6 Support Infrastructure

18.6.1 Existing Facilities

There are a number of existing facilities on the Project site that will continue to be used during the construction and operations period. These are described in the following sections. The site is serviced with municipal water and sewer services for the operation’s needs.

18.6.1.1 Office Building

A modular office building exists on site supporting the current site activities, approximately 8 m x 20 m in size. This office will be re-purposed as the geologists’ office facility once the mine is in operation.

18.6.1.2 Training Facility

Next to the modular office is a smaller building, approximately 5 m x 12 m, serving as a lunch and washroom facility for employees. This will be re-purposed as a training and meeting room for operations.

18.6.1.3 Maintenance Shop

An existing steel building, approximately 13 m x 25 m, serves as the site maintenance shop shown in Figure 18-4. It will continue to serve as the mobile equipment and site maintenance facility during operations.

The building has ventilation for welding and an air make up system, air compressors, and basic mechanical tooling. An overhead door, 5.5 m x 6 m high allows for mobile equipment access. A 12.1 m (40') container is has been modified to serve as lube and grease storage facility.

A 7.5 t overhead crane will be installed, and it will be outfitted with additional maintenance tooling and fixtures to service mobile equipment.

Figure 18-4: Maintenance Shop

Source: JDS (2019)

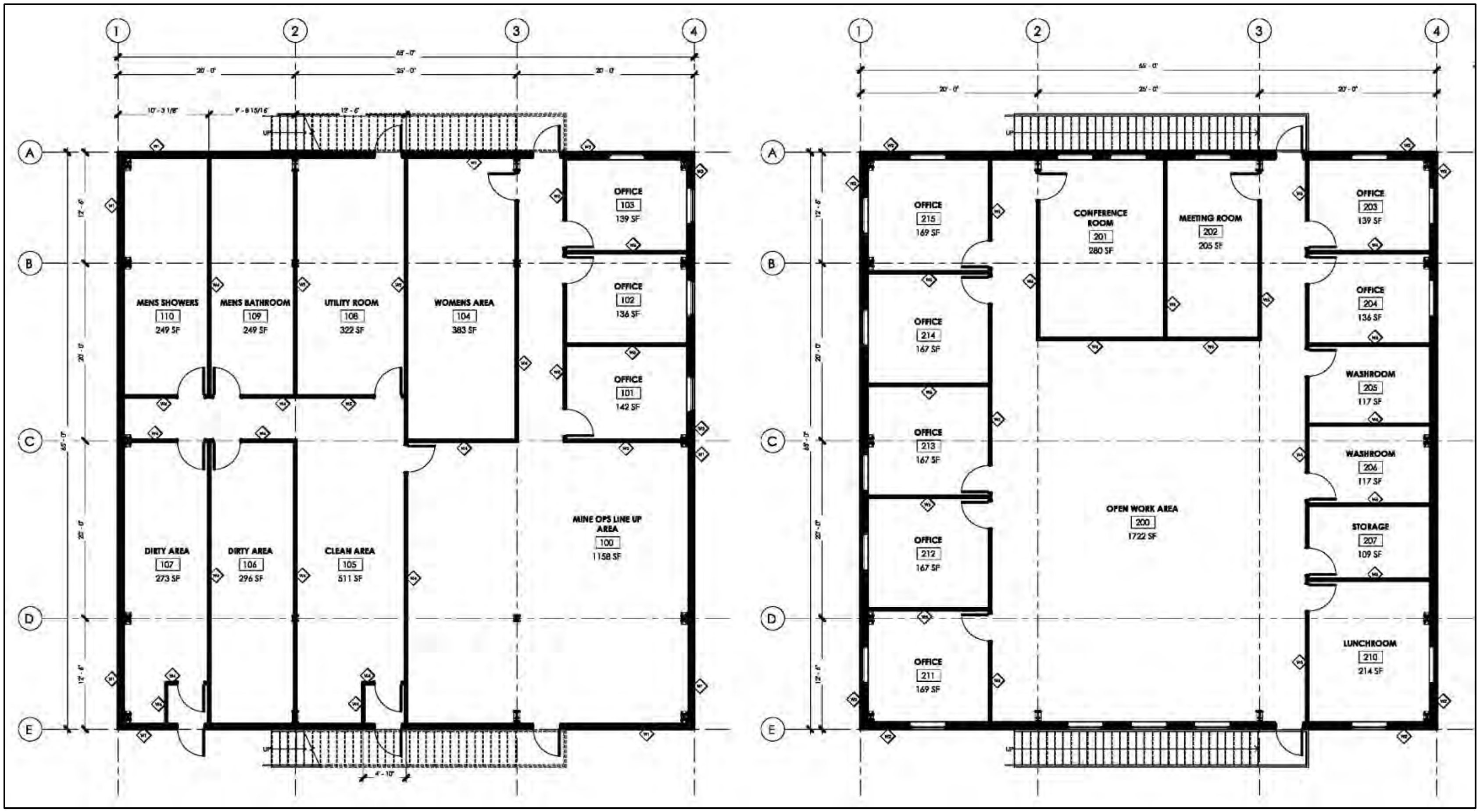
18.6.2 Mine Office / Dry Facility

A new two story pre-engineered building will serve as the administration office and mine dry. The design will consist of an 18 m x 20 m floor plan, with personnel locker and shower facility on the lower floor, and offices on the upper floor. A floor plan for each level is provided in Figure 18-5.

The lower level mine dry will service an 80 person shift, with lockers and storage for two shifts at a time. A separate women's area will accommodate up to 12 people.

The upper level area will provide offices for the site management, administration and mining technical staff.

Figure 18-5: Administration Office and Mine Dry Floor Plans



Source: Thomas Design Builders (2018)

18.6.3 Warehouse

The warehouse will be a shelter constructed from lockable 12.1 m (40’) storage containers in a U-Shape, with a sprung structure roof and concrete slab floor. One shipping container will be set up with a clerical office for control, and the facility will be enclosed and secured.

18.6.4 Core Facility

A core facility of using similar construction as the warehouse, will be set up in year one of operations. It will consist of modular units set up as enclosure, with a structural roof between and cement slab floor. It will be set up with two core saws in isolated area with dust control, and a separate working area for logging and boxing of core.

18.6.5 Security and Site Access

All public access will be through a single controlled security entrance. A pre-fabricated container will serve as the site security entrance control facility, and will be equipped with site monitoring systems. Existing security gates and fencing will be used, and new gates installed as required at road accesses. CCD cameras will be installed at the entrance gates and in the gold room. Only authorized personnel will be able to access the site facilities and operations.

18.6.6 Propane

Propane storage tanks of a suitable size would be located close to the portal area to be used for the initial and subsequent mine air heating system. Once the main ventilation and heating system is commissioned at the fresh air ventilation raise, storage tanks of a suitable size would be installed close to the heater. Propane will also continue to be used at the warehouse and maintenance shop.

18.6.7 Fuel Storage

The fuel storage facility would consist of one double walled 70,000 L capacity reservoir for diesel. A mine service fuel truck would deliver fuel underground to mobile equipment where applicable

18.6.8 Site Roads

Existing site roads will be upgraded for either mine rock haul travel or maintenance access. The haul roads will be for two way traffic, with 8 m useable width and safety berms. The road to the WTP, past the MRMF will be 6 m wide, and remote pipe and pump maintenance access roads will be 4 m wide.

18.6.9 Noise Abatement

Noise abatement structures will be installed to deflect nuisance noise from operations affecting the nearby town of Madsen. The haul road from the portal to the crusher site will have a 4 m high un-mineralized rock berm along the Madsen side of the road. An additional 4 m high berm, with a 3 m high wooden fence structure on top will be installed close to the crusher for noise abatement.

18.7 Mobile Equipment

A list of mobile equipment has been estimated for the site services is shown in Table 18-3. This service equipment is planned to be purchased by the Owner:

Table 18-3: Site Service Mobile Equipment

Equipment Required	Operating Description	Qty
Staff HD Pickup	¾ T pick-up trucks for staff site services	2
Staff Pickup	½ T staff trucks	2
Mechanics Truck	Site Services	1
Skid-steer Loader	Site Services	1
3T Forklift	Warehouse	1
20T Picker Truck	Site Services	1
Pipe Fusing machine (6"-18")	Maintaining site piping	1
Portable 25kW Generator	Site Services	1
Portable Diesel heaters	Site Services and Plant Maintenance	2
Portable Diesel Generator Light Plants	Safety lights for night work and maintenance	2

Source: JDS (2019)

Surface equipment for operations to handle the ore and mine rock will be provided by contractor services and equipment. The contractor's equipment will do the material handling of mine rock from the portal or hoist to the crusher and MRMF.

18.8 Waste Management

18.8.1 Solid and Hazardous Waste Management

18.8.1.1 Non-Hazardous Solid Waste

Local municipal landfill disposal will be used for inert solid waste disposal for construction and operations. The current practice of using local disposal will be continue to be used for non-hazardous construction waste, with disposal bins located near the main site, process plant, and at the portal.

18.8.1.2 Hazardous Waste Disposal

Anticipated hazardous wastes consist primarily of oils, process reagents and laboratory chemicals. Waste oils will be incinerated or recycled by the supplier. Most reagents and chemicals will be consumed during the processing, and the remainder will be recycled with the supplier.

All cyanide containers or packaging and other reagent containers will be washed using fresh water. Washing will be done in contained areas at the re-agent warehouse and storage area. Washing of cyanide containers will comply with the International Cyanide Code standards for disposal of cyanide products. Neutralized products and containers will be disposed of or recycled with the supplier in accordance with local regulations.

Laboratory fire assay wastes may contain small amounts of lead. These wastes along with any lead contaminated dust from the bag houses will be disposed of in accordance with local regulations.

Clean up of spills of hazardous materials on site will be given the highest operating priority and will generally involve the excavation of contaminated soils, neutralization of the affected site, and disposal and/or neutralization of the affected soils on site or at a licensed facility off site. The site surface equipment will be made available for use in such circumstances.

18.9 Tailings Management

18.9.1 General

The TMF at the Project has been in operation since the late 1930s and has gone through several design modifications. The feasibility design for the planned operations is based on an expansion of the existing TMF, with upgrades to meet current design standards. A portion of the new tailings (approximately 1.2 Mt) will be used for underground mine backfill and the remainder (approximately 2.3 Mt) will be managed on surface in the TMF. The principal design objectives for the TMF are to provide safe and secure storage of tailings while protecting groundwater and surface waters during operations and in the long term (after closure), and to meet regulatory mine closure and reclamation requirements.

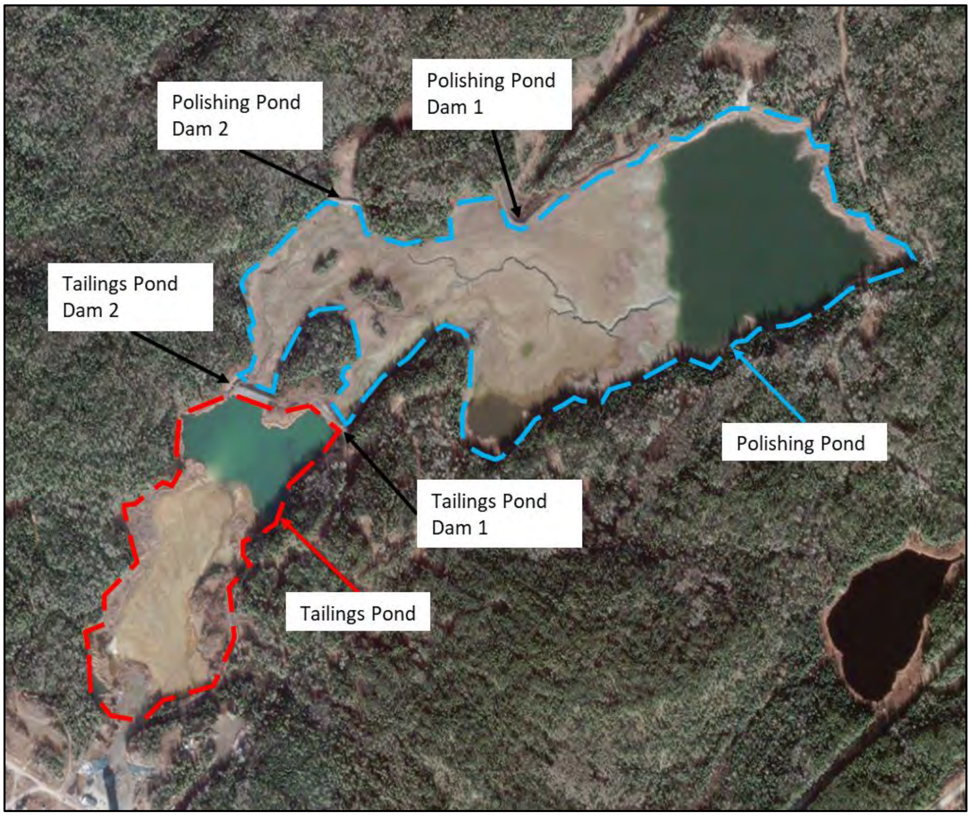
18.9.2 Existing TMF

Tailings were originally placed in depressions near the mill with no containment structures. Two dams were constructed to contain the increasing volume of tailings in the 1950s. These dams were referred to as Polishing Pond Dam 1 and Polishing Pond Dam 2. The TMF was upgraded in 1997 before mill operations restarted. The upgrade consisted of the construction of two rockfill dams across the tailings deposit, dividing the original tailings area into the Tailings Pond and the Polishing Pond. These dams are referred to as Tailings Pond Dam 1, and Tailings Pond Dam 2. Other upgrades included:

- Construction of a concrete stop-log control structure at the east abutment of Tailings Pond Dam 2 to regulate the discharge flow from the Tailings Pond;
- Upgrades to Polishing Pond Dam 1 and Polishing Pond Dam 2 to address stability issues; and
- The existing decant at the discharge of Polishing Pond was upgraded and used to address flood routing concerns and regulate the discharge flow from the Polishing Pond.

The existing TMF general arrangement is shown on Figure 18-6.

Figure 18-6: Existing TMF General Arrangement



Source: KP (2019)

18.9.3 TMF Geotechnical Characterization

The TMF geotechnical design parameters were developed from an extensive geotechnical database that incorporated data from previous site investigations and from a geotechnical site investigation program completed in 2018.

The following is a typical sequence of in-situ material types encountered at the TMF:

- Rockfill dams;
- Historic tailings within the tailings and polishing pond impoundments;
- Organic deposits;
- Upper glacifluvial deposit, only encountered in the south valley of the tailings and polishing ponds;
- Glaciolacustrine deposits;
- Fine-grained glaciofluvial subwash deposits;
- Glacial till deposits, found only in the south valley of tailings and polishing ponds; and

- Mafic volcanic bedrock.

18.9.4 Design Basis

The high level design parameters for the TMF are summarized below:

- Mill Site Elevation = 420 masl;
- Mill Throughput (Typical) = 800 t/d;
- Operating days/year = 360;
- Mill availability = 95%;
- Ore Mined = 3.5 Mt;
- Total tailings = 3.5 Mt;
- Total tailings to the TMF = 2.3 Mt;
- Total tailings to the Underground Backfill = 1.2 Mt;
- Tailings Specific Gravity = 2.94;
- Whole Slurry Tailings Solids Content = 50%;
- Cyclone Overflow Tailings Slurry Solids Content = 6%; and
- Mill and Overflow tailings pumped to the TMF in single pipe.

18.9.5 Tailings Characteristics

Laboratory testing was conducted to determine the geotechnical characteristics of the tailings. Representative samples of whole tailings and cyclone overflow tailings were obtained. The test program included index testing to enable geotechnical classification of the materials followed by slurry settling, air drying, consolidation and permeability testing to determine the tailings characteristics.

The tailings solids specific gravities were determined to be 3.05 for the whole tailings and 2.98 for the cyclone overflow tailings. Pure Gold defined the specific gravity for the design to be 2.94 based on all laboratory testing carried out for the project.

Whole tailings samples are described as non-plastic while the cyclone overflow tailings samples are determined as low plasticity, with a plasticity index of 2. The whole tailings classify as a silt with sand (ML) with 84.7% of the material passing the #200 sieve, and a D50 of between 30.5 µm and 30.8 µm. The cyclone overflow tailings are significantly finer, classified as a silt with some clay (ML), with 99.7% of the material passing the #200 sieve and a D50 of between 6.4 µm and 7.3 µm. Average settled dry densities for the whole tailings samples range from approximately 1.3 t/m^3 at very low stresses to 1.6 t/m^3 at high stresses. Average settled dry densities for the cyclone overflow tailings samples range from approximately 1.0 t/m^3 at low stresses to 1.4 t/m^3 at high stresses.

18.9.6 Tailings Management Strategy

The TMF is partitioned into four cells, designated Cell A, Cell B, Cell C and the Polishing Pond, to provide storage for tailings during operations and to achieve the closure objectives. Tailings will be deposited in the TMF in two separate phases:

- Phase 1 (Year 1 – 6): Tailings deposition in Cell A; and
- Phase 2 (Year 7 – 13): Tailings deposition in Cell C and Polishing Pond.

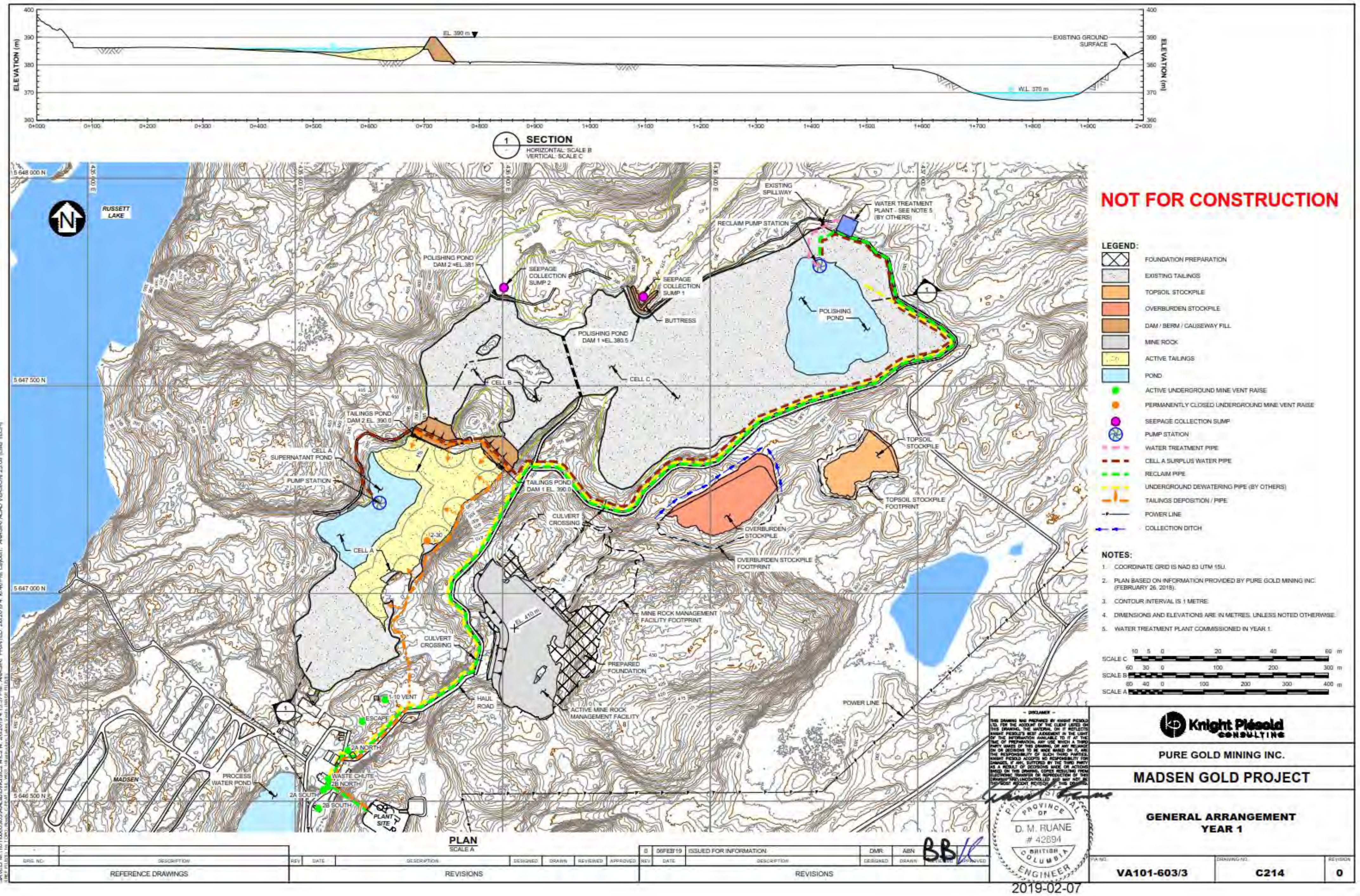
Phase 1 Tailings Deposition

Processing and tailings production begins at the start of Year 1. Tailings containment will be provided in Cell A by raising the existing Tailings Pond Dams. Tailings will be deposited from the Tailings Pond Dams and from the east side of the facility to develop tailings beaches against the dams and control the location of the supernatant pond so that water can be efficiently reclaimed to the Polishing Pond. Tailings will be delivered to Cell A in a single tailings pipe, delivering whole mill tailings and overflow tailings from the underground backfill cyclone activities. Surplus water accumulation in Cell A will be pumped or gravity decanted directly to the Polishing Pond for storage prior to treatment and release. Cell A will reach its maximum capacity in Year 6 and will be closed and reclaimed during the following two years of operations. Cell B, which is part of the existing TMF, will remain in operation until Cell A has reached its maximum capacity so seepage leaving Cell A can continue to flow to the Polishing Pond. The TMF layout at the end of Year 6 is shown on Figure 18-7.

Phase 2 Tailings Deposition

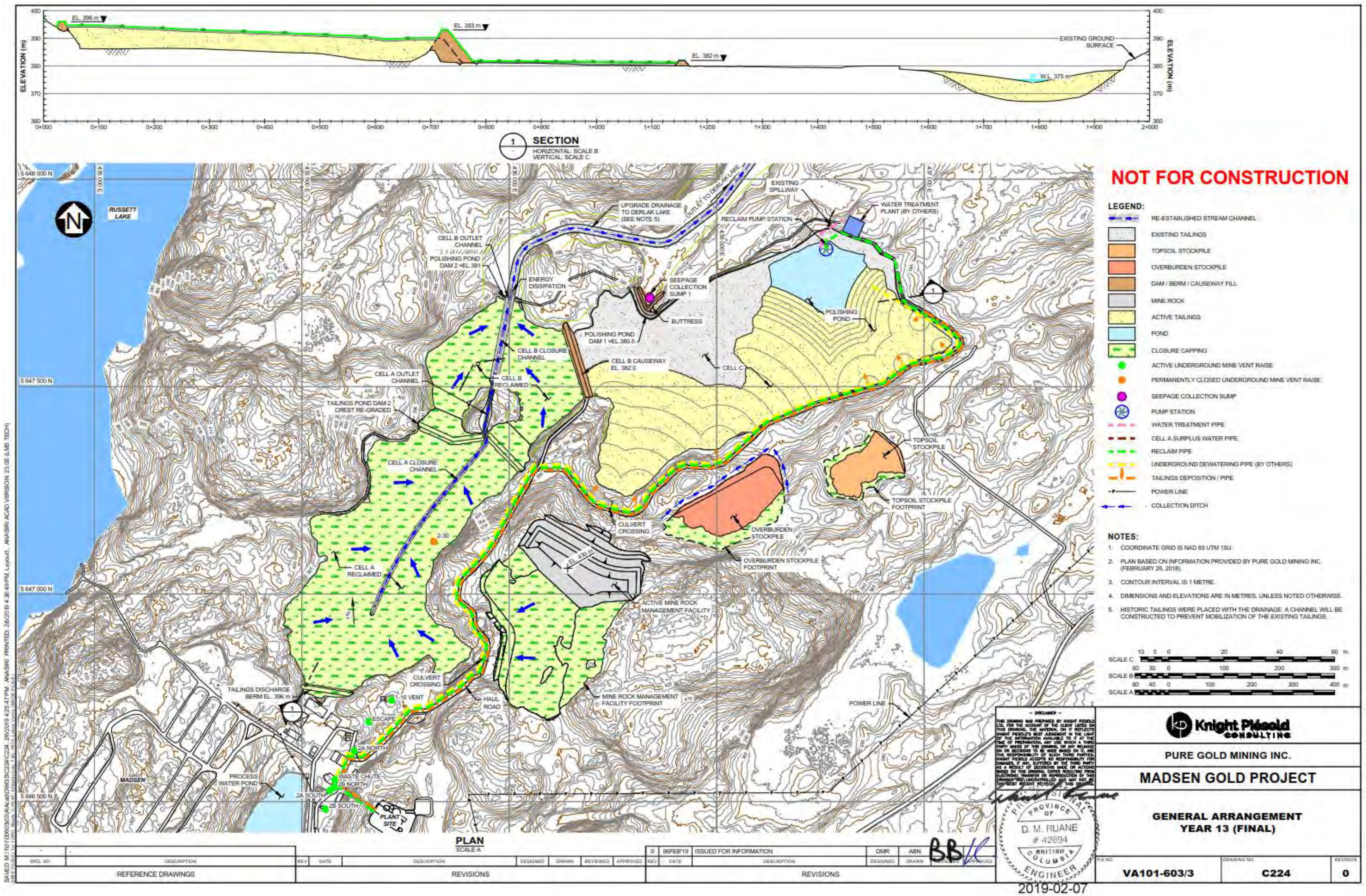
The existing confinement within Cell C and the Polishing Pond will provide containment for the remainder of the tailings. Slurry tailings deposition will commence into the natural confinement of the Polishing Pond in Year 7. Tailings will be deposited at the northeast end of the Polishing Pond to begin filling the topographic depression and directing the supernatant pond towards the reclaim pump located at the north end of the facility. At the end of Year 8, Cell A and Cell B will be fully closed with an engineered closure cover and non-contact runoff from this area will be directed downstream of the TMF by re-establishing the natural run-off in a closure channel. The remaining catchment and runoff from Cell C and the Polishing Pond will report to the Polishing Pond. Cell C and the Polishing Pond will be regraded using selective tailings deposition as part of the closure objectives and will be fully closed with an engineered closure cover. The TMF layout at the end of operation in the first quarter of Year 13 is shown on Figure 18-8.

Figure 18-7: Overall Site Plan General Arrangement



Source: KP (2019)

Figure 18-8: Overall Site Plan General Arrangement (Year 13 Final)



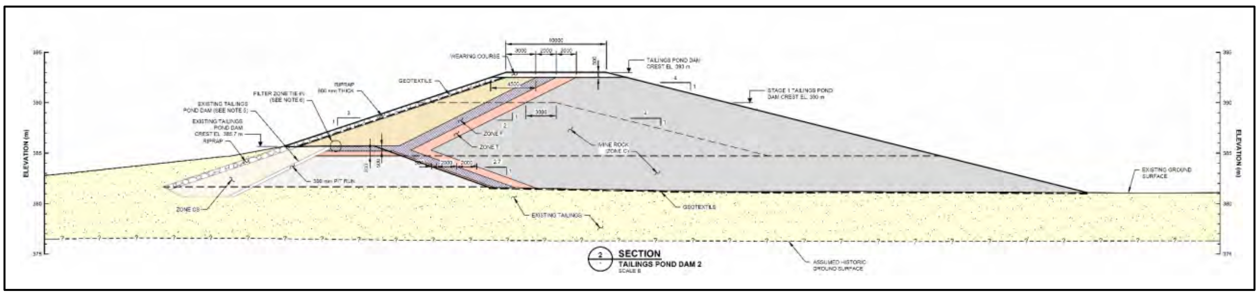
Source: KP (2019)

18.9.7 Tailings Pond Dams Expansion

Tailings Pond Dam 1 and Tailings Pond Dam 2 will be expanded and operated as free draining structures. The dams will be raised using the downstream construction method. They will comprise zoned structures that will have a low permeability compacted tailings upstream zone with appropriate filter and transition zones to prevent downstream migration of tailings and fine material through the rockfill. A layer of non-woven geotextile and erosion protection material will be placed along the upstream embankment face directly on top of the compacted tailings zone for erosion protection.

The dams will be raised in two stages to distribute capital costs. The dams will be raised to elevation 390 m during pre-production and to a final elevation of 393 m by the end of Year 3. Seepage through the dams will be controlled by the upstream compacted sand, tailings beaches and by minimizing the pond volume stored in Cell A. Seepage collected in the drainage zones will flow to the Polishing Pond through Cell B and Cell C. The downstream shell zone designated Zone C will support the tailings sand and filter zones. The Zone C shell zone will be constructed using pervious Mine Rock. A typical cross section through Tailings Pond Dam 1 at the end of construction is shown on Figure 18-9.

Figure 18-9: Tailings Pond Dam 2 Cross-Section – End of Construction



Source: KP (2019)

Instrumentation will be installed in the Tailings Pond Dams and underlying foundations and monitored during construction and operations to assess performance and to identify any conditions that differ from those assumed during design and analysis. Amendments to the ongoing designs, operating strategies and/or remediation work can be implemented to respond to changing conditions, should the need arise.

18.9.8 Seepage Collection System

Seepage is currently observed at the downstream toe of the Polishing Pond Dams. Seepage is a result of infiltration of ponded water through the tailings mass and the embankment fill as well as through the foundation and natural ground.

Seepage collection sumps will be constructed at the downstream toe of Polishing Pond Dams to collect existing seepage flows. Additional seepage collection ditches will be constructed along the toe of the dams to direct seepage and surface run-off to the sumps. The drainage systems will convey water to the seepage collection sumps for flow measurement and recycle back to the Polishing Pond. The sumps will be excavated, backfilled with coarse drain rock and will include vertical riser pipe wet wells with screens.

Seepage flows will be pumped back to the Polishing Pond seasonally using portable pump back systems to prevent freezing lines in winter months.

Seepage flows through the Polishing Pond Dams will be minimized in future operations by keeping ponded water away from the dam structures. Additional compaction of the tailings upstream of the dams can be carried out to densify the tailings and decrease the permeability.

18.9.9 Emergency Spillways

The operating basis for the TMF includes temporary storage of the Environmental Design Flood (EDF). This storage provision has been incorporated into the staging of Cell A for the Cell A catchment and within the Polishing Pond for the entire TMF catchment. Extreme inflows larger than the EDF exceeding the available storage capacity will discharge by way of emergency spillways constructed in Cell A and 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.

18.9.10 Mechanical Systems

The tailings distribution system is designed to deliver the tailings (whole tailings and cyclone overflow 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 will consist of a pump station and a single pressurized pipe to deliver water from the Polishing Pond to the Process Plant. The reclaim pump station will consist of two floating vertical turbine pumps (one operational and one standby).

The surplus water system will consist 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 will consist of two floating centrifugal pumps (one operational and one standby).

The WTP supply system will consist of a single pressurized pipe to deliver water from the Polishing Pond to the WTP holding tank. The WTP supply system pump station will consist of two floating vertical turbine pumps (one operational and one standby). The pumps will share a floating barge with the reclaim system.

18.10 Mine Rock Management

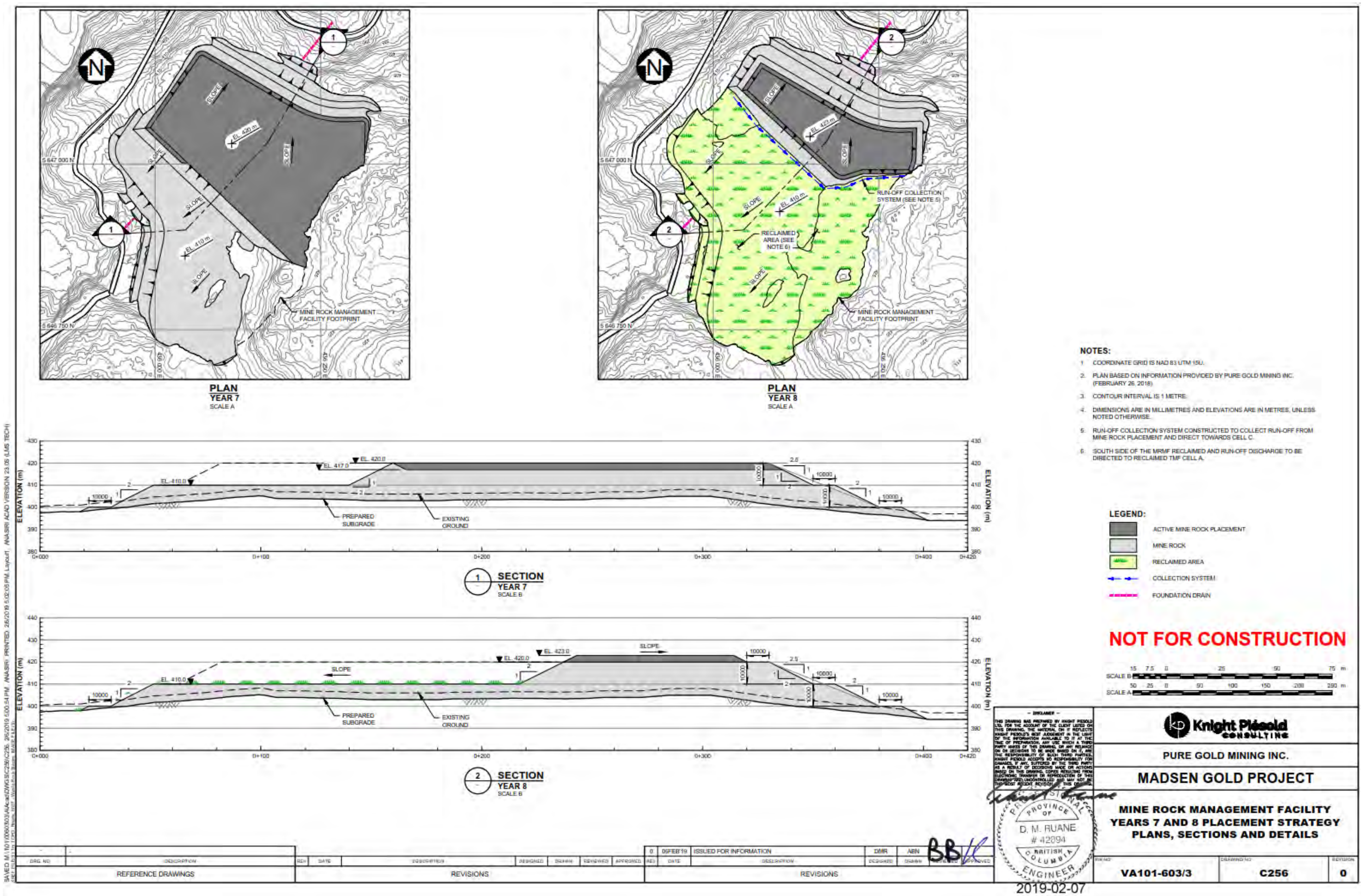
Approximately 4 Mt of mine rock material will be produced over the mine life. Surplus mine rock from underground mining development will be stored on surface in the MRMF located east of Cell A. Additional mine rock required for underground backfill and construction activities will be sourced from this stockpile. Mine rock will be used at closure to cover the TMF and this capping material will also be sourced from here. The MRMF is designed to store a maximum of 1.1 Mm^3 of mine rock over the life of the Project with approximately 350,000 m^3 of mine rock remaining in the facility at closure. A total of 865,000 m^3 of mine rock will be removed from the MRMF in Years 7 and 14 as part of the concurrent closure of the TMF. Non-acid generating mine rock will be used as part of the closure capping system. The general high level design parameters are summarized below:

- Total Mine Rock = 4.0 Mt;
- Mine Rock to MRMF = 2.7 Mt;
- Mine Rock Directly to Construction Activities = 0.3 Mt; and
- Mine Rock to Underground Backfill = 1.0 Mt.

The MRMF will be developed in two stages to defer site preparation capital costs. Stage 1 is on the south side of the facility and will provide storage for three years of operations. The Stage 1 MRMF naturally drains to Cell A and surface water run-off from the facility will be directed to Cell A. Stage 2 is on the north side of the facility and will provide storage for the remaining years of operation. The Stage 2 MRMF naturally drains to Cell C and surface water run-off from the facility will be directed to Cell C.

The MRMF is designed with 10 m-high lifts and typical 10 m-wide benches (2H:1V bench face slopes). The toe elevation of the proposed dump is approximately at 394 masl. The maximum crest is at 430 masl and the maximum height of the un-mineralized rock along the typical cross section is approximately 30 m with a resulting overall slope of approximately 2.5H:1V for closure and reclamation. The MRMF will be concurrently reclaimed to facilitate closing the south side of the facility in Year 8 to coincide with water management objectives in the TMF. The Mine Rock placement strategy in the MRMF in Year 7 and Year 8 are shown on Figure 18-10.

Figure 18-10: MRMF (Year 7 and Year 8)



Source: KP (2019)

18.11 Water Management

18.11.1 Water Management Strategy

Water management is a critical component in the overall mine waste management strategy. All mine contact water, tailings slurry water, underground mine dewatering flows and groundwater accumulating in the TMF will be stored and managed within the Polishing Pond. Seepage collected in collection sumps located downstream of the Polishing Pond Dams will be pumped back to the Polishing Pond seasonally until these dam areas are progressively reclaimed.

Surplus water will be treated and released downstream of the TMF to maintain sufficient storage capacity in the Polishing Pond for the operations. The release of treated water is limited by permitting which states no water can be released while water in the receiving waterway is frozen. It is assumed that water will not be released from November 30 to March 15 each year. Pumping from the underground mine and tailings slurry water inflows to the TMF are predicted to be constant throughout the year so sufficient capacity must be maintained in the Polishing Pond to allow for water accumulation in the facility during the no release period.

Phase 1 tailings deposition involves depositing tailings in Cell A. Run-off and tailings slurry water will be pumped to the Polishing Pond from Cell A. The Polishing Pond will need to be drawn down to elevation 370 m prior to start-up of operations to provide sufficient capacity for contact water accumulation in Year 1 before the WTP is commissioned and water can be treated and released.

Phase 2 tailings deposition involves depositing tailings in Cell C and the natural confinement of the Polishing Pond and as a result, reducing the containment volume for site contact water. Filling the depression in the Polishing Pond is a fundamental component of the closure plan so that the TMF surface can be regraded and all ponds removed. Water management objectives are achieved by concurrently reclaiming the TMF during operations so that the contributing catchment to the TMF is reduced and hence the total project surplus contact water to be managed is reduced.

Sufficient capacity for mine operations, underground dewatering and storm storage will still need to be maintained. The following water management strategies will be implemented throughout the pre-production and operations of the TMF:

- Reclaim water will be pumped directly from the Polishing Pond to the Process Plant;
- Underground dewatering flows will be pumped directly to the Polishing Pond, except during Pre-production Phase 1 when it will be discharged to the downstream environment;
- Site contact water will be managed in the Polishing Pond during operations;
- Excess water in the Polishing Pond will be pumped to the WTP, treated and released at the current effluent discharge point downstream of the TMF;
- Treated water can only be released from the WTP during periods when the receiving waterway is not frozen; and
- Water treatment and release required each year to maintain sufficient capacity in the Polishing Pond for the upcoming year of operations.

18.11.2 Water Balance

A water balance model was prepared, and the results indicate the TMF will operate in a surplus during all phases of operations and under the full range of variable climatic conditions, including prolonged wet and dry cycles. 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 depends on the phase of the Project and is the greatest during the maximum footprint of the TMF, which is Years 1 to 8 of operations. The TMF water balance demonstrates that the water management strategy can be executed under the full range of variable climatic conditions available based on the inputs presented. This includes achieving the reclaim water requirements under prolonged dry climate cycles for the entire mine life and determining that there is sufficient operating capacity within the TMF to manage surplus water during prolonged wet climate cycles.

18.12 Closure and Reclamation

The closure design philosophy for the TMF involves removing all surface water ponds at the end of operations and covering the tailings surface with an engineered cover system, which naturally sheds non-contact run-off to the downstream environment. This closure plan would eliminate the need for long-term water treatment post closure. The objective of the closure and reclamation initiatives will be to eventually return the TMF site to a self-sustaining facility with pre-mining land capability.

The closure plan regrades the TMF into a landform with a native species vegetation cover to the surrounding topography. Closure channels will be constructed in Cell A, Cell B, Cell C and the Polishing Pond to direct flows downstream of the TMF. The emergency spillways constructed during operations will be upgraded to closure discharge channels to meet CDA Post-Closure Guidelines.

Reclamation of the MRMF will be completed at mine closure. The closure of the MRMF will typically include re-sloping of the face to a 2.5H:1V or flatter slope. The final dump bench crests will be smoothed or rounded to improve the long term erosional stability of the facility.

The MRMF will be concurrently reclaimed to facilitate closing the south side of the facility in Year 8 to coincide with water management objectives in the TMF. Mine Rock produced during Years 8 to 13 will be placed and managed in the north side of the facility to so that contact water in the active placement area naturally flows to Cell C.

Concurrent reclamation will be carried corresponding to closure objective of the TMF. Reclamation of the MRMF will be conducted in conjunction with on-going geotechnical and environmental monitoring to assess slope stability, sediment control, and water quality objectives.

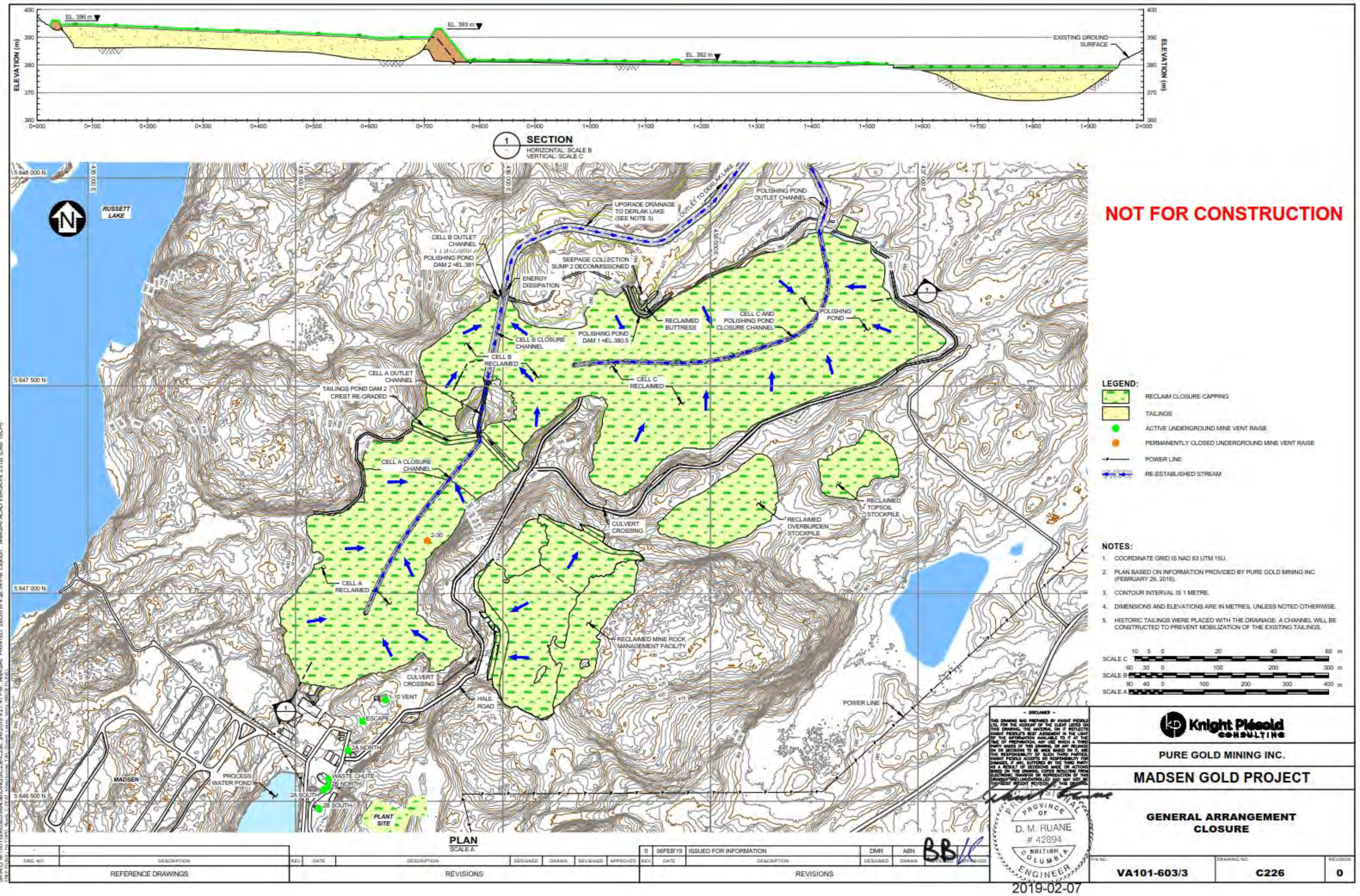
Closure monitoring at receiving waters will be measured against water quality objectives. The following items are planned for monitoring during closure:

- 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; and
    - Final environmental effects monitoring studies in accordance with water quality objectives needed to obtain status as a recognized closed mine from Environment Canada.

The overall TMF general arrangement at closure is shown on Figure 18-11.

Figure 18-11: General Arrangement – Closure Drawing



Source: KP (2019)

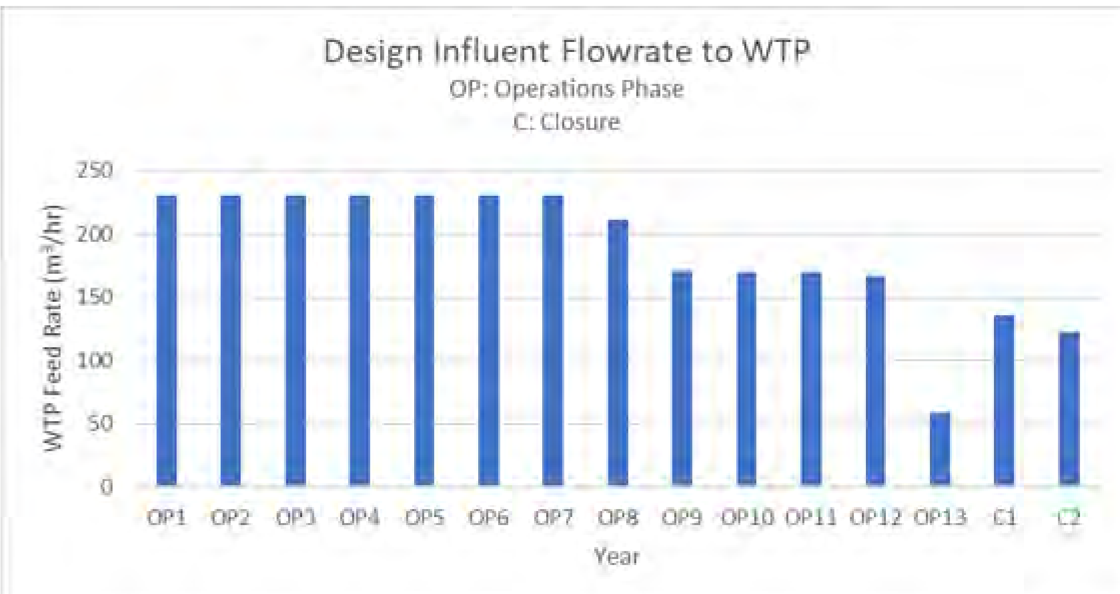
18.13 Water Treatment Plant

18.13.1 General

The Project requires a WTP which will treat the water contained in the Polishing Pond prior to discharge to the receiving environment. The WTP is designed to operate from March 15 to November 30 each year and treat up to 232 m^3/hr, with the Polishing Pond providing flow equalization. The treated effluent will be discharged into Derlak Lake where it will be monitored to ensure compliance with environmental requirements.

The WTP design capacity is based on results of the site wide water balance and resulting TMF annual surplus. Predicted WTP influent flow rates are shown in Figure 18-12 below.

Figure 18-12: Design In-Flows



Source: Integrated Sustainability (2019)

18.13.2 Process Overview

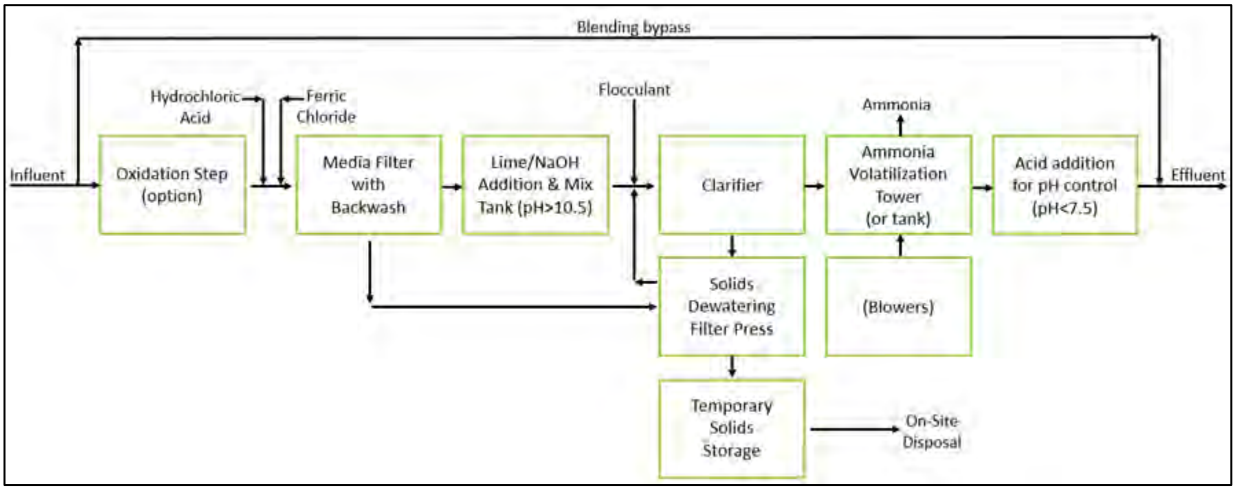
The WTP includes the following major processes:

- Chemical oxidation;
- Ferric chloride injection for arsenic precipitation;
- Multimedia filtration;
- Sodium hydroxide treatment metals precipitation;
- Clarification;
- Sludge dewatering of clarifier underflow and MMF backwash;

- Air stripping via packed column for ammonia removal; and
- pH control prior to release to environment.

A process schematic is shown in Figure 18-13.

Figure 18-13: WTP Process Schematic



Source: Integrated Sustainability (2019)

18.13.2.1 Chemical Oxidation

The first stage of the proposed treatment process is chemical oxidation. This treatment stage allows for the oxidation of dissolved metals into solid precipitates for removal via filtration, as well as conversion of arsenic from the As(III) state to As(V) state to facilitate more effective removal in subsequent treatment steps. Standard oxidizing agents such as hydrogen peroxide are sufficient for standard metals oxidation, whereas ozone oxidation may be required for arsenic oxidation state conversion and can be added as a modular treatment step to the front end of the plant.

18.13.2.2 Ferric Chloride Treatment

The second stage of the proposed treatment process is ferric co-precipitation using ferric chloride (FeCl3). In order to maximize the efficiency of arsenic removal in the ferric co-precipitation step, the incoming water is reduced to a pH of 3-4 prior to dosing with ferric chloride; the pH is then increased to pH 5+ to encourage the co-precipitation of arsenic with, and adsorption of arsenic onto, solid iron oxide precipitates that are generated. The ferric chloride provides additional coagulation benefits, encouraging the binding of metal hydroxide precipitates into larger particles for more efficient removal. The reaction is carried out in a mixing tank to allow for a complete reaction, with multimedia filtration following the mixing tanks to remove any solid precipitates generated.

18.13.2.3 Hydroxide Precipitation Treatment

The third stage of the proposed treatment process is metals removal via hydroxide precipitation. Following the multimedia filtration stage the water is dosed with sodium hydroxide (NaOH) up to a pH of 11 in order to promote the precipitation of metals such as copper, cadmium, nickel, zinc, and residual iron out of solution in the form of solid metal hydroxides. The reaction will occur in another mixing tank to ensure complete precipitation, with the water then being sent to a clarifier and injected with flocculant for settling. The use of an anionic, high molecular weight polymer flocculant will facilitate the formation of larger particles and enhance settling of the precipitated metal hydroxide solids in the clarifier.

The sludge generated by the hydroxide precipitation stage will be combined with the backwash of the multimedia filters following the ferric co-precipitation stage and sent for dewatering in a filter press. Assuming 45% solids content, the sludge generated will be 0.473 m^3/day; this sludge will contain primarily metal hydroxides and not be deemed hazardous waste, allowing it to be stored in a lined pond at surface or potentially mixed with paste backfill for underground disposal pending further geochemical study.

18.13.2.4 Ammonia Stripping

The final stage of the proposed treatment process involves ammonia volatilization and air stripping, followed by pH control prior to final discharge. At high pH levels ammonia in water is primarily present in the form of un-ionized ammonia (NH3); by maintaining a high pH prior to the volatilization stage, ammonia will be driven out of the liquid by passing the water through an aeration tank or packed column tower. Following the volatilization stage, the pH of the water will be carefully controlled and neutralized to ≤7.5 in order to ensure any residual ammonia is released predominantly in the form of non-toxic ammonium (NH4), thereby minimizing potential for toxicity non-compliance.

18.13.3 Effluent Quality

Concentrations of arsenic and ammonia in the TMF are expected to exceed the limits as outlined in the Metals and Diamond Mining Effluent Regulations (MDMER). The expected effluent quality resulting from the proposed treatment process is outlined in Table 18-4.

Table 18-4: Expected Effluent Quality

Parameter	Expected Influent	Expected Effluent
pH	7.2	≤7.5
Total Ammonia (mg-N/L)	16	5.3
NH_3 (mg-N/L) @ 15^0C, pH 7.5	0.283	0.045
Nitrite (mg-N/L)	0.027	Feed
Nitrate (mg-N/L)	15	Feed
Sulfate (mg/L)	1652	Feed
Calcium (mg/L)	208	Feed
Magnesium (mg/L)	11	7.5
Aluminum (mg/L)	0.09	0.07
Antimony (mg/L)	0.077	Feed
Arsenic (mg/L)	0.52	0.03
Cadmium (mg/L)	<0.00037	Feed
Chromium (mg/L)	<0.001	Feed
Cobalt (mg/L)	0.2	Feed
Copper (mg/L)	0.016	0.008
Iron (mg/L)	3.2	0.3
Lead (mg/L)	<0.01	0.005
Nickel (mg/L)	0.07	0.02
Zinc (mg/L)	0.13	0.02

Source: Integrated Sustainability (2019)

19 Market Studies and Contracts

19.1 Market Studies

No market study was completed on the potential sale of doré from the Madsen Project, but indicative gold refining terms were obtained from the Canadian Mint on January 27, 2019. The indicative terms were reviewed and found to be reasonable by QP Michael Makarenko, P.Eng.

This study recommends that as the Project advances towards development, a detailed marketing report and logistics study should be undertaken to ensure the accuracy of the terms. Table 19-1 outlines the terms used in the economic analysis.

Table 19-1: Net Smelter Return Assumptions

Parameter	Unit	Value
Gold (Au) Payable	%	99.97
Gold (Au) Refining Charge	US$/pay oz	0.38
Transportation	US$/pay oz	1.35
Royalties	%	-

Source: Royal Canadian Mint (2019)

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.

19.3 Royalties

The Probable Mineral Reserves for the Madsen Gold Project are not subject to any royalties. Royalties do exist on other lands on the Project and these are described in Table 4-2.

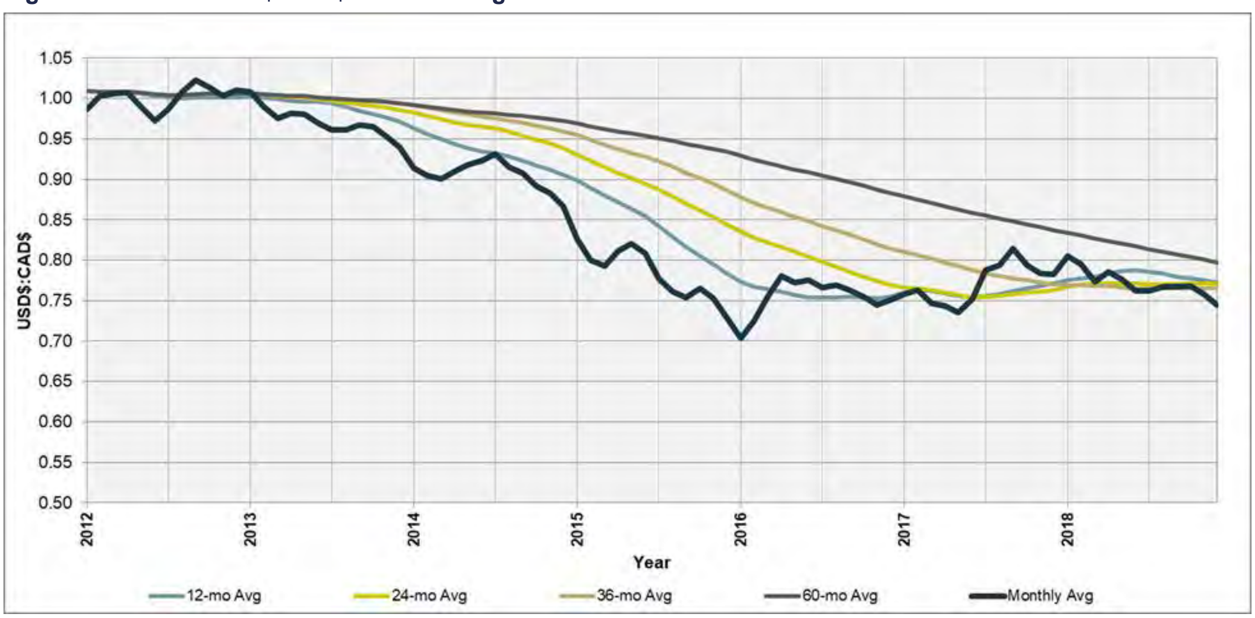
19.4 Metal Prices and Exchange Rates

The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical gold prices and exchange rates (USD$:CDN$) are shown in Figures 19-1 and Figure 19-2.

Figure 19-1: Historical Gold (Au) Price

Source: Kitco (2019)

Figure 19-2: Historical $USD:$CAD Exchange Rate



Source: Kitco (2019)

The selected gold price and exchange rate is discussed in Section 24.1.

20 Environmental Studies, Permitting and Social or Community Impacts

20.1 Introduction

Pure Gold is the 100% owner of the Madsen Project in the Red Lake Mining District. The project consists of both early and advanced-stage mineral exploration across a primarily patented claim property covering 47 square kilometers. The Property encompasses two former mining operations at Starratt Olsen and Madsen Mine. The Madsen Mine was in operation from 1938 to 1976 and again from 1997 to 1999 and is currently in a state of Temporary Suspension. The Starratt Olsen Mine operated from operated from 1948 to 1956 and is closed. The Properties have had several owners over the years. Old mines typically have legacy environmental issues, many that arise from now outdated standards and operating procedures that have improved over time.

Pure Gold purchased the Madsen Mine property in 2014 and subsequently consolidated additional surrounding properties. Pure Gold has embarked on a major program of investment in exploration and rehabilitation of the project site in preparation for potential renewed operations. The mine is currently in a state of Temporary Suspension, while Pure Gold undertakes advanced exploration drilling from both surface and underground.

20.2 Environmental Assessment

20.2.1 Federal CEAA Environmental Assessment Process

Pure Gold has 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 will not be classed as a designated project under the Canadian Environmental Assessment (CEA) Act (2012) as 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 has 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.

The requirements of the CEA Act are also potentially triggered when a federal authority (i.e. a department or agency) exercises one or more of the following duties, powers or functions in relation to a project. These are shown in Table 20-1.

Table 20-1: Summary of CEA Act Triggers and Reasons Pure Gold is Unlikely to Trigger a Federal Environmental Assessment

Potential Trigger	Reason Project is not Anticipated to Trigger CEA Act
1. The project is a federally sponsored project.	This is not a federally sponsored Project.
2. The federal government grants money or financial assistance to the proponent for purpose of enabling a project to proceed.	The project will not receive any federal grants money or financial assistance.
3. The federal government grants an interest in federal land to enable a project to be carried out.	The Project does not require the federal government to grant an interest in federal land.
4. A federal agency or department exercises a regulatory duty in relation to a project. The following Acts have been considered:	
Fisheries Act - authorization is required for the harmful alteration, disruption or destruction of fish habitat.	Fish do occur in streams and lakes downstream of the existing TMF, including Coin Creek, Derlak Lake, Snib Lake, Coin Lake and Red Lake. These waterbodies have been the focus of aquatic biology baseline studies and water quality predictions in compliance with the requirements from Ministry of Environment, Conservation and Parks of Ontario (MECP) for the purposes of an updated Industrial Effluent Permit under Policy 2 – no further harm beyond that of the historical legacy will be permitted. MECP require active water treatment prior to release due to the historical mining and naturally elevated background levels of some metals in these streams. Treated water of a better quality than that currently in these water bodies will be discharged and will meet or exceeding regulatory compliance and protect fish or fish habitat. Discussions are ongoing with MECP, Ministry of Energy, Northern Development and Mines, Ontario (ENDM) and Department of Fisheries and Oceans (Canada) (DFO).
Navigable Waters Protection Act – if the project potentially affects navigability through the construction or alteration of works on, over, under, through or across a navigable waterway.	No aspect of the project has the potential to affect navigable waterways.
Migratory Birds Conservation Act and Species at Risk Act - requires evidence that no impact on migratory birds or any species at risk by the project.	Based on investigations conducted to date and the activities proposed to re-open the Madsen Project, no impact on migratory birds is anticipated as a result of the proposed project. Pure Gold will develop a management plan to avoid migratory bird impacts by scheduling activities to minimize disturbance. Most activities are short duration allowing for scheduling alternatives.

Potential Trigger	Reason Project is not Anticipated to Trigger CEA Act
Species at Risk Act - requires evidence that there will be no impact on any species at risk by the project.	The Project site provides potential habitat for several Species at Risk (SAR) that are known to occur in the region. Of these only the Myotis bats have been encountered on the site to date. Pure Gold currently holds a SAR permit for Myotis species bats. Surveys for SAR species are ongoing by FRi Corp Ecological Services (FRi) and a staff biologist undertakes regular surveys and maintains a Pure Gold management system for SAR. As part of the management system should SAR species be encountered, Pure Gold will notify MNRF, as in the past, and prepare required Management and Protection Plans for the identified species.
Explosives Act - requires federal approval to locate explosives factory on project site.	Neither an Explosives Factory License nor an ANFO Manufacturing Certificate will be required for the mine as packaged explosives will be utilized.
Canadian Transportation Act - applies to certain projects where a rail line crossing or relocation is contemplated.	No railways are affected by the project.
Indian Act and Natural Resources Act - covers projects that are located on or require access through federal lands such as national parks, First Nation reserves, or national defense bases.	The land is all privately owned and no access through federal lands, First Nations Reserves, national parks or national defense bases is required.

Source: Pure Gold (2019)

20.2.2 Provincial Environmental Assessment

The MECP has indicated that there will be no requirement for the submission of a Provincial Environmental Assessment based on the presentation of the March 2017 Project Definition and the materials presented in December 2018 for the new Industrial Sewage Permit.

It is anticipated that the Pure Gold Madsen Project will require approvals from MECP, ENDM, Ministry of Natural Resources and Forestry (MNRF), Ministry of Labour (MoL) and Ontario Energy Board.

20.3 Environmental Authorizations and Permits

Pure Gold has worked to maintain the permits that existed for the Madsen Mine under previous operators. As the project has advanced, operational enhancements and regulatory changes require some of the permits to be updated. Permit status has been confirmed with both ENDM and Ministry of the Environment and Climate Change (MOECC) and the following permits and authorizations are in good standing:

- **Permit to Take Water (0202-AHJL45):** This permit was updated in 2017 and allows for the pumping of 6.5 million litres of water per day from the mine workings. No pumping is currently being undertaken.
- **Advanced Exploration Closure Plan:** In 2016, the Madsen Portal Advanced Exploration area was moved from Temporary Suspension to Advanced Exploration status to facilitate underground access for feasibility level studies. The closure plan for these activities was accepted by the regulators along with additional closure funding for the Advanced Exploration program closure. This

plan is in good standing and closure funding can be rolled over into a new mining operation or mine closure if a new operation does not proceed.

- **Advanced Exploration Closure Plan Amendment:** In 2018, Pure Gold requested an amendment to the Madsen Portal Advanced Exploration Closure Plan to allow for the taking of a bulk sample. This amendment was acceptable to ENDM and local First Nations and required nominal additional closure funding. This plan is in good standing and closure funding can be rolled over into a new mining operation or complete mine closure.
- **SAR Exemption and Benefit Program Under Clause 17(2)c of SARA for Endangered Bats:** Myotis species bats were encountered in the underground portal area in 2017. A permit allowing Pure Gold 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 of 2017. The permit is in good standing and MNRF has accepted two annual reports, 2017 and 2018, under this permit.
- **Registered Hazardous Waste Generation Site:** Pure Gold 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. This permit is renewed annually and is in good standing for 2019.
- **PCB Storage site closure:** Pure Gold has decommissioned and closed a legacy PCB storage site established by a previous operator and met all conditions set by MOECC. A closure confirmation letter has been received from MOECC.

The following existing permits are currently in good standing but would require updating due to process changes or regulatory changes:

- **Environmental Compliance Approval (ECA) Industrial Sewage Works Permit:** Pure Gold currently has an Amended Certificate of Approval for Industrial Sewage Works (0047-7V9PW9) that allows for the release of suitable quality water from the tailings facility. This ECA allows Pure Gold 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 conducted and annual reports are presented to MECP. The permit is in good standing.

  Changes in the effluent regulations since the mine and mill facility last operated as well as the water management described in the Project Definition will require an updated Industrial Sewage Works Certificate of Approval. Pure Gold is in the process of updating this ECA for future operations and has undertaken baseline studies focused on optimizing water resource usage, recovery and recycling and has presented an updated operation general arrangement in the Project Definition. JDS, Knight Piésold and Integrated Sustainability, were contracted to provide tailings and process water treatment designs and Lorax Environmental and Minnow the baseline studies for this work, which is aimed at bringing Pure Gold into compliance with the Metal Mining Effluent Regulation, Ontario Water Resources Act, Regulation 560/94 (Municipal Industrial Strategy for Abatement (MISA) Regulation) and the Metal Mining Effluent Regulation. Designs, baseline reports and predicted water quality modelling have been provided to MECP and ENDM as the first part of updating this permit. Full submission for a new permit application is expected by mid-2019 and a new permit is expected to be issued in 2019 to early 2020.

- **ECA for Air and Noise:** With the proposed use of new equipment and some operational changes to enhance power, energy and water usage, a new air and noise ECA is required. DST Consulting and Aeroacoustics were contracted to provide the baseline study for this work and a new permit application was submitted in 2018. A new ECA for Air and Noise is expected to be issued in 2019.
- **Mine Closure Plan, ENDM:** A Mine Closure Plan was submitted to ENDM on May 24, 1995 with Closure Plan Amendments in July 2011 and April 2014. Current funding is considered to be adequate for closure of the current mine in temporary suspension. Pure Gold has also submitted closure plans for the Madsen Portal Advanced Exploration. It is anticipated that funding from these plans will be rolled into a further Amended Madsen Mine Closure Plan in advance of mine reopening and that additional closure funding may be required for reopening.

  It should also be noted that the mine is currently in a state of Temporary Suspension and considerable site cleanup has been undertaken by Pure Gold outside of the closure plan funding. Given the new Project Definition which is focused on progressive reclamation and the considerable effort that has been made to clean up the site at Pure Gold's own cost it is anticipated that the 2019 Madsen Mine Closure Plan will only require limited additional funding.
- **Permit to Mine, ENDM:** A Notice of Project Status was received and acknowledged by ENDM on April 24, 2007. A new notice of Project Status will need to 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:** Any construction/relocation of a transmission line for work on Crown land or for in-water works.
  - **Plans and Specifications Approval:** For construction of dams or berms, including those associated with tailings facilities and/or new ponds and ditches.
  - **Forest Resource License:** Annual license for clearing of any Crown merchantable timber.
  - **Aggregate Permit:** Aggregate Resources Act For extraction of any aggregate for dam construction.
  - **Leave to Construct:** For approval to construct a transmission line.
  - **Notice of Construction:** Notice is required before any contractor or construction activities take place.
  - **SARA Approvals:** Although the area falls within the range of several terrestrial species at risk these have not been encountered on or near the site. FRi has been retained to evaluate habitat suitability and to undertake surveys and develop management plans prior to any construction and operational activities associated with mine reopening.

20.4 Consultation

Pure Gold has committed to engagement and consultation with local First Nations, Metis Nation of Ontario, provincial and federal governments, the public, and stakeholders throughout all stages of Project planning, regulatory review, and construction. The intent is to provide all interested parties with opportunities to learn about the Project, identify issues, and provide input with the goal of positively enhancing Project planning and development.

Pure Gold recognizes the importance of timely, full and open discussion of the issues and options associated with the development of the Project and the related concerns those individuals or communities may have in relation to the activities. In light of this, Pure Gold will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the Project. Pure Gold intends to 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. Pure Gold has captured these commitments in a Consultation Plan, the main contents of which are summarized below.

20.4.1 First Nations Considerations

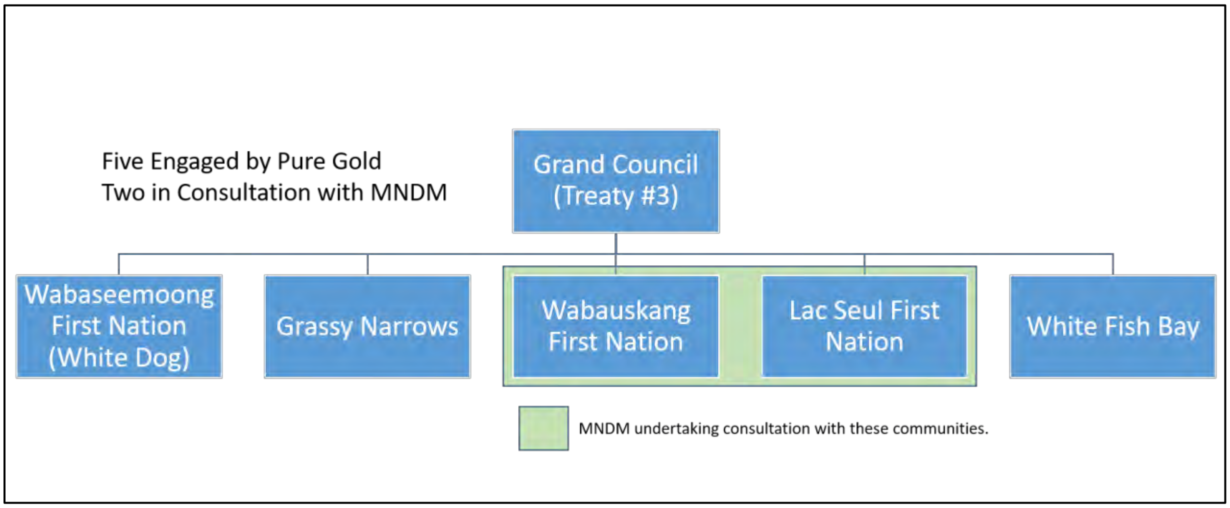
The property is located within the boundaries of Treaty # 3 (1873 and adhesions). Lac Seul First Nation and Wabauskang First Nation have identified the project area as lying within their communities' traditional territory (Figure 20-1).

ENDM has advised Pure Gold 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 ENDM will undertake and fulfill the Crown's Duty to Consult. Currently, the primary role of Pure Gold with First Nations is to ensure that appropriate information sharing occurs (Figure 20-2).

Figure 20-1: Treaty 3 First Nations

Source: Pure Gold (2019)

Figure 20-2: Treaty 3 First Nations Engaged by Pure Gold



Source: Pure Gold (2019)

Pure Gold proactively approached Grand Council Treaty #3 immediately upon acquiring the Madsen Project property to receive guidance on which Treaty 3 communities might have any interest in the project. Pure Gold has consistently provided advance notice of the nature, scope, timing and methodology of any proposed activities on site and has actively solicited feedback and guidance on the ways and means to conduct proposed activities in a respectful and professional manner. Pure Gold maintains contact and dialogue with the Treaty 3 First Nations of: Wabauskang First Nation; Lac Seul First Nation; Wabaseemoong First Nation; Grassy Narrows First Nation; Naotkamegwanning First Nation (Figure 20-2). This information is also shared with the Métis Nation of Ontario.

Pure Gold has reached terms, and expects to finalize an agreement for the re-commissioning, and operation of the Madsen Mine with Wabauskang and Lac Seul First Nations in 2019. Pure Gold proactively shares all documents required for regulatory review to First Nations in advance for their review and comment before they are submitted finally to regulators. All comments and guidance received is recorded, and where appropriate, incorporated into final submissions to regulatory bodies.

Pure Gold has provided closure plan reports and footprint GIS files to the First Nations to assist in their review with respect to potential land use conflicts with known cultural, archaeological, sacred sites. ENDM has followed up with First Nations and no feedback or concern was expressed.

Pure Gold has also presented all information related to the Air and Noise ECA submission to MECP in 2018. Although still under review, to date no concerns have been expressed to Pure Gold.

Furthermore, Pure Gold has also approached the local First Nations to determine the presence and or status of any active land claims. It has been confirmed that there are no active land claims which overlap the Pure Gold Land tenure.

Pure Gold will be presenting the Project Definition and proposed Closure Plan for the reopening of the Madsen Project to local communities in 2019.

Pure Gold considers that it has good relations with First Nations and is making efforts to enhance and strengthen those relations. Pure Gold has an extensive record of all consultation with First Nations since taking ownership of the mine. Pure Gold and the First Nations identified are currently seeking to develop a Benefit Agreement though it is recognized by both parties that this is an existing mine and no new impacts are expected. Pure Gold directly and through its contractors offers employment to First Nations people at the mine. Pure Gold has gained respect from First Nations for this inclusive approach to reopening the mine.

20.4.2 Community Considerations

Pure Gold has had ongoing communication with Red Lake Municipality and local business community who have an interest in the Project. Pure Gold has held a Community Meeting in October of 2016 to share information, to present its closure plan and obtain feedback for the Advanced Exploration at the Madsen Portal. Pure Gold presented project updates to the local Chamber of Commerce in 2017 and twice to the Municipality in 2018. Pure Gold has also developed a Consultation Plan that forms the basis of their ongoing plans for community, regulator and First Nations consultation. Pure Gold's plan to reopen the Madsen mine is largely seen as a benefit to the local community, which has suffered economically from the decline in mining over recent years. Pure Gold will be presenting the Project Definition and proposed Closure Plan for the reopening of the Madsen Project to the community in 2019.

20.4.3 Regulator Considerations

Pure Gold has engaged with regulators and established positive working relationships. Regulators from, CEAA, MECP, MoL, ENDM, MNRF and Red Lake Municipality have been involved with Pure Gold at the mine or in evaluation of proposed reopening activities at the mine.

20.5 Environmental Considerations

20.5.1 Effects of Mining

For reopening of the mine, as defined in the Feasibility study and Project Definition, Pure Gold intends to establish a slightly enlarged tailings facility, a mine rock storage area, overburden and soil stockpiles and an improved road haulage network, as well as a refurbishment of the existing processing plant. Cyanide detoxification (destruction) and a water treatment facility will be added. Based on the recent independent noise and air quality baseline study and modelling, noise and dust management features have been designed that are anticipated to eliminate any effect to local communities. All new features have been designed to occur within the existing catchment, so no new impacts are expected.

20.5.1.1 Site Clean Up

The Madsen Mine, currently in Temporary Suspension, has had a long legacy of mining and processing at the site. Pure Gold has expended considerable effort, at their own cost, to clear legacy waste, monitor for new impacts, and reclaim the surface as well as manage both public and mine related activities at the site. This expenditure has contributed to a much safer and better working environment and is expected to reduce closure costs at the end of mine life.

20.5.1.2 Waste Water Management

The main infrastructure for the Project and TMF are located within the Coin Creek Catchment. The TMF currently receives runoff from the mine site area, pumped waters from the underground workings, natural overland runoff from undisturbed areas of the catchment, and inputs from the municipal sewage facility. Since 1977, the community of Madsen has operated a primary treatment (settling) only 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, tailings were discharged directly to the receiving environment with no containment measures, resulting in the deposition of tailings in the current Polishing Pond, Derlak Lake, and potentially other water bodies downstream. Impounded water within the Polishing Pond is the last point of control before discharge.

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; and
- 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, Pure Gold will implement several management measures to safe guard 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 concern related to mill water recirculation and reuse; and
- 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 foot print 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) predicts that the water to be released from the TMF water treatment plant during operations will meet water quality guidelines and provincial regulations. Based on this it is expected that an updated Industrial Sewage Permit from MECP will be achievable and a DFO Environmental Assessment is not required.

20.5.1.3 Tailings Management Facility Progressive Closure

Progressive reclamation was specified as a design requirement in order to reduce the footprint of the TMF and work towards closure. This will reduce tailings remobilization and potential for arsenic and other metals mobilization along with reducing seepage from the tailings pond. By strengthening the dam wall and migrating the water away from the tailings pond wall and then closing it during operations a significant benefit in water quality will be achieved in downstream water bodies. To achieve this, the Madsen Community sewage needs to be removed and adequate rock and soil material needs to be sourced for cover and closure. No soil or mine rock was historically set aside for closure and this has necessitated the development of the 38 ha footprint with the catchment of the TMF to store benign mine rock (Lorax 2018), top soil and sub soil materials for closure covers. Lorax have shown that a substantial portion of the non-ore bearing mine rock that will be mined is benign and poses no risk for metal leaching and acid rock drainage. This material is ideal for cover material and the current design is to build a tailings cover with 1.5 m of mine rock covered by 50 cm of overburden (subsoil) covered by 30 cm of topsoil. Appropriate drainage will be designed and implemented on the closed tailings system. This plan accommodates more than half of future closure activities during the operations phase substantially reducing post closure and decommissioning costs and activities. The Polishing Pond will remain open to receive tailings until the end of mine life when it would be decommissioned and closed.

20.5.1.4 Noise Management

The Madsen Mine site has been in temporary suspension for several years and the area has been relatively quiet. With the reopening of the mine there will be more activity and a crusher will be located near the head frame. To prevent additional noise from propagating toward the Madsen Community, sound barriers will be placed along the haul road and acoustic barriers around the crusher unit. Modelling conducted by Aeroacoustics (2018 and 2019) indicates that these features will substantially reduce noise levels from operations to acceptable levels in the Madsen Community.

20.5.2 Sewage

Currently, the community of Madsen has a sewage outfall that reports to the Pure Gold Madsen Mine tailings pond after primary solids separation. Although Pure Gold 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 untreated sewage, sewage will affect the mineral processing circuit and could reduce recoveries of gold leading to a higher loading of cyanide to be used during processing. 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 is released to the environment from the tailings system. 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 closed and for ultimate closure.

Currently, some 50,000 m^3 of untreated sewage is added to the tailings facility annually and this will ultimately have to be routed away from the tailings facility so that long term closure of the tailings facility can be achieved. Engagement with the Municipality of Red Lake, MECP, ENDM and Northern Waterworks is underway and a working group has been established to find an alternative to the current system. Fifteen alternative solutions have been tabled and the working group will work towards determining a solution during the course of 2019.

20.5.3 Consultation Plan and Agreements with First Nations

Lac Seul and Wabauskang, two communities which Pure Gold is required to consult with, have come together on a Shared Territory Protocol for resource projects that are of mutual interest. This allows Pure Gold to consult openly with both parties functioning as one. Pure Gold and the communities are working towards a project agreement (Project Agreement). The Project Agreement will be 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. Pure Gold and the First Nations have agreed draft terms and are working toward finalizing the agreement.

Pure Gold retains a record of all communications and discussions and to date all interactions have been mutually positive and supportive.

20.5.4 Infrastructure and Service Requirements

All power, provincial roads and water services are already available at or near the site and Pure Gold will connect locally to these services.

Limited infrastructure is required and is primarily related to office space, dry, and crusher with acoustic screening. All significant infrastructure will be placed onto historically used mine areas therefore not increasing the mine footprint.

20.5.5 Environmental Permitting

The baseline scientific studies, predictions and forward modelling required to inform permits has been completed. These 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; and
- 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.

20.5.6 Social, Community and Economics Effects

Mines in the Red Lake area have over recent years been reducing their staffing and operations and service providers have consolidated. As a result, skilled personnel with expertise in gold mining and processing are available in the area and the availability of affordable housing has improved. It is anticipated that the potential reopening of the Madsen Mine will occur in a timely fashion such that a skilled workforce will be available locally and that conditions in the regional housing market will be favourable.

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. Pure Gold will need to work with other mines as well as the Red Lake Municipality to ensure that the Pure Gold aspiration to leave a positive and lasting mining heritage is achieved by building an economic base that can become independent of mining. Pure Gold 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.

20.5.7 Environmental Liability – Closure Plans

Pure Gold 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 2014. Pure Gold has also undertaken, at its own expense, a site cleanup that has seen significant amounts of waste removed from the site, off-site tire and metal recycling, derelict building removal, PCB storage site decommissioning and closure, security and road improvements and revegetation of some of the disturbed areas. Reopening of the mine will require an update of the closure plan and closure bond reassessment. These actions will facilitate final closure by allowing Pure Gold to fund further progressive reclamation of the Project site, as well as areas within the Madsen town site.

20.6 Summary

Pure Gold 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. Pure Gold has focussed its efforts since acquiring the project on reducing the uncertainty and risk associated with any new mining development and is actively designing operations to minimise water resource use, improve water quality and bring overall benefit to local communities and First Nations.

21 Capital Cost Estimate

LOM Project capital costs total $327 M, consisting of the following distinct phases:

- Pre-production capital costs – includes all costs to develop the property to an 800 t/d production rate. Initial capital costs total $95 M and are expended over a 15-month pre-production period on engineering, construction and commissioning activities.
- Sustaining capital costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $233 M and are expended in operating Years 1 through 13; and
- Closure Costs – includes all costs related to the closure, reclamation, and salvage value, post operations. Closure costs total $12 M (after salvage credits), and are primarily incurred in Year 14, with costs extending into Year 15.

The capital cost estimate was compiled using a combination of quotations, database costs, and factors; the overall cost estimate was benchmarked against similar operations. Table 21-1 presents the capital estimate summary for initial and sustaining capital costs in Q1 2019 dollars with no escalation.

21.1 Capital Cost Summary

Table 21-1: Capital Cost Summary

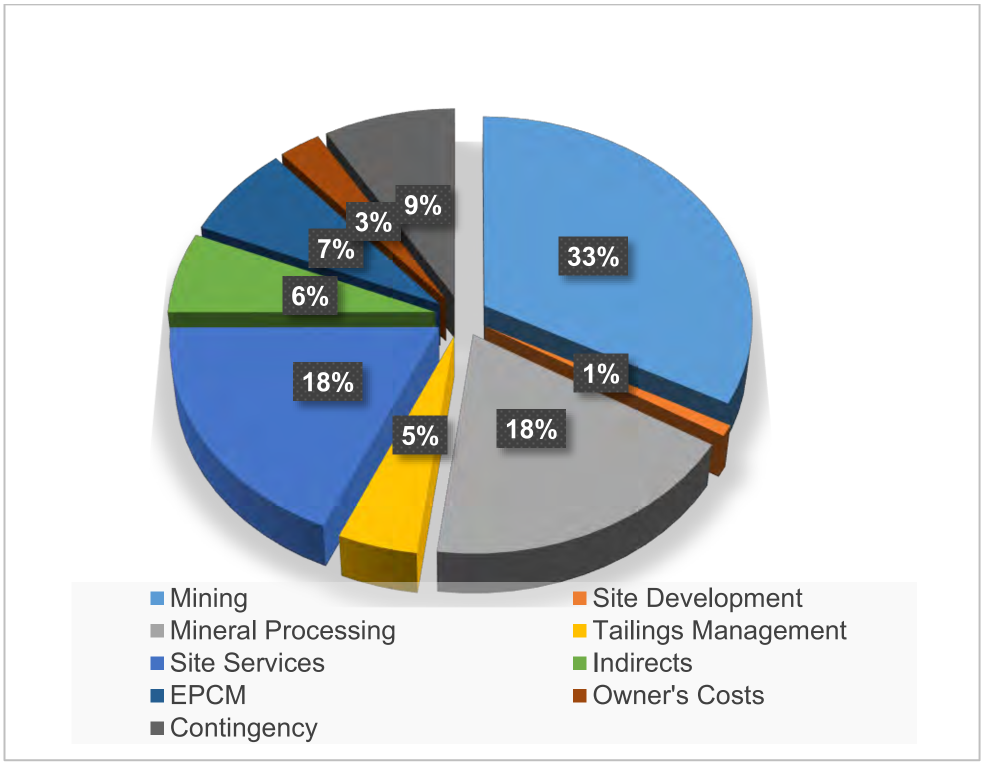
WBS AREA	WBS DESCRIPTION	Pre-Production Cost ($M)	Sustaining Cost ($M)	Project Total Cost ($M)
1000	**Mining**	**31.2**	**209.2**	**240.4**
1100	Underground Mobile Equipment	6.9	45.5	52.5
1200	Underground Infrastructure	7.1	8.2	15.3
1300	Capital Development	6.4	137.5	143.9
1400	Capitalized Production	8.2	-	8.2
1500	Hoisting	-	12.3	12.3
1600	Backfill Plant	2.6	4.1	6.7
1700	Dewatering and Shaft Rehab	-	1.6	1.6
2000	**Site Development**	**0.8**	**-**	**0.8**
2100	Bulk Earthworks (Pads)	0.1	-	0.1
2200	Site Roads	0.7	-	0.7
3000	**Mineral Processing**	**17.4**	**-**	**17.4**
3100	Crushing Area	1.7	-	1.7
3200	Coarse Ore Storage & Reclaim	1.6	-	1.6
3300	Grinding	3.2	-	3.2
3400	Pre-Leach Thickening	0.2	-	0.2
3500	Leach	1.0	-	1.0
3600	ADR & Gold Recovery	1.4	-	1.4
3700	Cyanide Destruction & Tailings Pumping	0.8	-	0.8
3800	Reagents Area	0.3	-	0.3
3900	Process Plant Utilities	6.5	-	6.5
	Capitalized Production	0.9	-	0.9
4000	**Tailings Management**	**4.3**	**7.9**	**12.1**
4100	Tailings Management Facility	0.8	0.9	1.7
4200	TMF Tailings Distribution and Reclaim System	1.8	5.9	7.6
4300	TMF Water Management	0.0	0.5	0.6
4400	Mine Rock Management Facility MRMF	1.7	0.6	2.3
5000	**On-Site Infrastructure**	**17.5**	**0.5**	**18.0**
5100	Power Supply & Distribution,	6.4	-	6.4
5200	Water Supply, Treatment & Site Distribution	6.9	-	6.9
5300	Potable Water System	0.0	-	0.0
5400	Ancillary Buildings	2.9	0.5	3.4
5500	Surface Mobile Equipment	1.0	-	1.0

WBS AREA	WBS DESCRIPTION	Pre-Production Cost ($M)	Sustaining Cost ($M)	Project Total Cost ($M)
5600	Bulk Fuel Storage & Distribution	0.2	-	0.2
5700	IT & Communications	0.2	-	0.2
6000	**Closure**	**-**	**16.8**	**16.8**
6100	TMF WRMF Closure	-	16.8	16.8
7000	**Project Indirects**	**6.1**	**-**	**6.1**
7100	Contracted Support Services	0.5	-	0.5
7200	Temporary Facilities & Utilities	0.2	-	0.2
7300	Contractor Indirects	1.5	-	1.5
7400	Logistics & Freight	1.6	-	1.6
7500	Start Up and Commissioning	2.3	-	2.3
8000	**EPCM**	**7.0**	**-**	**7.0**
8100	Engineering & Procurement	3.8	-	3.8
8200	Construction & Project Management	3.2	-	3.2
9000	**Owner's Costs**	**2.6**	**-**	**2.6**
9100	Contingency (Direct + Indirect + Owner's)	2.6	-	2.6
10000	**Contingency**	**8.1**	**2.1**	**10.2**
10100	Contingency (Direct + Indirect + Owner's)	8.1	2.1	10.2
11000	**Salvage**	**-**	**-4.2**	**-4.2**
11100	Salvage	-	-4.2	-4.2
	TOTAL CAPEX	**95.1**	**232.2**	**327.3**

Source: JDS (2019)

Figure 21-1 and Figure 21-2 present the capital cost distribution for the pre-production and sustaining phases. As typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development.

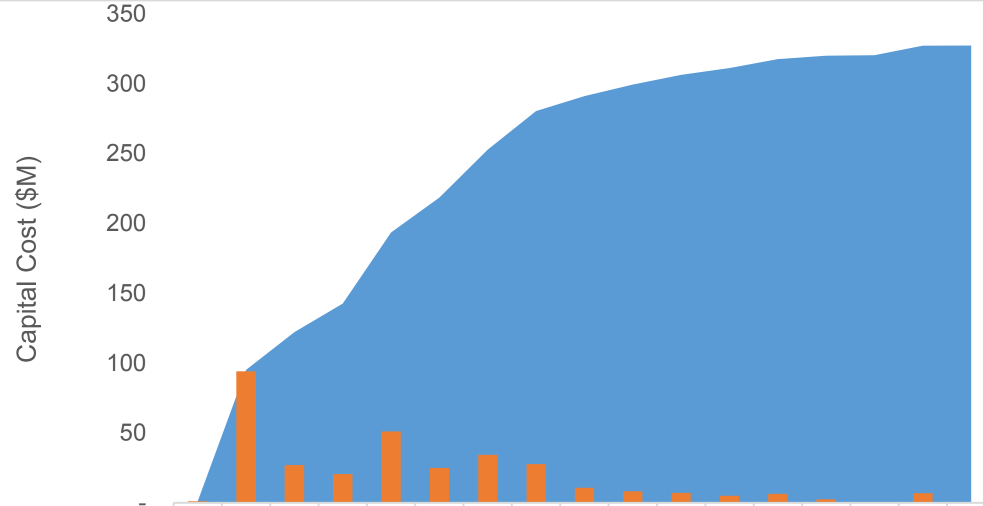
Figure 21-1: Distribution of Initial Capital Costs



Source: JDS (2019)

All capital costs for the Project have been distributed against the development schedule to support the economic cash flow model. Figure 21-2 presents an annual life of mine capital cost profile including closure years (Years 14-15).

Figure 21-2: Capital Cost Profile



	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15
Cumulative Capital Cost	1	95	122	143	193	218	253	280	291	299	306	311	318	320	320	327	327
Annual Capital Cost	1	94	27	20	51	25	34	28	11	8	7	5	6	2	0	7	0

Source: JDS (2019)

21.2 Basis of Estimate

The Project capital estimates include all costs to develop and sustain the project at a commercially operable status. The capital costs do not include the costs related to operating consumables inventory purchased before commercial production; these costs are considered within the working capital estimate described in Section 24. Sunk Costs and Owners Reserve accounts are not considered in the Feasibility Study estimates or economic cash flows.

The following key assumptions were made during development of the capital estimate:

- The capital estimate is based on the contracting strategy, execution strategy, and key dates described within the Project Execution Plan described in Section 23 of this report;
- Underground mine development activities will be performed by The Owner's team; and
- All surface construction (including earthworks) will be performed by contractors.

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 3[1] feasibility cost estimates (-15%/+15%). The overall project definition is estimated at 30%.

[1] ACEE defines a Class 3 estimate as a budget authorization estimate based on 10% to 40% project definition, semi-detailed unit costs with assembly level line items, and an accuracy of between -20%/+30% and -10%/+10%.

- Estimate Base Date: The base date of the capital estimate is Q1 2019. No escalation has been applied to the capital estimate for costs occurring in the future. Proposals and quotations supporting the Feasibility Study Estimate were received in Q4 of 2019 and Q1 of 2019.
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate, except pipe sizes which are included in Nominal Pipe Size (NPS) inches; and
- Currency: All capital costs are expressed in CAD$. Table 21-2 presents the exchange rates used for costs estimated in foreign currencies and the portions of the capital costs estimated in those currencies.

Table 21-2: Foreign Currency Exchange Rates

USD	Exchange Rates	Currency
1 USD =	1.33	CAD

Source: JDS (2019)

21.3 Mine Capital Cost Estimate

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in North America. Table 21-3 summarizes the underground mine capital cost estimate.

Table 21-3: Mine Capital Costs

Capital Costs	Pre-Production (M$)	Sustaining / Closure (M$)	Total (M$)
Underground Mobile Equipment	6.9	45.5	52.5
Underground Infrastructure	7.1	8.2	15.3
Capital Development	6.4	137.5	143.9
Capitalized Production	8.2	-	8.2
Hoisting	-	12.3	12.3
Backfill Plant	2.6	4.1	6.7
Dewatering and Shaft Rehab	-	1.6	1.6
Total Mining (excl. Contingency)	**31.2**	**209.2**	**240.4**

Source: JDS (2019)

21.3.1.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities from original equipment manufacturers (OEMs). The major mobile equipment, trucks LHDs and drills are planned to be leased from the equipment supplier. Secondary mining support equipment will be purchased outright.

21.3.1.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling. Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.3.1.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

- Lateral development fuel, equipment usage, power, and consumables requirements were developed based on the mine plan requirements. Manufacturer database equipment usage rates were applied to the required operating hours; and
- Lateral development labour requirements were determined by the required equipment fleet in operation. Supervision and support services were pro-rated to the development costs, based on the mix of underground activities occurring.

21.3.1.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, production, mine maintenance, and mine general costs) incurred prior to the introduction of feed to the processing facilities and the commencement of Project revenues. They are included as a pre-production capital cost.

The basis of these costs is described in Section 22, Operating Costs, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.3.1.5 Hoisting

Hoisting construction and install costs were developed by Nordmin Engineering, with the intention of using the existing headframe to hoist ore material to surface. Construction is occurring in year 3 to support the commissioning of the hoist at the start of Year 4.

21.3.1.6 Backfill Plant

The backfill plant equipment costs and labour hours were determined by MineFill using database costs. The building for the backfill plant was estimated by JDS using material take offs and rates developed for the other surface infrastructure items.

21.3.1.7 Dewatering and Shaft Rehabilitation

Shaft rehabilitation costs were determined by Nordmin Engineering and include the labour hours, equipment rentals, materials and mobilization costs.

Dewatering budgetary costs were provided by Technosub and adapted to suit the overarching dewatering plan by JDS.

21.4 Site Development and On-Site Infrastructure Cost Estimate

Surface construction costs include site development, mineral processing plant, tailings management facility, and on-site and off-site infrastructure. These cost estimates are primarily based on material and equipment costs from material take-offs and detailed equipment lists. Pricing for main equipment and bulk materials was primarily determined from quoted sources, with some factors applied for minor cost elements.

Table 21-4 presents a summary basis of estimate for the various commodity types within the surface construction estimates. Growth factors were included above neat material take-off quantities for all areas.

Table 21-4: Surface Construction Basis of Estimate

Commodity	Basis
Access Roads	Material take-offs for surface works and roads from preliminary 3D model. Database unit rates for earthworks
Bulk Earthworks	Model volumes from preliminary 3D grading model. Database unit rates for bulk excavation and fill. Material take-offs for surface works and roads from 3D model.
Concrete	Material take-offs measured in neat quantities and quoted rates from multiple, local subcontractors.
Structural Steel	Material take-offs estimated from similar structure and database steel rates from similar projects in Ontario.
Pre-Engineered Buildings, modular buildings and warehouses.	Buildings sized according to general arrangements, with quotations for overall building structures, inclusive of lighting, small power, electrical/control rooms, and fire detection and protection systems.
Mechanical Equipment	Over 95% of major mechanical equipment is quoted by multiple vendors. Install hours based on equipment size, database rates for similar equipment, and from vendor quotes applied to database labour rates for Ontario and Northern Canada.
Piping	Material take-offs for major pipelines equal to or greater than 4” NPS, with factoring for small bore piping based on mechanical equipment cost. Process plant piping will use existing piping in place.
Electrical and Instrumentation	Major electrical equipment list prepared and detailed major cable runs prepared in neat line material take-offs. Major equipment and cabling quoted, with some areas utilizing factored approach based on mechanical equipment. An instruments list prepared and priced using database pricing and installation hours.
Power Transmission Line and Major Sub-stations	Quantities developed based on general arrangements and site layouts. A combination of quoted and database costs applied from similar projects utilizing an experienced electrical contractor in the region.

Source: JDS (2019)

21.4.1.1 Surface Construction Sustaining Capital

The surface construction sustaining capital considers the site infrastructure installed during the mine operating years. The TMF and MRMF expansions and reclamation work, and the core processing facility built during the operating period us the same basis of estimate as provided in Table 21-4.

21.4.2 Pre-Production Process Plant Operations

The following processing related costs are included in the initial capital:

- Management, technical, operations, and maintenance labour employed during the construction phase;
- First fills of consumables and reagents, and initial consumption during process commissioning to initiate operations; and
- Energy costs for power consumed during process commissioning and start-up activities.

21.5 Closure

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF and MRMF;
- Removal of all fixed underground equipment;
- Closure of the underground mine portals and ventilation raises;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing water treatment until final closure.

The majority of closure costs are incurred immediately following completion of operations (Year 14 and 15), however, a significant reclamation of the TMF and MRMF takes place in Years 7 and 8. Table 21-5 provides a summary of closure cost categories.

Table 21-5: Closure Estimate Summary

Item	Estimated Cost ($ Millions)	
	Mid Years 7 & 8	Final Years 14 & 15
Site Roads and Facility Pads		0.4
Process and Backfill Plant		0.7
Tailings Management Facility	5.1	5.6
Mine Rock Management facility	0.5	0.4
Mine Portals and Headframe		0.7
Infrastructure		0.5
Water Treatment Plant Operation		1.4
Indirect Costs		0.7
Project Management		0.4
Owner's Costs		0.5
Total Closure	**5.6**	**11.2**

Source: JDS (2019)

21.6 Indirect Cost Estimate

Indirect costs are classified as costs not directly accountable to a specific cost object. Table 21-6 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21-6: Indirect Cost Basis of Estimate

Commodity	Basis
On Site Contract Services	Heavy Lift Crane Services based on estimated durations and historical rates for crane services
Contractor Field Indirects	Estimated by first principles, and including the following items:
	Time based cost allowance for general construction site services (temporary power, heating and hoarding, contractor support, etc.) applied against the surface construction schedule
	Construction offices and wash car facilities
	Safety training, tools and equipment
	Environmental cost
	Materials Management and Warehouse Operations
	Site Maintenance and Temporary Services
	Surveying and Quality Assurance
	Communications
	Contractor facilities and related cost
	Construction team facilities, fuel
Freight and Logistics	Historical data base rates for major equipment and infrastructure, or included in the quoted price. Remainder is factored (6%) for freight and logistics related to the materials and equipment required for the remainder of bulk materials and equipment. Factor

Commodity	Basis
	excludes mining equipment as prices quoted include delivery to site.
Vendor Representatives	Estimated by first principles, assessing the equipment supply packages and vendor services hours required for commissioning equipment.
Capital Spares	Based on material take-offs provided by vendors where supplied, with remainder factored (5%) of equipment.
Start-up and Commissioning	Included under EPCM (personnel), Owner's team costs (material and consumables), and first principles build-up of estimated contractor labour requirements
First Fills	Based on requirements determined by engineering and database pricing.

Source: JDS (2019)

21.7 EPCM Cost Estimate

21.7.1 Detailed Engineering and Procurement

Estimate based on deliverables for engineering and drafting, and time based on project management services required to oversee project development. Engineering and procurement services / costs are based on estimates provided by the consultants working on the project, JDS, Nordmin, KP and MineFill, based on the deliverables for their scopes of work in this FS. The estimate is based on the (EPCM) execution strategy. A schedule of rates was applied to a staffing plan aligned with the project schedule and execution plan described in Section 23.

21.7.2 Project and Construction Management

Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration. Costs are based on an EPCM execution strategy. A schedule of rates was applied against a staffing plan aligned with the project schedule and execution plan described in section 23.

21.8 Owner Cost Estimate

Owner costs are capitalized in the initial capital costs during the construction phase. Owner costs for the project start in Month 10 of Year -2 of the CAPEX cash flows. Any Owner costs prior to this are assumed to be within the Owner approved budget expenses and are considered sunk costs.

21.8.1 Pre-Production G&A – Labour

Costs for general and administrative labour are included for the following sectors:

- General management;
- Finance;
- Health and Safety;

- Environmental;
- Human resources;
- Procurement & logistics;
- Security; and
- Site services and facilities maintenance.

21.8.2 Pre-Production G&A – Equipment

Costs for owner site support equipment usage are included for the following sectors:

- Site Services;
- Warehouse / Material Management; and
- Administration / Management.

21.8.3 Pre-Production G&A – Expenses and Services

Costs for general and administrative expenses and fees are included for the following sectors:

- Health, safety and medical supplies;
- New hire orientations;
- Staff safety equipment (surface staff PPE);
- Surface support equipment operation fuel and maintenance;
- Contract load and haul of ore and mine rock from Madsen portal;
- Surface facilities electrical power consumption;
- Site facilities maintenance supplies and consumables;
- Environmental services, fees, and outside laboratory costs;
- Human resources (training, recruitment);
- Construction insurance;
- Mineral tenure and surety bond annual fees;
- Legal and regulatory, including property tax;
- External consulting;
- IT and communications;
- Site office costs;
- Office equipment lease and services;
- Waste disposal; and

- Senior staff travel allowance.

21.9 Salvage Estimate

Various on-site equipment was considered to have residual value at the end of the project life. The net cash value of each item was estimated from 5% to 10% of the initial purchase price for a total of approximately $4.2 million.

21.10 Contingency

Contingency has been applied to the estimate determined by the estimate class of the sub-categories. An overall contingency of 10% was applied to the pre-production CAPEX, which resulted in approximately 9% of direct, indirect, and Owner costs. Where additional contingency was deemed necessary, based on the level of confidence for that sub-category, a growth allowance was added to those specific items. The overall contingency is applied after the growth allowance has been added.

21.11 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any Project sunk costs (studies, exploration programs, etc.);
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

22.1 Operating Cost Summary

The operating cost estimate in this study includes the costs to mine and process the mineralized material to produce doré, along with general and administrative expenses (G&A). These items total the Project operating costs and are summarized in Table 22-1. The target accuracy of the operating cost is -10/+15%.

The operating cost estimate is broken into three major sections:

- Underground mining;
- Processing; and
- General and Administrative (G&A).

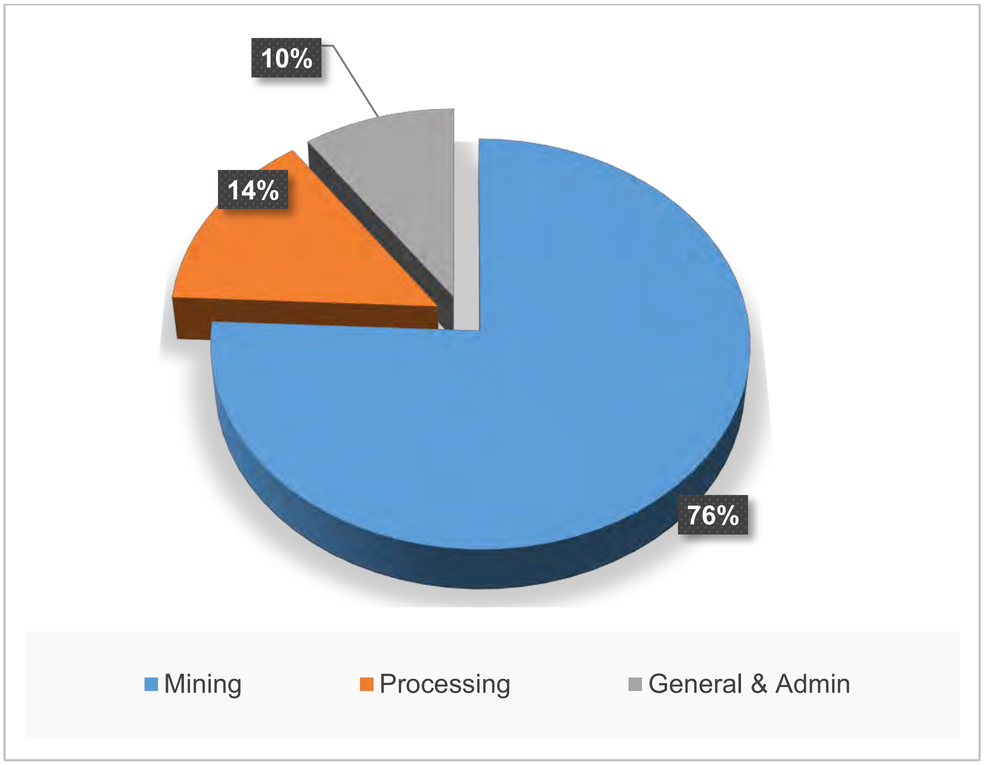
The total operating unit cost is estimated to be $223/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 22-1. The unit rates in this table do not include material mined during pre-production. Figure 22-1 illustrates the operating cost distribution. No allowance for inflation has been applied.

Table 22-1: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed	LOM (M$)
Mining	48.4	168.80	592.8
Processing	9.3	32.30	113.3
G&A	6.3	21.80	76.7
Total	**63.9**	**222.90**	**782.9**

Source: JDS (2019)

Figure 22-1: Operating Cost Distribution



Source: JDS (2019)

The main operating cost component assumptions are shown in Table 22-2.

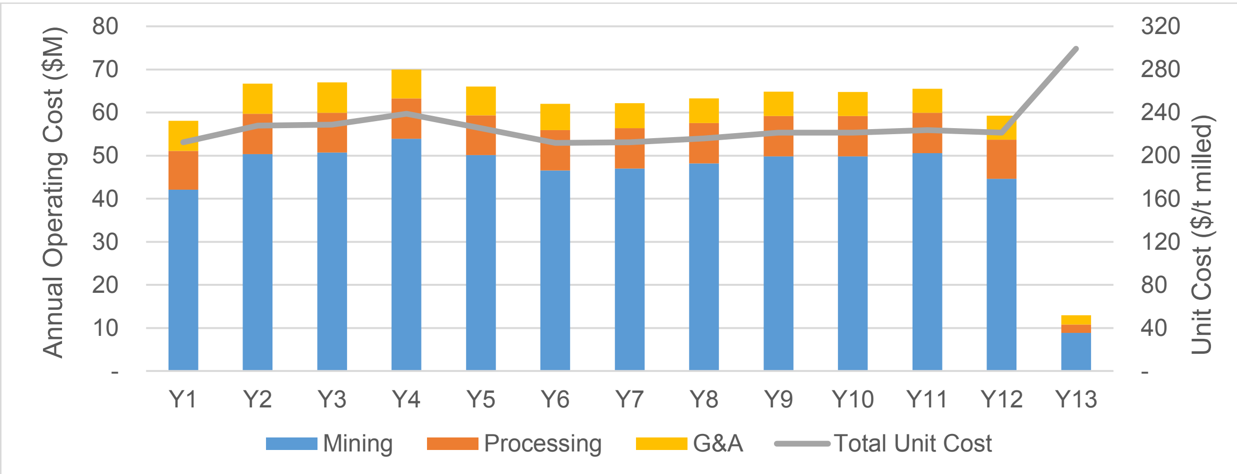
Table 22-2: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.100
Overall Power Consumption (all facilities)	kWh/t processed	9.87
Diesel Cost (delivered)	$/litre	1.072
LOM Average Operations Workforce	employees	400

Source: JDS (2019)

All operating costs for the project have been estimated against the development schedule to support the economic cash flow model. Figure 22-2 presents the annual LOM operating cost profile.

Figure 22-2: Life of Mine Operating Cost Profile



Source: JDS (2019)

22.2 Basis of Estimate

Operational labour rates have been estimated by applying legal and discretionary burdens against base labour rates. Wage scales were defined and applied to the various operational positions based on skill level and expected salary.

The Owner team averages 360 positions during operations. Levels will fluctuate depending on the amount of yearly underground development.

Table 22-3 lists the total labour positions by area for the construction and operations phases.

Table 22-3: Total Labour Table Construction and Operations

Position	Max Construction	Max Operations	LOM Average
Mining	223	392	334
Processing	39	39	37
G&A	23	25	23
Total	**285**	**456**	**360**

Source: JDS (2019)

22.3 Mine Operating Cost Estimate

22.3.1 Underground Mining Operating Costs

Mine operating unit costs are summarized below in Table 22-4 and Figure 22-3 and include:

- Longhole Stoping – costs related to the drilling blasting, mucking and hauling of longhole stopes including sublevel and access development;

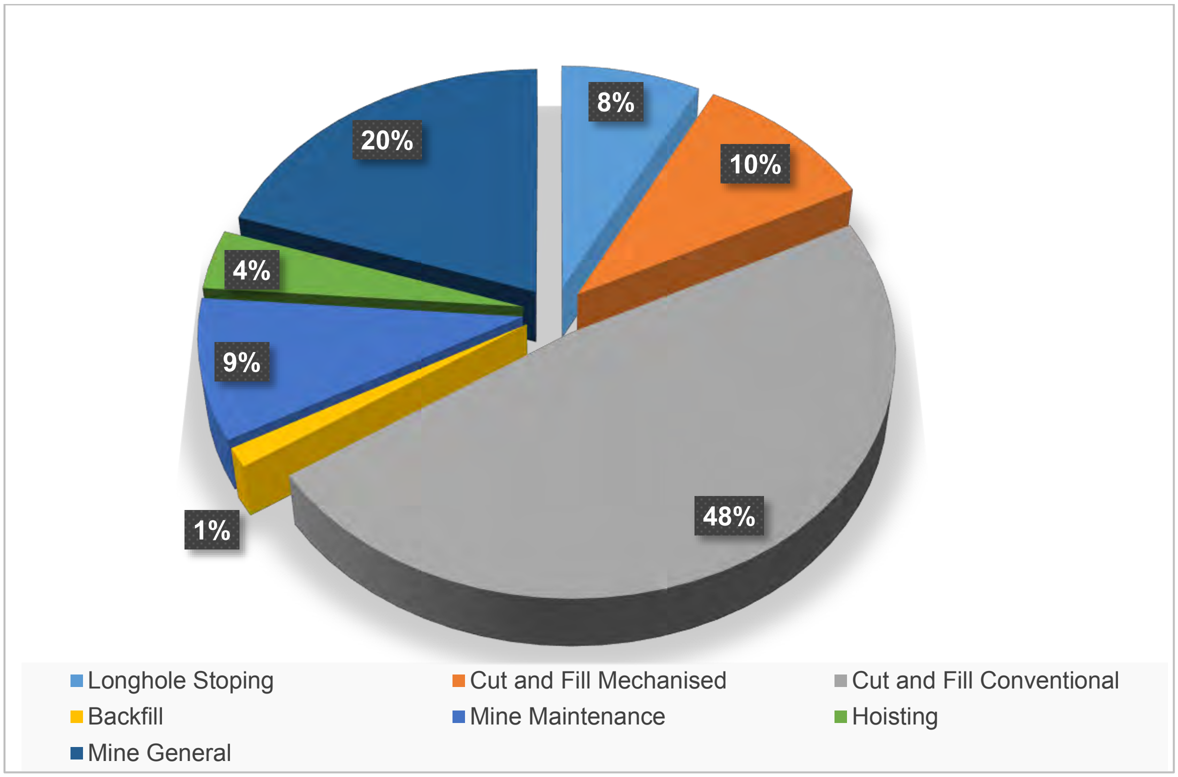
- Cut and Fill Mechanized – costs related to the drilling blasting, mucking and hauling of MCF stopes including attack ramp access;
- Cut and Fill Conventional – costs related to the drilling blasting, mucking and hauling of CCF stopes including raise and lateral access development;
- Backfill – costs related to rockfill and hydraulic backfill operations, backfill plant;
- Mine Maintenance – maintenance labour costs that support all other sectors;
- Hoisting – cost related to operation of the underground hoisting system; and
- Mine General – costs related to mine support activities, such as technical services, shared infrastructure, support equipment, and definition drilling.

Table 22-4: Underground Mine Operating Costs

Mining Category	Unit Cost ($/t processed)	LOM Cost (M$)
Longhole Stoping	12.7	44.6
Cut and Fill Mechanized	16.9	59.3
Cut and Fill Conventional	81.1	285.0
Backfill	2.2	7.6
Mine Maintenance	15.8	55.4
Hoisting	6.6	23.3
Mine General	33.5	117.6
Total	**168.8**	**592.8**

Source: JDS (2019)

Figure 22-3: Operations Costs by Category



Source: JDS (2019)

22.3.2 Underground Mining by Component

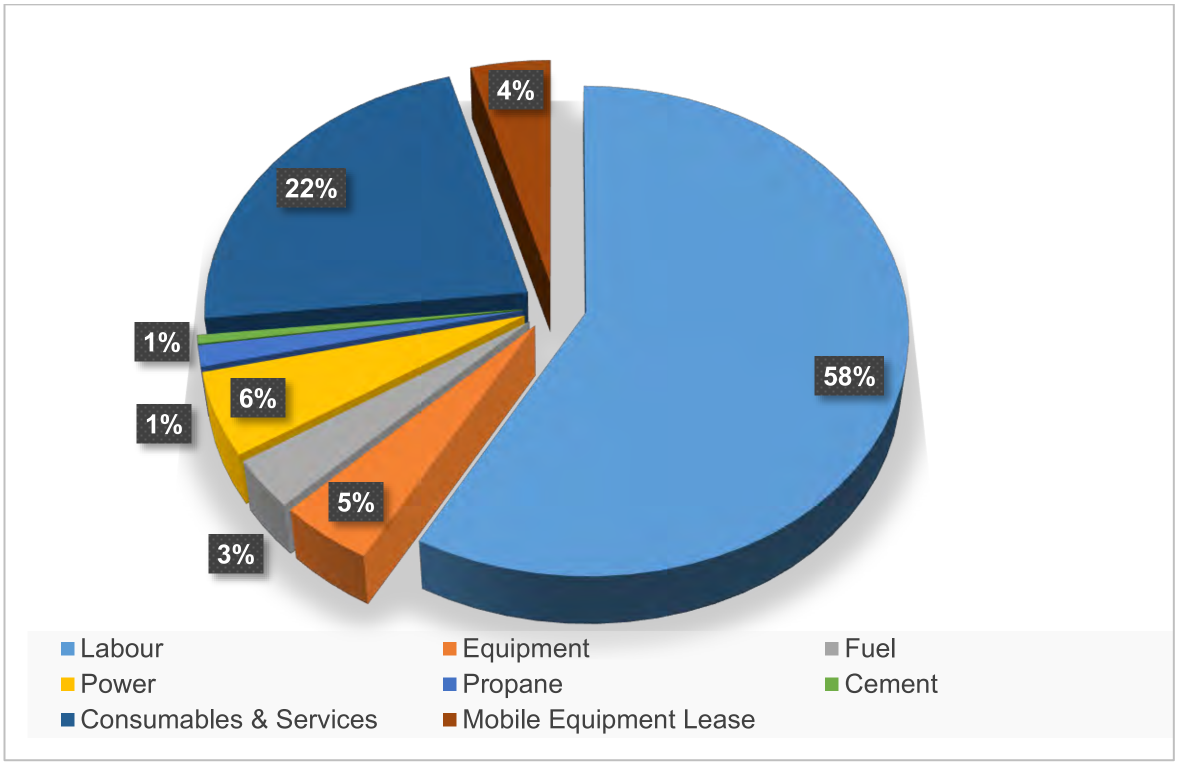
Table 22-5 and Figure 22-4 illustrate the mining operating cost broken down by major component.

Table 22-5: Mine Operating Cost by Component

Mining Component	Unit Cost ($/t processed)	LOM Cost (M$)
Labour	98.0	344.3
Equipment	7.4	26.1
Fuel	5.6	19.8
Power	9.3	32.8
Propane	2.4	8.3
Cement	1.0	3.4
Consumables & Services	37.7	132.6
Mobile Equipment Lease	7.3	25.6
Total	**168.8**	**592.8**

Source: JDS (2019)

Figure 22-4: Underground Mining Operating Costs, by Cost Component



Source: JDS (2019)

22.3.2.1 Labour

Underground mining staffing levels related to production activities are built up based on the productivities (work-hours) required for mining activities occurring within a given time period. As such, mining workforce fluctuates throughout the mine life.

Mine labour (including supervision and support) related to development drifting is distributed between capital development (sustaining capital costs) and operating development (operating costs), based on the activities being performed within a given time period. As such, only a portion of the mine staffing is allocated within the mining operating costs.

22.3.2.2 Equipment Operations

Underground mining equipment usage costs are based on the equipment operating hours required to meet the life of mine plan. Equipment usage costs include unit costs ($/hr) for the following elements:

- Maintenance parts;
- Tires;
- Lubricants; and
- Boxes, buckets, and ground engaging tools.

Unit costs for the elements above have been obtained from equipment manufacturers databases. Equipment replacements and major (mid-life) overhauls are included in the sustaining capital costs.

22.3.2.3 Fuel Consumption

Underground mining fuel consumption has been built up based on the required equipment operating hours based on the mine plan for development or production-based equipment, and annual allowances for support or fixed infrastructure equipment. The unit fuel price used in the estimate is $1.072/litre, inclusive of delivery to site.

22.3.2.4 Power

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level. Underground mining power includes the power consumption of the hoist. Electricity unit cost is based on a budgetary rate of $0.100/kWh.

22.3.2.5 Propane

Propane consumption has been based on the mine air heating requirements and the ventilation flow. Propane unit cost is based on a budgetary rate of $0.559/L.

22.3.2.6 Cement

Consumption has been based on the estimated structural hydraulic fill requirements of the mine. Structural hydraulic fill is estimated to require 5% cement. Cement unit cost is based on a budgetary rate of $276.06/t.

22.3.2.7 Consumables and Services

Mining consumable usage rates are built up based on the mine plan quantities for development and production activities. Unit costs are typically based on budgetary quotations. Minor item costs are based on catalog or database values. Three percent (3%) of the base quoted or database pricing for consumables has been added within the commodity pricing to account for delivery (freight) to site.

22.3.2.8 Mobile Equipment Lease

Select Major equipment up to the end of the 2nd year of production were leased. Leasing terms include a 15% down payment, with 60 monthly payments at a rate of 5.95%.

22.4 Processing Operating Cost Estimate

Processing operating costs were estimated to include all gold and silver recovery activities to produce unrefined gold doré on-site. The crushing and process plants are designed for a throughput of 800 t/d. Labour rates and benefit loadings are based on information supplied by JDS and Pure Gold. All reagent cost estimates are detailed in Section 21.2.3.4. The process operating costs are summarized in Table 22-6.

Table 22-6: Process Operating Costs

Processing Category	Unit Cost ($/t processed)	LOM Cost (M$)
Labour	15.99	56.2
Power	5.71	20.1
Maintenance and Consumables	10.56	37.1
LOM Processing Costs	**32.27**	**113.3**

Source: JDS (2019)

22.4.1 Processing Labour Requirements

Milling operations and maintenance staffing levels have been built up based on experience at similar operations. Labour costs are based on fully burdened staffing wage.

22.4.2 Processing Power

Electrical power consumption has been based on the equipment connected loads, operating time and the anticipated operating load level.

The total estimated annual process plant energy consumption is estimated to be approximately 16.7 MWh/year. At an estimated power cost of $0.10/kWh, the LOM power cost is $20.1 M ($5.71/t processed).

22.4.3 Processing OPEX Wear Parts & Consumables

Liners for the grinding mills and primary crusher have been estimated based on Vendor quotes, ore hardness, and on experience at similar operations. Grinding media consumptions for the grinding mills have been estimated on a kilogram/tonne basis, mill power draw and abrasion index. Budgetary quotations for liners and grinding media were received from equipment Vendors and local suppliers.

Annual maintenance parts costs have been factored at a rate of four percent of the direct capital costs of the equipment within the crushing and process areas.

22.4.4 Reagent Costs

Reagent costs were based on recent quotes including freight to site. Table 22-7 summarizes the reagent and chemical requirements. The annual costs are based on 800 t/d, 365 days per year.

Table 22-7: Reagent Requirements and Costs

Reagents	Reagent Consumption (t/a)	Unit Reagent Cost ($/t)	Annual Cost ($)
Cyanide	110	4,480	492,800
Lime (CaOH)	204	500	102,000
Carbon	11	5,760	63,360
Caustic Soda	17	890	15,130
HCL	38	620	23,560
$CuSO4.4H_2O$	36	3,264	117,504
SMBS	302	768	231,936
Lead Nitrate ($PbNO_3$)	73	3,200	234,667
Flocculant	4	4,750	19,000
Antiscalant	15	4,500	67,500
Total			**1,367,457**

Source: JDS (2019)

22.5 General and Administration Operating Cost Estimate

Table 22-20 presents a summary of General and Administration operating costs.

Table 22-8: General and Administration (G&A) Operating Cost Summary

Area	Average $ M/year	Life of Mine $ M	$/t processed
G&A Labour	**2.1**	**26.2**	**7.5**
General Management	0.2	2.9	0.8
Finance	0.3	3.3	0.9
Environmental	0.2	3.0	0.9
Health and Safety	0.1	1.2	0.3
Security	0.3	4.1	1.2
Human Resources	0.1	1.2	0.4
Purchasing and Logistics	0.3	3.1	0.9
Surface and Infrastructure	0.6	7.2	2.1
G&A On-Site Items	**3.8**	**46.7**	**13.3**
Health & Safety, Medical, & First Aid	0.1	1.3	0.4
Surface Support Equipment	0.2	1.9	0.5
Contract Loader and Trucking	0.7	9.0	2.6
Surface Infrastructure Power	0.3	3.7	1.1
Facilities & WTP Maintenance/Consumables	0.8	10.0	2.8
Environmental	0.4	4.9	1.4
Human Resources	0.1	0.9	0.2

Area	Average $ M/year	Life of Mine $ M	$/t processed
Insurance & Legal	1.0	12.3	3.5
External Consulting	0.1	1.2	0.3
IT & Communications	0.1	0.9	0.3
Office & Miscellaneous Costs	0.0	0.5	0.2
Employee Travel	**0.3**	**3.8**	**1.1**
Total G&A	**6.3**	**76.7**	**21.8**

Source: JDS (2019)

22.5.1 General and Administration (G&A) Labour

G&A staffing levels have been prepared in, consultation with Pure Gold management, and experience at other, similar operations. Labour costs are based on fully burdened staffing wages.

22.5.2 G&A Services and Expenses

G&A services and expenses have been estimated in consultation with Pure Gold management, and with consideration for other, similar operations. Major items are based on current Pure Gold budgeted and anticipated operating expenses. A budgetary insurance proposal for the project was used for the insurance costs. Minor items are factored, based on estimate parameters such as numbers of staff, or are general allowances based on experience with other projects and input from Pure Gold.

22.6 Contingency

No operating cost contingency provision has been included in the estimate.

23 Project Execution

23.1 Introduction

The purpose of the Madsen Project Execution Plan (PEP) is to describe the project development strategies that were considered for the FS capital cost estimate and project schedule, and to provide the future framework for organizing the engineering, procurement, and construction. The PEP will also serve as a guide in:

- Promoting safety in design, construction, and operations to succeed;
- Promoting sustainable social investment and community relations within the area during the planning, construction and operations phases;
- Ensuring environmental aspects and impacts of the project are considered completely and comprehensively during the construction, operational and post-operational phases of the project; and
- Negotiating contracts with suppliers, contractors, and engineers with proven track records developing similar projects.

Although the PEP provides guidance for executing the Project, the planning stage will evaluate alternate execution strategies and other opportunities that add further value overall. This may include items such as variations to portions of the execution strategy (i.e. EPCM, Engineering, Procurement and Construction (EPC), Engineering, Procure and Supply (EPS), etc.) or, inclusion of Owner resources for smaller scopes of work.

23.2 Project Development Schedule

A level 3 procurement schedule as defined in the AACE International Practice standards was developed for the Project, using the capital cost estimate as the basis for on-site work hours to drive activity durations. The Project development period is 15 months long to the start of first commercial production. Figure 23-1 presents a summarized version (level 1, AAEC) schedule for the development of the Madsen Project.

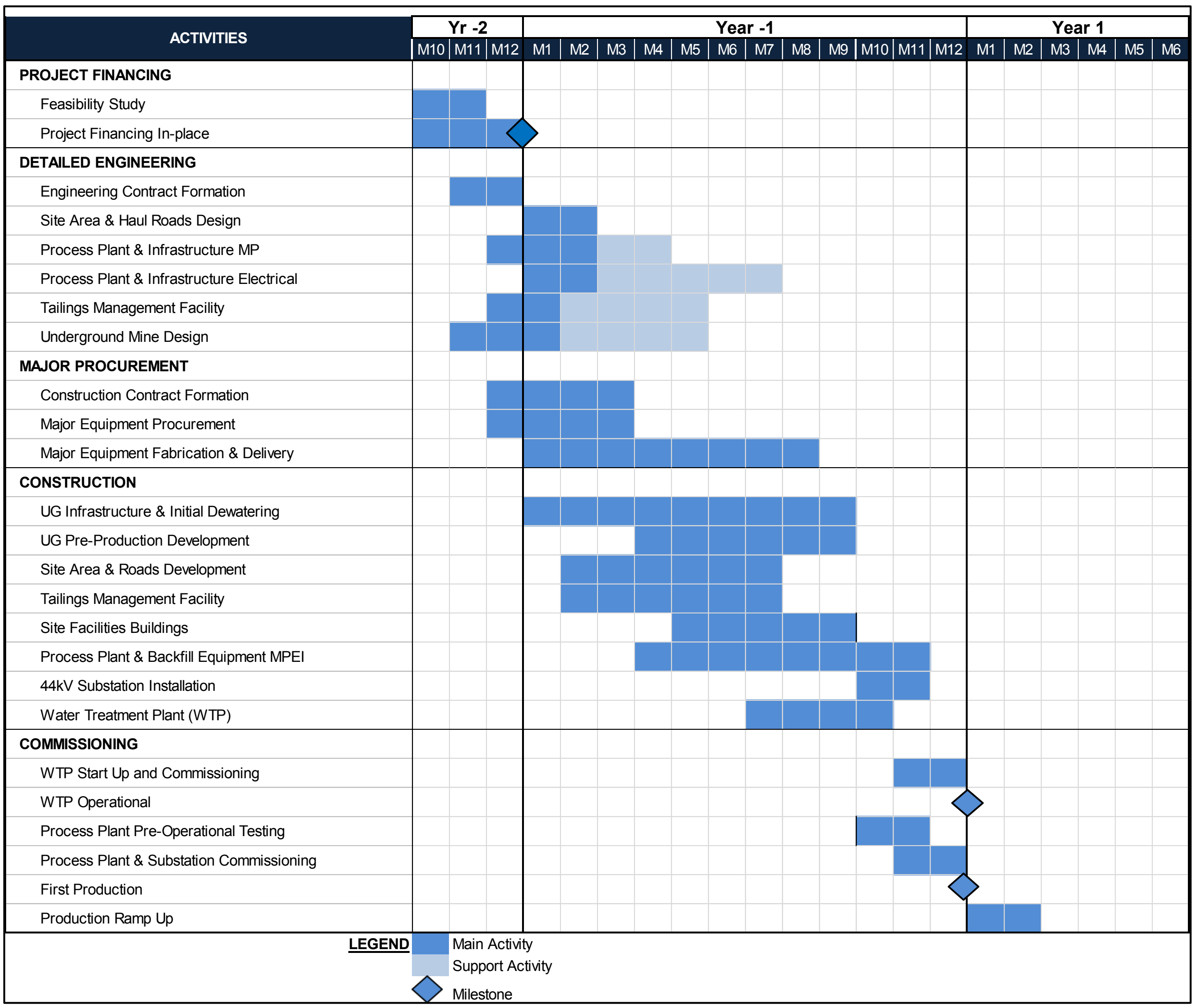
The critical path of the schedule runs through the following activities:

- EPCM contract formation;
- Mine and mechanical detailed engineering;
- Mine dewatering and pre-production development;
- Crusher and ball mill procurement and installation;
- Electrical engineering and procurement for substation to connect to electrical grid; and
- Process commissioning and production ramp-up.

Other near-critical activities include:

- Gravity concentrator circuit equipment procurement and installation;
- Gold room equipment procurement and installation; and
- Process plant controls wiring and installation completion.

Figure 23-1: Summarized Project Schedule



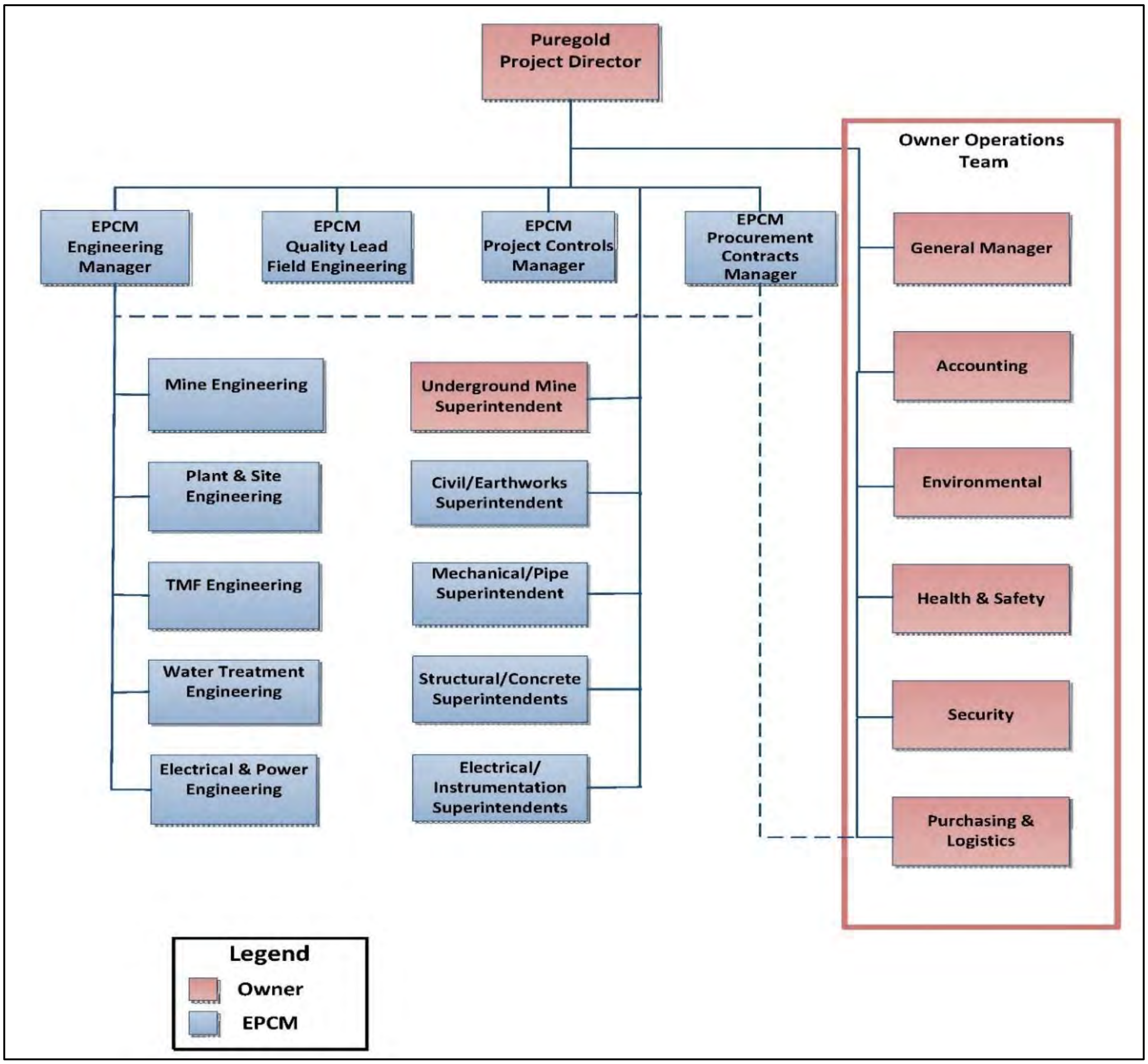
Source: JDS (2019)

23.3 Project Management

23.3.1 Organization and Responsibilities

The Project Management Team (PM Team) will be an integrated team including the Owner's personnel, the EPCM Contractor, and various engineering contractors. The PM Team will oversee and direct all engineering, procurement, and construction activities for the Project. Figure 23-2 presents the preliminary project organization chart.

Figure 23-2: Project Organization Chart



Source: JDS (2019)

23.3.1.1 Senior Project Management

Overall delivery of the Project to the defined metrics is the responsibility of the Project Director. The Project Director provides high level direction to the PM Team, with support from the EPCM Contractor and the Owner's Pre-Operational team to manage Project activities.

The Project Director will be accountable for the execution of Project activities, including detailed engineering, procurement, logistics, construction, commissioning, and Project controls. The Project Director is also responsible to ensure that all safety, environmental and CSR project standards, requirements, codes and regulations incorporated into the project charter, directives and management plans are adhered to.

23.3.1.2 Owner Operations Team

A portion of the Owner Operations team will be mobilized during the development phase of the Project because these functions will also be required during operations (i.e. after construction is completed). The Owner operations team will be responsible for the following functional areas:

- Health and Safety;
- Environmental;
- Project Support Services, inclusive of:
  - Supply Chain Management;
  - Information Technologies (IT);
  - Accounting;
  - Legal;
  - Security;
- Human Resources;
- Operations, inclusive of:
  - Mining operations;
  - Process plant, including maintenance;
  - General and Administrative; and
  - Site Services (reporting to the EPCM Construction Manager during construction).

23.3.1.3 Engineering Team

The Engineering Manager will oversee, coordinate, and integrate engineering activities. The Engineering Team will consist of various engineering sub-contractors, who will develop the detailed designs and specifications for the Project, and then transition to the field to provide quality assurance (QA), field engineering, and commissioning support.

23.3.1.4 Procurement Team

The Procurement & Contracts Manager will oversee and manage contracts administration and procurement activities undertaken by engineering contractors (formation and administration of engineering and construction contracts will be overseen and managed by EPCM Contracts personnel). The procurement team will use the prepared engineering design packages to obtain competitive tenders, and secure vendors and construction contractors to provide the appropriate goods and services.

23.3.1.5 Construction Management Team

The Construction Superintendents will be responsible for construction safety, progress, and quality. The Construction Management Team will coordinate and manage all site activities to ensure construction progresses on schedule and within budget.

23.3.1.6 Commissioning Team

The Engineering Manager will oversee the commissioning team and be responsible for the timely handover of process and infrastructure systems to the owner once construction activities have been substantially completed. The commissioning team will be supported by disciplined engineering resources to complete pre-commissioning activities and to obtain technical acceptance and transfer care, custody, and control of completed systems to the owner.

23.3.1.7 Project Controls Team

The Project Controls Manager will oversee the Project controls team, and be responsible for the development, implementation, and administration of the processes and tools for Project estimating, cost control, planning, scheduling, change management, progressing, and forecasting.

23.3.1.8 Project Procedures

During the Project setup phase (immediately upon project approval), a Project Procedures Manual will be developed, which will outline standard procedures for project execution. This document will focus on the interfacing between the Owner, EPCM team and engineering contractors, and address delegation of authority, change management, procurement workflows, QA/QC, and reporting standards.

23.4 Project Support Services

The project support execution strategy for the Project will be to utilize the owner’s team to provide the support services for:

- Security;
- Legal;
- Operations’ supply chain management;
- IT services; and
- Accounting.

Coordination of EPCM contractors and the Pure Gold project support services team interfaces will be the responsibility of the Project Director.

23.5 Engineering

23.5.1 Engineering Execution Strategy

The general engineering execution strategy for the Project will be to utilize multiple engineering firms with specialized knowledge of their assigned scope. Coordination of engineering interfaces and overall management of engineering schedule and deliverables will be the responsibility of the EPCM Technical Manager. The following major engineering contract packages have been identified for the Project:

- Detailed engineering and procurement of mine and process facilities and select on-site infrastructure, including field engineering support;
- Detailed engineering of the TMF, MRMF and associated tailings and water management infrastructure;
- Hydrological characterization, water balance, and water management systems including dewatering, water conveyance and treatment systems;
- Electrical power study and detailed design for grid connection and substation equipment (potential EPC contract); and
- Hydraulic fill plant design.

23.5.2 Engineering Management

23.5.2.1 Project Technical Organization

The preliminary project technical organization is as shown in Figure 23-2, under the EPCM Engineering Manager.

23.5.2.2 Baseline Engineering Data

Engineering data from the FS, including (but not limited to) design criteria, flow sheets, material take-offs, and drawings are considered the engineering baseline data, and form the basis for the capital cost estimate and schedule. Deviations from these baseline engineering inputs, beyond clarifying and finalizing scope, and detailing of designs will be subject to the project change management processes.

23.5.2.3 Design Criteria Approval

The Project critical path includes timely completion of engineering activities. To prevent delays or late changes in engineering deliverables and to keep efforts focused, an approval procedure will be developed and adhered to.

23.5.2.4 Engineering Progress and Performance Monitoring

Each engineering contractor will provide a deliverables list as part of their services proposal. Deliverables (and their associated budgets) will be grouped into logical Engineering Work Packages (EWPs), which will be used as the metric for tracking engineering progress for the Project.

23.6 Procurement and Contracting

23.6.1 Procurement Execution Strategy

The general procurement execution strategy for the Project will include utilizing known suppliers, with a preference for local or regional suppliers and construction contractors. The commercial manager (with input from the engineering manager) will have overall responsibility for the majority of pre-purchased procurement and contract formation activities. Contract administration will be the responsibility of the commercial manager.

23.6.2 Construction Contracting Strategy

Table 23-1 presents a listing of the major contract packages identified for the Project. For the purpose of the Feasibility Study, all mechanical, piping, electrical, and instrumentation (MPEI) works have been identified as performed by a single entity within the Project estimate and schedule. During contractor pre-qualifications, if multi-discipline contractors cannot be sourced, then a horizontal contracting strategy will be employed (separate contractors for each trade, i.e. mechanical, piping, electrical, and instrumentation).

Construction Work Packages (CWPs) will be developed to manage the execution of the scopes of work on site. A CWP is an executable construction deliverable that is defined by logical subdivision of the overall construction scope into manageable packages based primarily on geographical boundaries that do not overlap. It includes a budget and schedule that can be compared against actual performance and used to track construction progress. The CWPs will be used as a scoping document for Requests for Proposals, contracts, and / or portions of contract.

The strategy for the underground mining activities on the Project is to use the Owner's mining team to install the underground infrastructure and complete pre-production development.

Table 23-1: Major Construction Contracts (Capital Phase)

Contract Number	Contract
CC001	Earthworks, Roads and TMF Construction
CB001	Concrete Supply and Install
CF001	Pre-Eng Building Supply and Install
CF002	Modular Building Supply and Install
CM001	Crushers, Conveyors and Ore Bin SMP
CM002	Process Plant & Backfill SMP
CM003	Water Treatment Plant SMP
CL001	HV Substation and Connection to Grid
CL002	Process & Backfill Plants E&I
CL003	Site HV/MV Distribution
CL004	Water Treatment Plant E&I

Source: JDS (2019)

23.6.3 Procurement Schedule and Critical Activities

The major procurement phase of the Project will occur in Year -1. Procurement activities will be prioritized to schedule critical items, both due to fabrication/delivery time of the equipment (such as the mine cooling and dewatering infrastructure, paste pumps and grinding mill package), and due to the necessity to obtain certified vendor data to complete structural and foundation designs. Table 23-2 presents the critical and long lead packages for the Project, with estimated manufacturing lead times shown in weeks, After Receipt of Order (ARO).

Table 23-2: Long Lead Purchased Equipment

Procurement Package	Description	Estimated Lead Time (ARO, weeks)
PM001	Crusher Plant	22 - 24
PM002	Ball Mill	28 - 30
PM003	Gravity Concentrator Intensive Leach System	20 – 22
PM004	Gold Room Package	24 – 28
PM005	Water Reclaim System	25

Source: JDS (2019)

Tendering and award of the following packages are considered time critical (on or near critical path):

- EPCM contractor selection;
- Procurement of ball mill;
- Procurement of dewatering pumps; and
- Detailed engineering contracts, particularly for UG mining, dewatering, processing and water treatment plant packages.

23.6.4 Selection of Suppliers and Contractors

A competitive bidding process will be applied to achieve the best commercial and technical results from the procurement effort. During the project setup phase, preferred vendors will be identified (where deemed to add value to the project) and sole source strategies implemented into the procurement plan.

The level of vendor quality surveillance / inspection (VQS) required for each package will be established during bid evaluations and will be determined by evaluating a supplier's ability to achieve suitable quality according to specifications and project QA requirements.

23.6.5 Alternatives and / or Variations to the Procurement and Contracting Strategy

During the project setup phase, the Project Team will evaluate, assess and implement alternates and / or variations to the Procurement and Contracting Strategy that will add value or de-risk the project. Variations may include changes to the scope of the contracts, the number of contracts, or the number of contractors on site depending on the results of the contractor pre-qualifications or the detailed engineering. Alternatives may include an EPS or EPC contract for portions of the project scope.

23.7 Logistics and Materials Management

23.7.1 Logistics Execution Strategy

The logistics execution strategy includes the following objectives:

- Ensure expediting activities are sufficient to achieve schedule requirements;
- Manage global freight movements to optimize freight movement cost; and
- Identify and optimize various aspects such as logistics, customs clearance and local content.

23.7.2 Shipping Routes

Most international freight will be shipped through the Montreal or Toronto, on the Atlantic side, or Vancouver on the Pacific side. Most domestic goods are expected to originate within the regional area, or from suppliers from Winnipeg, to the southwest and Thunder Bay, to the southeast. All road transport is on paved road.

23.7.3 Pre-Assembled Equipment

Pre-assembly strategies reduce overall site work hours and the associated indirect costs, but require more careful engineering and logistics planning. The following goods have been identified for pre-assembly within the FS estimate:

- Electrical houses;
- Crusher plant;
- Fuel, water and process tanks (up to 5 m diameter);
- Conveyors (shipped in pre-fabricated lengths; and
- Transfer towers, braced frames, and stair towers.

23.7.4 Site Materials Strategy

The general strategy for site materials control is as follows:

- Control and supervise materials movement at site through materials / inventory control from receiving, preservation, inventory, and free-issue to contractor to meet the Project requirements for all equipment and materials procured by the construction team or EM (i.e. process equipment);
- Include in contractor's scope requirements for receipt, storage, and retrieval of procured materials required for its work;
- Utilize a common labour pool for warehouse and laydown staff (equipment operators and labourers) to support the management and movement of freight, except for items requiring special handling or rigging (such as structural steel); and
- Utilize a temporary warehouse for the receipt and storage of all equipment requiring climate controlled indoor storage. Equipment and material that do not require climate-controlled storage will be stored in laydown areas within the construction site. Use of sea containers and/or temporary shelters will be required to store goods that need to be protected during construction.

23.8 Construction

23.8.1 Construction Execution Plan Overview

The main objectives of the construction execution strategy include:

- Execute all activities with a goal of zero harm to people, assets, the environment, or reputation;
- Strive to eliminate process, operational and maintenance safety hazards;
- Meet or exceed environmental regulatory and permit requirements to minimize impact;
- Cultivate an atmosphere of positive social impact in the surrounding communities;
- Deliver a high-quality facility that meets or exceeds the defined project goals;
- Establish and maintain a high level of motivation by providing a positive working environment for all personnel;
- Identify and remove barriers that affect project progress; and
- Recognize, identify and communicate outstanding achievements during construction and commissioning of the Project.

23.8.2 Site Management

During the construction Phase, the Project Director (or his designate) will be responsible for the development and construction areas. The designated EPCM Site Manager and Pure Gold General Manager will closely coordinate site activities, and responsibilities will be separated for areas such as the underground mine.

23.8.3 Construction Management

The EPCM Construction Team will be responsible for construction contractors' oversight.

23.8.4 Safety Management

A comprehensive Safety Management Plan (SMP) will be developed prior to site mobilization. The SMP will address overall safety policies, procedures, and standards for the Project, including standard operating practices and emergency response plans for both above ground and underground.

23.8.5 Quality Management

Construction quality will be managed through the implementation of a Site Quality Management Plan (SQMP), which will detail the site quality management systems to be used for all construction activities. The SQMP encompasses all activities of the Project, including design, procurement and construction. Site QA is the responsibility of the EPCM Field Engineering team, as is verification that QC is being performed by the contractor, subcontractor, laboratory and third party inspection services.

23.8.6 Construction Quantities

Table 23-3 presents the estimated major commodity quantities for the Project. Quantities are based on the Feasibility Study engineering take-offs and capital estimate.

Table 23-3: Project Construction Work Hours by Discipline

Discipline	Commodity Quantity	
	UOM	Approximate Quantity
Concrete	hrs	3,440
Civil	hrs	10,360
Architecture	hrs	3,500
Electrical & Instrumentation	hrs	32,960
Mechanical	hrs	30,840
Platework	hrs	580
Piping	hrs	4,400
Structural Steel	hrs	4,800

Source: JDS (2019)

23.8.7 Construction Milestones

Table 23-4 presents the major construction milestones for the Project.

Table 23-4: Major Construction Milestones

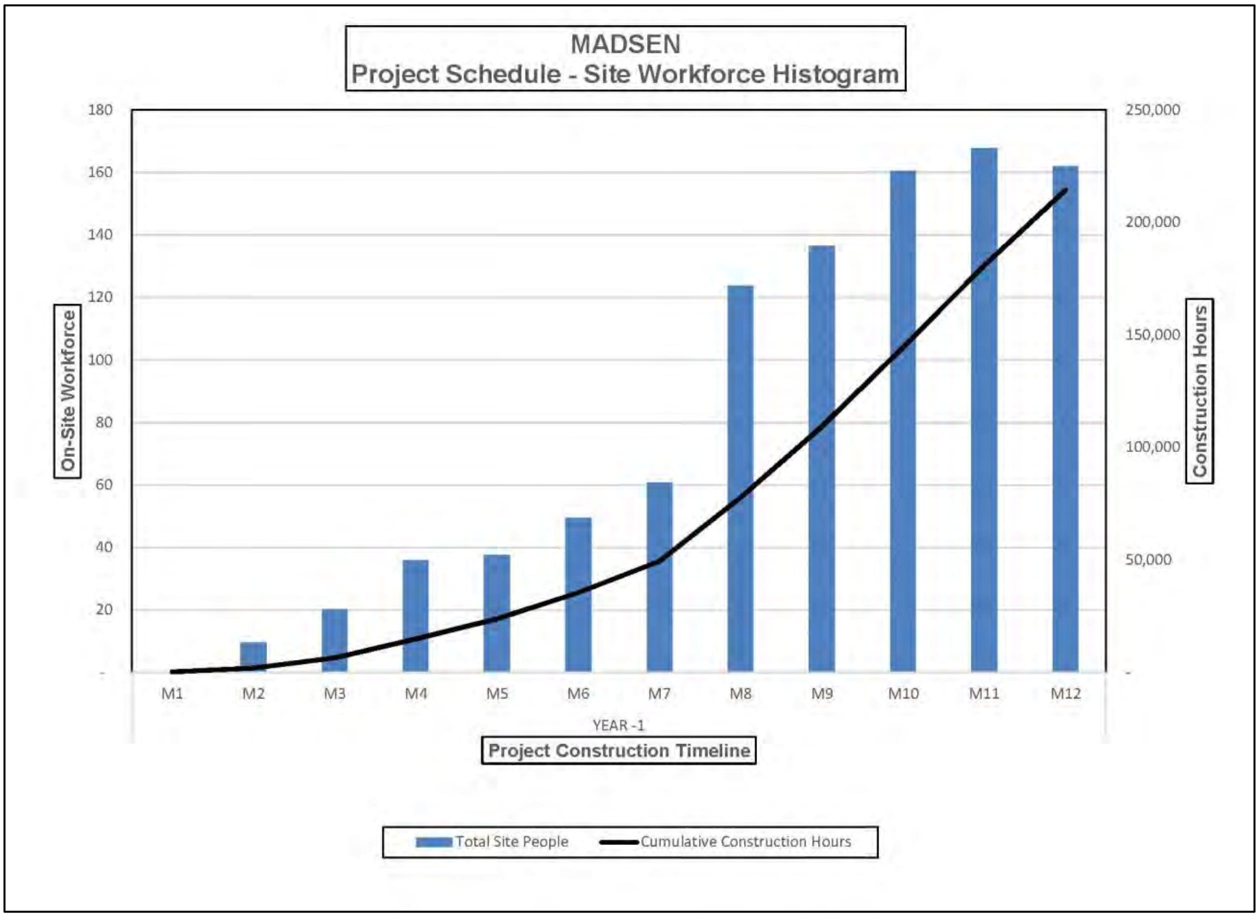
Milestone	Date
Mine Contractor Mobilized to Site *Initiate dewatering and installation of temporary mining support infrastructure*	Month 4
Mine Shaft Rehabilitation Initiated	Month 5
Project Site Construction Start *(mobilize earthworks contractor and begin Project site surface clearing and bulk cut/fill activities at the TMF and plant area;*	Month 5
Begin Process Plant Work *Plant inspection and initial concrete preparation work*	Month 6
Mine Development Initiated	Month 7
Tailings Management Facility Complete	Month 10
HV Power Line and Substations Connections Complete	Month 13
Processing Facilities Complete Construction	Month 14
Process Facilities Commissioning Complete and Commercial Production Declared	Month 15

Source: JDS (2019)

23.8.8 Site Workforce Loading

The site workforce will ramp up gradually during the first seven months, and then increase at about Month 8 of Year -1 as the mining development activities commence. The final five months prior to the start of operations will have an average approximately 140 personnel onsite, with a peak of approximately 160 to 170 people. Figure 23-3 shows the workforce during the pre-production period, on a monthly basis. Total workforce includes construction personnel (both surface and underground), underground mining personnel, EPCM team, vendor representatives, commissioning team and Owner’s team.

Figure 23-3: Construction Period Site Workforce



Source: JDS (2019)

23.9 Commissioning

23.9.1 Commissioning Methodology

Progressive commissioning for the Project will be performed by subsystem. A system will be defined as a logical grouping of equipment or systems that can be placed in service more or less by itself and that contribute to a common purpose or functionality. Wherever possible, facilities will be commissioned early in the development schedule (as in the case of dewatering and injection wells and system, etc.) and be turned over to EM for ownership and operation. A detailed Commissioning Plan will be developed during detailed engineering.

23.9.2 Commissioning Safety and Training

The Health, Safety, and Environmental Plan (HSE Plan) developed during execution will address specific safety procedures that will apply during the commissioning stage of the Project. The commissioning and turnover phase presents significant and unique safety risks. A comprehensive lock-out tag-out program is an effective control to manage these risks.

23.9.3 Commissioning Stages

- Construction Release (Stage 1): Construction contractor completes a system subject to agreed punch list items;
- Pre-Operational Equipment Testing (Stage 2): Energize and test individual equipment within subsystems to ensure functionality includes equipment functionality tests controlled by the Plant Control System (signed off loop diagrams);
- Pre-Operational Systems Testing (Stage 3): Systems tested with water, air and insert materials, and capable of continuous and safe operation with all instrumentation connected, the control system operational, and all interlocks functional;
- Ore Commissioning (Stage 4): Plant ready to accept ore and all operating and maintenance staff are fully trained to operate and maintain the plant; individual systems operate successfully under load for a defined period of time; and
- Ramp-Up (Stage 5): Increase ore feed design throughput rate.

24 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as Project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 and Section 22 of this report (presented in 2019 dollars). The economic analysis has been run with no inflation (constant dollar basis).

24.1 Summary of Results

The summary of the mine plan and payable metals produced is outlined in Table 24-1.

Table 24-1: Life of Mine (LOM) Summary

Parameter	Unit	Value
Mine Life	Years	12.3
Resource Mined	kt	3,512
Gold (Au) Grade	g/t	8.97
Processing Rate	kt/d	0.8
Gold (Au) Payable	koz	970
	koz/a	79

Source: JDS (2019)

Other economic factors include the following:

- Discount rate of 5%;
- Nominal 2019 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;

- Working capital calculated as one month of operating costs (mining, processing, site services and G&A) in Year 1;
- Results are based on 100% ownership and no royalties on The Mineral Reserves;
- No management fees or financing costs (equity fund-raising was assumed); and
- The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 24-2 outlines the metal prices and exchange rate assumptions used in the economic analysis. The gold price selected was based on the 9-month average as of December 2018 and is in line with recently released comparable Technical Reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 24-2: Metal Prices and Exchange Rates

Parameter	Unit	Value
Gold (Au) Price	US$/oz	1,275
FX Rate	CDN$:US$	0.75

Source: JDS (2019)

24.2 Assumptions

Mine revenue is derived from the sale of doré bars into the international marketplace. No contractual arrangements for refining currently exist. Table 24-3 indicates the NSR parameters that were used in the economic analysis.

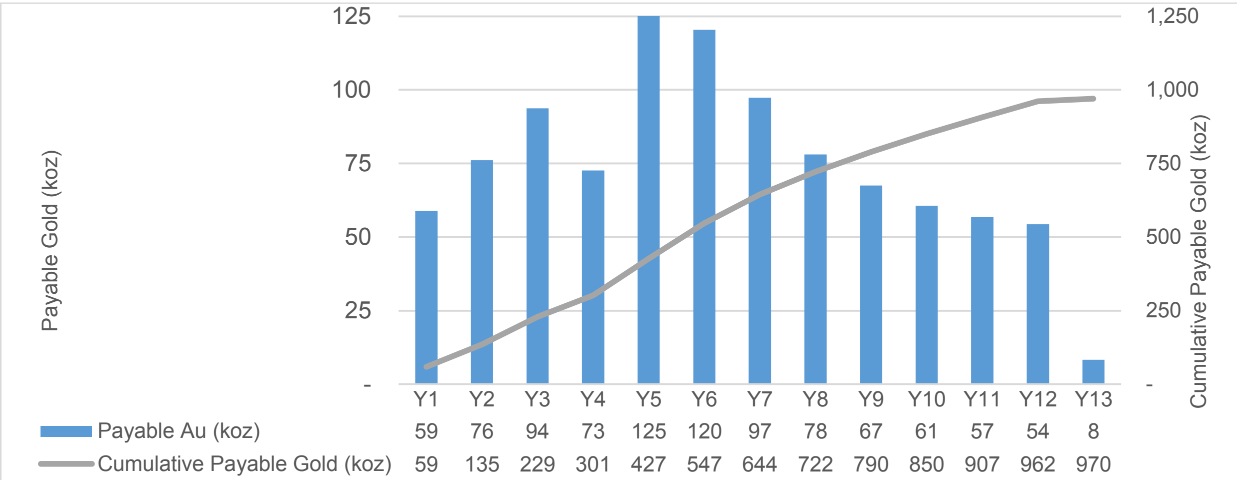
Table 24-3: NSR Parameters

Parameter	Unit	Value
Gold (Au) Recovery	%	95.8
Gold (Au) Payable	%	99.97
Gold (Au) Refining Charge	US$/pay oz	0.38
Transportation	US$/pay oz	1.35
Royalties	%	-

Source: JDS (2019)

Figure 24-1 shows the grade and the amount of gold (Au) recovered during the mine life. A total of 970 koz of gold is projected to be produced over the life of mine.

Figure 24-1: LOM Payable Gold



Source: JDS (2019)

24.3 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the potential Project economics. A tax model was prepared by Wentworth Taylor, an independent tax consultant, and reviewed by JDS and Pure Gold personnel. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- Federal Income Tax: 15%;
- Ontario Mineral Tax: 10%;
- Capital cost allowance applied on units of production basis and at specific rates in the tax act; and
- Exploration carry forwards of $27 M.

Total taxes for the Project amount to $153 M.

24.4 Royalties

The Probable Mineral Reserves for the Madsen Gold Project are not subject to any royalties. Royalties do exist on other lands on the Project and these are described in Table 4-2.

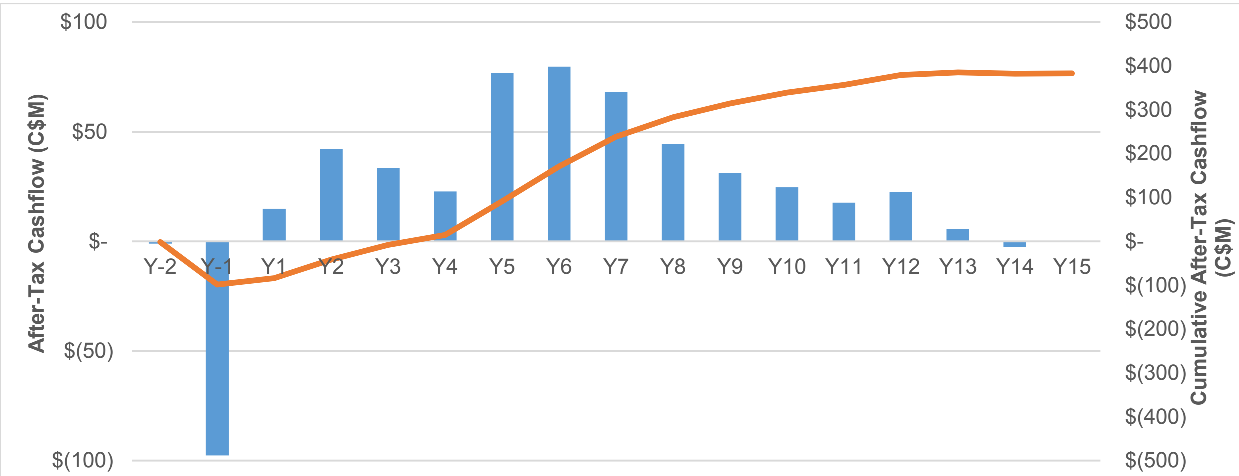
24.5 Results

The Madsen Project is economically viable with a post-tax IRR of 36% and a net present value using a 5% discount rate ($NPV_{5\%}$) of $247 M using the metal prices described in Section 24.1. Figure 24-2 shows the projected cash flows, and Table 24-4 summarizes the economic results of the Project.

The post-tax break-even gold price is approximately US$860/oz, based on the LOM plan presented herein. This is the gold price at which the Project NPV @ 0% discount rate is zero.

The life of mine all-in sustaining cost (AISC) is US$787/oz. The straight AISC cost is calculated by adding the refining, transport, royalty, operating, and sustaining and closure costs together and dividing by the total payable ounces of gold.

Figure 24-2: Annual After-Tax Cash Flow



Source: JDS (2019)

Table 24-4: Summary of Results

Summary of Results	Unit	Value
AISC*	US$/oz	787
Capital Costs		
Pre-Production Capital	M$	87
Pre-Production Contingency	M$	8
Total Pre-Production Capital	**M$**	**95**
Sustaining and Closure Capital	M$	230
Sustaining and Closure Contingency	M$	2
Total Sustaining and Closure Capital	M$	232
Total Capital Costs Incl. Contingency	**M$**	**327**
Working Capital	M$	4
Pre-Tax Cash Flow	LOM M$	536
	M$/a	44
Taxes	LOM M$	153
After-Tax Cash Flow	LOM M$	383
	M$/a	31
Economic Results		
Pre-Tax NPV$_{5\%}$	**M$**	**353**
Pre-Tax IRR	**%**	**43**

Summary of Results	Unit	Value
Pre-Tax Payback	**Years**	**3.0**
After-Tax NPV$_{5\%}$	**M$**	**247**
After-Tax IRR	**%**	**36**
After-Tax Payback	**Years**	**3.4**

*All-in Sustaining Cost is calculated as: (Refining & shipping costs + royalties+ operating costs + sustaining and closure capital) / payable gold ounces.

Source: JDS (2019)

24.6 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -15% to +15%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 15% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

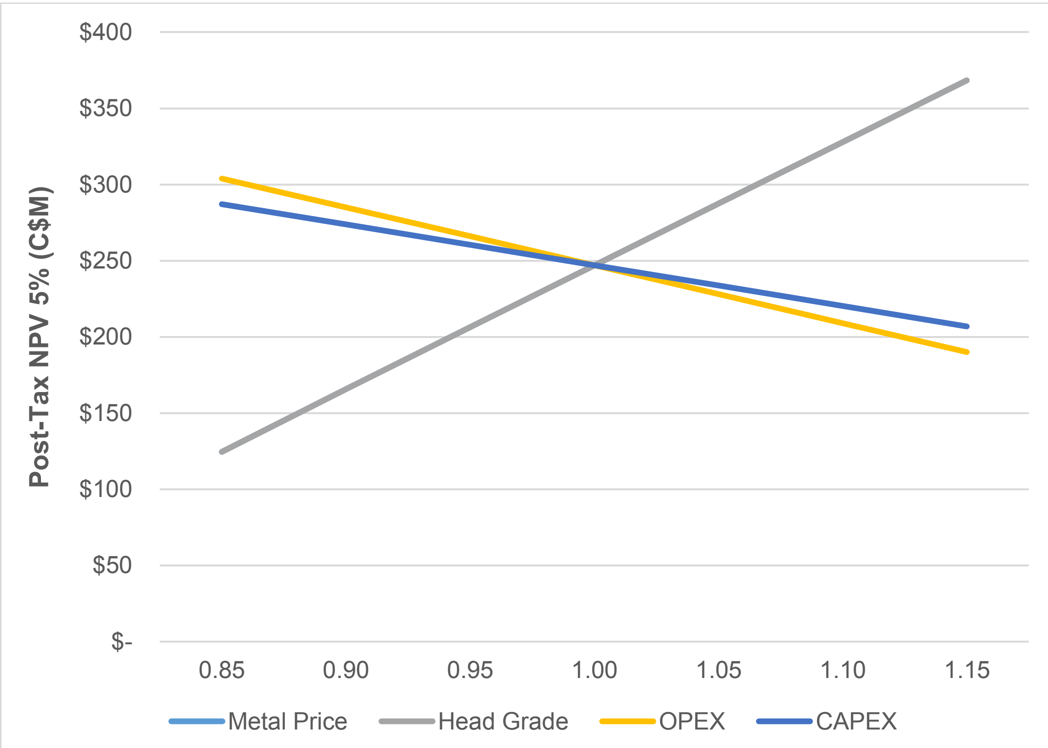
Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the Project is most sensitive to metal prices and head grade. The Project showed the least sensitivity to capital costs. Table 24-5 and Figure 24-3 show the results of the sensitivity tests, while Table 24-6 shows the NPV at various discount rates.

Table 24-5: Pre-Tax and After-Tax Sensitivity Results on NPV @ 5%

Variable	After-Tax NPV$_{5\%}$ (M$)			Pre-Tax NPV$_{5\%}$ (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Metal Price	125	247	368	177	353	529
Mill Head Grade	125	247	368	177	353	529
OPEX	304	247	190	436	353	271
CAPEX	287	247	207	394	353	313

Source: JDS (2019)

Figure 24-3: Post-Tax $NPV_{5\%}$ Sensitivity



Source: JDS (2019)

Table 24-6: Project NPV at Various Discount Rates

Discount Rate (%)	Pre-Tax NPV (M$)	Post-Tax NPV (M$)
0	536	383
5	353	247
6	326	226
7	300	208
8	277	190
10	235	160
12	200	134

Source: JDS (2019)

The economic cash flow model for the Project is illustrated in Figure 24-7.

Table 24-7: Cash Flow Model

Item	Unit	Pre-Production	Production Total	Life of Mine Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15
KEY PARAMETERS																					
Metal Prices																					
Gold (Au)	US$/oz	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275
Exchange Rates																					
F/X	USD:CAD	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
MINING																					
Production																					
Ore Mined	ktonnes	13	3,499	3,512	-	13	285	293	293	289	293	293	293	293	293	293	293	266	43	-	-
Waste Mined	ktonnes	156	3,855	4,011	-	156	537	508	553	469	539	440	148	123	139	131	150	97	20	-	-
Total Mined	ktonnes	170	7,354	7,524	-	170	803	800	846	758	832	733	441	416	432	424	443	363	64	-	-
Operating Days	days	152	4,471	4,623	-	152	365	365	365	365	365	365	365	365	365	365	365	365	91	-	-
Average Mining Rate	tpd	88	783	760	-	88	726	802	802	793	802	802	802	802	802	802	802	728	476	-	-
MINERAL PROCESSING																					
Ore Processed	ktonnes	-	3,512	3,512	-	-	273	293	293	293	293	293	293	293	293	293	293	268	43	-	-
Au Head Grade	g/t	-	8.97	8.97	-	-	7.03	8.52	10.51	8.15	13.71	13.15	10.66	8.62	7.53	6.79	6.36	6.67	6.28	-	-
Contained AU	koz	-	1,013	1,013	-	-	62	80	99	77	129	124	100	81	71	64	60	57	9	-	-
Operating Days	days	-	4,471	4,471	-	-	365	365	365	365	365	365	365	365	365	365	365	365	91	-	-
Average Plant Throughput	tpd	-	786	786	-	-	748	802	802	802	802	802	802	802	802	802	802	733	476	-	-
Rehandle	kt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
RECOVERED METAL																					
Au Recovery	%	0.0%	95.8%	95.8%	0.0%	0.0%	95.6%	94.9%	94.7%	94.7%	97.1%	97.3%	96.9%	96.2%	95.3%	94.9%	94.8%	94.7%	95.0%	0.0%	0.0%
Au Recovered	koz	-	970	970	-	-	59	76	94	73	125	120	97	78	68	61	57	54	8	-	-
SALES & NSR																					
Payable Metals																					
Au Payable	%	0.00%	99.97%	99.97%	0.00%	0.00%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	99.97%	0.00%	0.00%
Au Payable	koz	-	970	970	-	-	59	76	94	73	125	120	97	78	67	61	57	54	8	-	-
Payable Value	US$M	-	1,236.6	1,236.6	-	-	75.2	97.0	119.5	92.6	159.7	153.4	124.0	99.5	86.0	77.3	72.4	69.3	10.6	-	-
Payable Value	C$M	-	1,648.9	1,648.9	-	-	100.2	129.4	159.3	123.5	213.0	204.6	165.4	132.7	114.7	103.1	96.5	92.4	14.2	-	-
Royalties																					
Royalty Rate	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Royalties Paid	US$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Royalties Paid	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Refining Charges																					
Au Refining Unit Charge	CAD$/oz	-	0.50	0.50	-	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	-	-
Au Refining Costs	US$M	-	0.4	0.4	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-
Au Refining Costs	C$M	-	0.5	0.5	-	-	0.0	0.0	0.0	0.0	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-
Transport Charges																					
Shipment Unit Cost	US$/shipment	-	6,800	6,800	-	-	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	6,800	-	-
Shipments	#	-	147	147	-	-	12	12	12	12	12	12	12	12	12	12	12	12	3	-	-
Additional cost by value	US$/US$1000 value	-	0.09	0.09	-	-	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	-	-
Additional cost per by weight	US$/kg	-	6.50	6.50	-	-	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	6.50	-	-
Transportation Cost	US$M	-	1.3	1.3	-	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-	-
Transportation Cost	C$M	-	1.7	1.7	-	-	0.1	0.1	0.1	0.1	0.2	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-	-
Net Smelter Return																					
Net Smelter Return	**US$M**	-	**1,235.0**	**1,235.0**	-	-	**75.0**	**96.9**	**119.3**	**92.5**	**159.6**	**153.3**	**123.9**	**99.4**	**85.9**	**77.2**	**72.2**	**69.2**	**10.6**	-	-
(before Royalties)	**C$M**	-	**1,646.6**	**1,646.6**	-	-	**100.1**	**129.2**	**159.1**	**123.4**	**212.8**	**204.4**	**165.2**	**132.5**	**114.6**	**102.9**	**96.3**	**92.2**	**14.1**	-	-
	US$M	-	**1,235.0**	**1,235.0**	-	-	**75.0**	**96.9**	**119.3**	**92.5**	**159.6**	**153.3**	**123.9**	**99.4**	**85.9**	**77.2**	**72.2**	**69.2**	**10.6**	-	-
Net Smelter Return	US/t	-	351.62	351.62	-	-	274.74	330.88	407.47	315.97	545.02	523.50	423.09	339.49	293.43	263.55	246.71	258.42	243.80	-	-
(after Royalties)	**C$M**	-	**1,646.6**	**1,646.6**	-	-	**100.1**	**129.2**	**159.1**	**123.4**	**212.8**	**204.4**	**165.2**	**132.5**	**114.6**	**102.9**	**96.3**	**92.2**	**14.1**	-	-
	C$/t	-	468.83	468.83	-	-	366.32	441.18	543.29	421.30	726.69	698.00	564.12	452.65	391.24	351.41	328.94	344.56	325.07	-	-
OPERATING COSTS																					
Mining	C$/t ore mined	-	169.44	168.79	-	-	158.78	171.93	173.18	186.41	171.19	159.11	160.52	164.59	170.33	170.21	172.68	168.16	204.71	-	-
	C$M	-	592.8	592.8	-	-	42.1	50.3	50.7	54.0	50.1	46.6	47.0	48.2	49.9	49.8	50.6	44.7	8.9	-	-
Processing	C$/t processed	-	32.27	32.27	-	-	32.89	32.00	31.53	32.00	31.53	32.00	32.00	32.00	32.00	32.00	32.00	33.82	43.43	-	-
	C$M	-	113.3	113.3	-	-	9.0	9.4	9.2	9.4	9.2	9.4	9.4	9.4	9.4	9.4	9.4	9.1	1.9	-	-
General & Administration	C$/t processed	-	21.84	21.84	-	-	25.57	24.00	24.25	22.76	22.87	20.82	19.82	19.59	19.17	19.14	19.19	20.76	51.10	-	-
	C$M	-	76.7	76.7	-	-	7.0	7.0	7.1	6.7	6.7	6.1	5.8	5.7	5.6	5.6	5.6	5.6	2.2	-	-
Total Operating Costs	**C$/t processed**	-	**222.90**	**222.90**	-	-	**212.60**	**227.93**	**228.96**	**239.02**	**225.60**	**211.92**	**212.34**	**216.18**	**221.50**	**221.35**	**223.86**	**221.43**	**299.24**	-	-
	C$M	-	**782.9**	**782.9**	-	-	**58.1**	**66.7**	**67.0**	**70.0**	**66.1**	**62.1**	**62.2**	**63.3**	**64.9**	**64.8**	**65.5**	**59.3**	**13.0**	-	-
INCOME																					
Net Operating Cash Flow	**C$M**	-	**863.7**	**863.7**	-	-	**42.0**	**62.4**	**92.0**	**53.4**	**146.7**	**142.3**	**103.0**	**69.2**	**49.7**	**38.1**	**30.8**	**33.0**	**1.1**	-	-
	C$/t processed	-	**245.92**	**245.92**	-	-	**153.71**	**213.25**	**314.34**	**182.28**	**501.09**	**486.08**	**351.78**	**236.47**	**169.74**	**130.06**	**105.08**	**123.12**	**25.83**	-	-
CAPITAL COSTS																					
Mining	C$M	31.2	209.2	240.4	-	31.2	26.5	20.4	42.0	24.4	34.3	27.4	5.5	7.7	7.0	4.7	6.4	2.5	0.4	-	-
Site Development	C$M	0.8	-	0.8	-	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mineral Processing	C$M	17.4	-	17.4	-	17.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Management	C$M	4.3	7.9	12.1	-	4.3	0.0	-	6.8	0.5	-	0.2	0.1	-	-	0.2	-	-	-	-	-
Site Services	C$M	17.5	0.5	18.0	-	17.5	0.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Closure	C$M	-	16.8	16.8	-	-	-	-	-	-	-	-	5.1	0.5	-	-	-	-	-	9.3	1.9
Salvage	C$M	-	-4.2	-4.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-2.1	-2.1
Indirects	C$M	6.1	-	6.1	0.1	6.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EPCM	C$M	7.0	-	7.0	1.0	6.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Costs	C$M	2.6	-	2.6	-	2.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal - Pre-Contingency	**C$M**	**87.0**	**230.1**	**317.1**	**1.0**	**85.9**	**27.0**	**20.4**	**48.8**	**25.0**	**34.3**	**27.6**	**10.7**	**8.2**	**7.0**	**4.9**	**6.4**	**2.5**	**0.4**	**7.1**	**-0.2**
Contingency %	%	9.3%	0.9%	3.2%	0.0%	9.4%	0.0%	0.0%	4.3%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Contingency	C$M	8.1	2.1	10.2	-	8.1	-	-	2.1	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Costs	**C$/t processed**	-	**66.10**	**93.17**	-	-	**98.96**	**69.82**	**173.94**	**85.31**	**117.25**	**94.18**	**36.48**	**28.17**	**23.81**	**16.63**	**21.81**	**9.17**	**8.35**	-	-
	C$M	**95.1**	**232.2**	**327.3**	**1**	**94**	**27**	**20**	**51**	**25**	**34**	**28**	**11**	**8**	**7**	**5**	**6**	**2**	**0**	**7**	**-0**
WORKING CAPITAL																					
Working Capital & Reserves Accounts																					
Working Capital	C$M	3.5	-3.5	-	-	3.5	-	-	-	-	-	-	-	-	-	-	-	-	-3.5	-	-
CASH FLOWS																					
Pre-Tax																					
Net Pre-Tax Free Cash Flow	**C$M**	**-98.6**	**635.1**	**536.5**	**-1.0**	**-97.6**	**15.0**	**42.0**	**41.1**	**28.4**	**112.4**	**114.7**	**92.3**	**61.0**	**42.7**	**33.2**	**24.4**	**30.5**	**4.3**	**-7.1**	**0.2**
Cumulative Pre-Tax Free Cash Flow	**C$M**			**536.5**	**-1.0**	**-98.6**	**-83.6**	**-41.7**	**-0.5**	**27.8**	**140.2**	**255.0**	**347.3**	**408.3**	**451.0**	**484.2**	**508.6**	**539.1**	**543.4**	**536.3**	**536.5**
Post-Tax																					
Income Taxes	C$M	-	111.6	111.6	-	-	-	-	7.6	5.5	26.3	25.6	17.1	11.4	8.0	5.9	4.6	5.5	1.3	-4.3	-0.4
Ontario Mining Tax	C$M	-	41.8	41.8	-	-	-	-	-	0.1	9.3	9.5	7.2	5.1	3.6	2.6	2.0	2.4	-	-	0.0
Net After-Tax Free Cash Flow	**C$M**	**-98.6**	**481.6**	**383.0**	**-1.0**	**-97.6**	**15.0**	**42.0**	**33.5**	**22.8**	**76.8**	**79.7**	**68.0**	**44.5**	**31.1**	**24.7**	**17.7**	**22.5**	**5.5**	**-2.8**	**0.6**
Cumulative After-Tax Free Cash Flow	**C$M**			**383.0**	**-1.0**	**-98.6**	**-83.6**	**-41.7**	**-8.2**	**14.6**	**91.4**	**171.1**	**239.1**	**283.6**	**314.7**	**339.4**	**357.1**	**379.7**	**385.2**	**382.4**	**383.0**

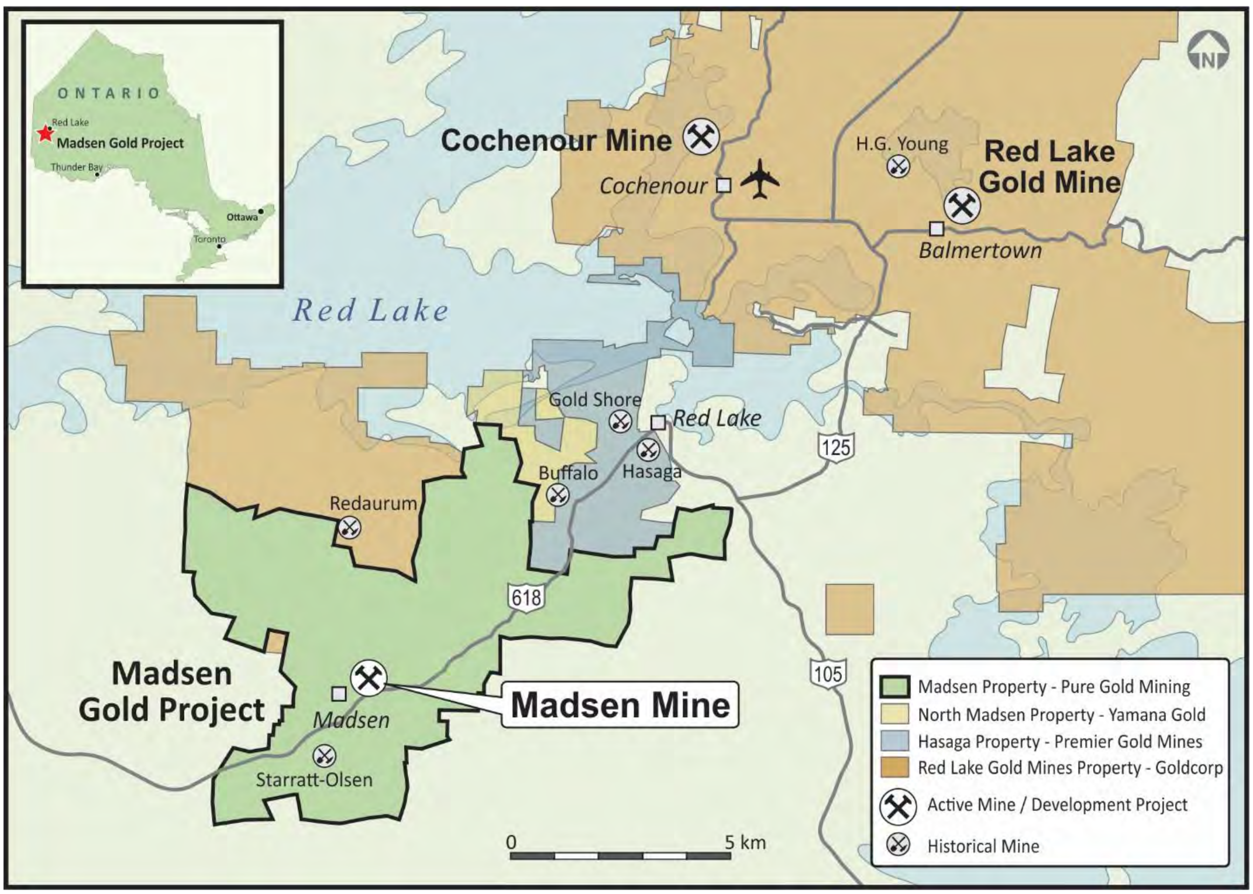
Source: JDS (2019)

25 Adjacent Properties

Relevant information is provided herein for three adjacent properties to the Madsen Property – the Hasaga Property of Premier Gold Mines Limited (Premier) (Jourdain et al., 2017); the North Madsen Property of Yamana Gold Inc. (Yamana) (McCracken and Utiger, 2014); and the Red Lake Gold Mines Property of Goldcorp Inc. (Goldcorp) (Goldcorp Inc., 2017).

The qualified person has been unable to verify the information provided with respect to the adjacent properties which was obtained from publicly disclosed documents as indicated and such information is not necessarily indicative of the mineralization on the Madsen Property. The proximity and geologic similarities between these adjacent properties and Madsen does not mean that Pure Gold will obtain similar results on the Madsen Property.

Figure 25-1: Location of Madsen Project and Adjacent Properties



Source: Pure Gold (2018)

25.1 Hasaga Property – Premier Gold Mines Limited

Premier has recently been exploring the Hasaga Property which is contiguous to the Madsen Property on the northeast boundary (Figure 25-1). 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 (Lichtblau et al., 2017). Exploration has been conducted on this property since 1927 and recently (2015–2016) Premier has completed diamond-drilling of 259 holes, totaling 110,166 m (Jourdain et al., 2017).

25.1.1 Hasaga Property Mineral Resource Estimate

Hasaga was the largest deposit where mineralization is hosted within a quartz-feldspar porphyry dyke which intruded Balmer Assemblage basalt. Gold mineralization is structurally controlled and occurs in veins, lenses and fractures. Competency contrast between host rocks provided an important focus for gold mineralizing fluids (Jourdain et al., 2017). Table 25-1 summarizes the recent mineral resource estimate for the Hasaga Property.

Table 25-1: Hasaga Property Mineral Resource Estimate

Zone	Category	Tonnage	Grade (g/t)	Cut-off Grade (g/t)	Contained Ounces
Central	Indicated Resources	31,613,000	0.8	0.5	803,900
Central	Inferred Resources	23,733,000	0.8	0.5	582,700
Hasaga	Indicated Resources	9,050,000	0.9	0.5	258,100
Hasaga	Inferred Resources	806,000	1.0	0.5	26,000
Buffalo	Indicated Resources	1,632,000	1.2	0.5	61,900
Buffalo	Inferred Resources	604,000	1.1	0.5	21,800

Source: Jourdain et al. (2017)

Pure Gold holds a 1.0% net smelter return royalty on the southwestern portion of the Hasaga Property (Buffalo Claims). The proximity and geologic similarities of Hasaga does not mean that Pure Gold will obtain similar results on the Madsen Property.

25.2 North Madsen Property – Yamana Gold Inc.

Recently, Yamana has been exploring their North Madsen Property which is contiguous along on the northeast boundary of Pure Gold's Madsen Property. The North Madsen Property has been explored since 1925, however no gold production has resulted.

25.2.1 North Madsen Property Mineral Resource Estimate

Table 25-2 summarizes the recent mineral resource estimate 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 over-printing quartz-tourmaline veins (McCracken and Utiger, 2014). The proximity and geologic similarities of the North Madsen Property does not mean that Pure Gold will obtain similar results on the Madsen Property.

Table 25-2: North Madsen Property Mineral Resource Estimate

Category	Tonnage	Grade (g/t)	Cut-off Grade (g/t)	Contained Ounces
Measured Resources	16,728,310	1.3	0.6	685,891
Indicated Resources	6,230,600	1.0	0.6	202,862
Measured and Indicated Resources	22,958,910	1.2	0.6	888,752
Inferred Resources	10,138,000	1.2	0.6	383,936

Source: McCracken and Utiger (2014)

25.3 Red Lake Gold Mines Property – Goldcorp Inc.

Goldcorp's Red Lake Gold Mines Property is contiguous to the Madsen Property on the northern boundary and the Madsen Mine and the Red Lake Gold Mines (RLGM) complex are approximately 16 km apart. The RLGM is the largest mining operation in the Red Lake mining district and has been in continuous operation since 1948. At January 1, 2018, the workforce comprised 876 full-time employees and 168 contractors. Mines on what is now the RLGM Property have produced more than 24.7 million ounces of gold to 2018 including gold production of 209,000 ounces in 2017 (Lichtblau et al., 2018).

25.3.1 RLGM Reserves and Resources

Table 25-3 provides Reserve and Resource values for the RLGM as recently disclosed by Goldcorp (Goldcorp Inc., 2019).

Table 25-3: RLGM Reserves and Resources

Category	Tonnage	Grade (g/t)	Cut-off Grade (g/t Au)	Contained Ounces
Proven Reserves	1,530,000	10.2	7.5*	500,000
Probable Reserves	4,730,000	9.1	7.5*	1,390,000
Probable Reserves (Stockpiles)	2,930,000	1.73	not reported	160,000
Proven + Probable Reserves	9,190,000	6.9	not reported	2,050,000
Measured Resources - Red Lake	1,500,000	18.28	6.9	880,000
Measured Resources - Cochenour	30,000	9.9	5.6	10,000
Indicated Resources- Red Lake	3,200,000	14.1	6.9	1,450,000
Indicated Resources - Cochenour	580,000	10.4	5.6	190,000
Inferred Resources – Red Lake	3,540,000	15.7	6.9	1,790,000
Inferred Resources – Cochenour	1380,000	13.57	5.6	600,000

*Mineral Reserves are reported using variable cut-off grades depending on the mineralization type and zone. The mineral reserve cut-off grade averages 7.90 g/t Au. *Mineral Resources are reported using variable cut-off grades depending on the mineralization type and zone. The mineral resource cut-off grade averages 6.9 g/t Au for the Red Lake Complex and 5.6 g/t Au for the Cochenour Complex.

Source: Goldcorp Inc. (2019)

The RLGM deposits are hosted by basalts of the Balmer Assemblage. The host rock sequence has been intruded by Balmer-aged ultramafic, mafic and felsic dykes, and sills and uncomformably overlain by felsic and intermediate volcanic, volcaniclastic, and sedimentary rocks of the Bruce Channel Assemblage and Confederation Assemblage calc-alkaline volcanic rocks. The deposit lies near the transition between greenschist and amphibolite metamorphic facies. Key, early alteration assemblages comprise widespread pervasive carbonatization and aluminous alteration. Potassic alteration (biotite and potassium feldspar) and silicification overprint the earlier alteration. Gold is associated with the silicification event and is overprinted by a second quartz-tourmaline +/- gold event associated with the Dome Stock. Ore types include silica replaced carbonate veins, siliceous replacement-style mineralization, disseminated sulphide mineralization along major shear zones, and minor sulphidized chemical sedimentary rock-hosted ore (Cadieux et al., 2006).

The Red Lake Mill has a permitted capacity of 1250 t/d and the Campbell Mill has a permitted capacity of 1800 t/d. On the basis of Mineral Reserves only, the life-of-mine production plan is based on ten years of underground production to 2028 and reflects six years of production at an average annual rate of approximately 250,000 oz/year, followed by four additional years of decreasing production (Goldcorp, 2019).

A 12,900 t bulk sample was completed at the Cochenour Project in 2018. The project is in the execution phase, establishing material handling, pastefill and other, and infrastructure towards the planned start to commercial production in 2019.

Near mine exploration in 2019 is focused on the Aviation Complex and targets proximal to the High Grade Zone. Regional exploration targets considered by Goldcorp (2019) to have the potential to host a significant new gold deposit include:

- The Western Discovery area;
- Wilmar, in joint venture with Premier Gold;
- West Red Lake in joint venture with West Red Lake Mines Ltd.; and
- North Madsen, immediately adjacent the Madsen Property.

The proximity and geologic similarities to RLGM does not mean that Pure Gold will obtain similar results on the Madsen Property.

26 Other Relevant Data and Information

26.1 Preliminary Economic Assessment of the Fork, Russet South, and Wedge Deposits

The results of a Preliminary Economic Assessment (PEA) of gold mineralization contained in the Fork, Russet South, and Wedge deposits are considered relevant to the conclusions and recommendations of the Madsen Gold Project Feasibility Study, since they describe Pure Gold's work on this project and indicate the future potential of the Project.

This Section summarizes the results of the PEA and is prepared according to the guidelines of the Canadian Securities Administrator's National Instrument 43-101 (NI 43-101) and Form 43-101F1.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the project presented in the PEA will be realized.

The PEA summarizes and evaluates the mining, processing and closure of the Fork, Russet South and Wedge deposits that are mined and processed after the Madsen deposit described in the Feasibility Study portion of this report. The PEA assumes that all mining and processing infrastructure described in the FS are completed as outlined. Only the contents relevant to the PEA are contained in this section in order to avoid duplicating and repeating the same information presented in Sections 2-13, 17 and 18 of this Technical Report.

26.2 Mineral Processing and Metallurgical Testing

Based on the test work flowsheet and reagent scheme completed on the Madsen deposit confirmatory test work was performed on the Fork, Russet South, and Wedge deposits. The results are summarized below in Table 26-1.

Table 26-1: Summary of Leach Results

Sample ID	Head Grade	Gold Extraction – Percent Cumulative over Time, h					Reagent Consumption, Kg/t	
	Au, g/t	0	2	6	24	48	NaCN	Lime
Fork	5.0	58.1	89.4	95.7	95.8	97.0	0.28	0.68
Russet South	6.1	60.4	90.3	97.0	98.3	98.5	0.17	0.70
Wedge	11.9	71.8	95.0	97.8	98.2	98.4	0.24	0.74

Source: BaseMet (2018)

26.2.1 Fork, Russet South, and Wedge Deposits – Test Program BL0354

Drill core representing the Fork, Russet South, and Wedge deposits, were provided for metallurgical test work at BaseMet. The drill core was used to create three global composites. The samples were sent out for mineralogy analysis (BMA). Similar to the Madsen deposit the sulphide mineral content is mainly associated with Pyrrhotite and then Pyrite.

BaseMet completed comminution test work and leach test work using the optimized flowsheet from the Madsen test program (BL0288) on all three composites. The comminution results indicate the material is moderately hard with a BWi ranging from 13.4 kWh/t to 14.4 kWh/t. The abrasion index ranged from 0.250 to 0.281 g similar to the Madsen deposit. Gravity leach test work results demonstrate high gravity recovery to the pan concentrate ranging from 61.1% to 74.3%. The final results indicate gold extraction in the range of 96.5% to 98.3% in 24 hours with over 61% of the gold recovered in the gravity circuit.

A processing gold recovery of 95.8% has been assumed for the Fork, Russet South, and Wedge deposits.

26.3 Mining

Mining of the Fork, Russet South, and Wedge deposits will consist of three separate mines using underground mining methods. Each mine will have its own portal with a decline ramp serving as the main haulage route and ventilation exhaust. Level spacing between footwall drives for the Fork and Wedge Mines will be in 20 m vertical increments, between 170 to 330 levels and -40 to 380 levels respectively. Levels will be connected by the primary spiral ramp, sized at 4.2 m W x 4.2 m H and located in the footwall of the deposits. Level spacing between footwall drives for the Russet South deposit are located approximately in 13 m vertical increments, 202 to 328 levels. Levels will be connected by the primary spiral ramp, sized at 4.2 m W x 4.2 m H and located in the footwall of the deposit.

Fork and Wedge mines will be mined by longitudinal long-hole with rockfill as backfill replacement in the mined-out voids. Wider portions of the orebody, over 30 m along strike, will employ permanent pillars to avoid the requirements of cemented self-standing backfill. Russet South deposit will be mined by MCF with rockfill (RF) as backfill replacement in the mined out voids.

Mineralized material will be extracted from the three mines at a combined rate of 800 t/d. Underground haul trucks will haul the mineralized material to surface and dump it at their respective portal stockpiles. Once on surface, the mineralized material will be loaded onto surface highway dump trucks and transported to the Madsen Mill, at a distance of 2.1, 3.1, and 13.5 km respectively from the Fork, Russet South, and Wedge mines.

Un-mineralized material will be hauled to surface via underground haul trucks. At each portal, there is a designated dumping location were the material will be stockpile for rockfill. Once a mining panel has been exhausted, the void will be filled with rockfill from either un-mineralized underground development or back hauled from the surface rockfill stockpile. Each mine is expect to produce a surplus of rockfill material. To reduce surface re-handling of material, the remaining rockfill stockpile will transition to a MRMF.

26.3.1 Geotechnical Analysis and Recommendations

26.3.1.1 Geotechnical Data and Characterization

Geotechnical specific drilling and testing programs have not yet been carried out for the project; however, Pure Gold has logged basic geotechnical parameters such as rock quality designation (RQD), total core recovery (TCR) and fracture count on a number of resource drill holes within the Fork, Russet South, and

Wedge deposits. JDS has spot checked the Pure Gold geotechnical data against the respective core photographs for internal data consistency and considers the data to be suitable for a PEA of study.

The existing Pure Gold database was supplemented with estimates of more detailed geotechnical parameters based on review of core photographs for intervals from a total of 10 drill holes. The drill hole intervals selected included three drill holes at Fork, two at Russet South and five at the Wedge deposit.

High level estimates of rock mass quality suitable for a PEA were made by JDS according to the Barton Q' rock mass rating system for the immediate 6 to 10 m of hanging wall and footwall as well as the mineralized zones. The RQD data from the Pure Gold database was used along with reasonably conservative estimates of the number of joint sets (Jn) and joint condition parameters (Jr and Ja) based on core photograph review, along with more detailed geotechnical assessments completed at the main deposits (Section 16.3) and experience in similar geologic environments to determine Q'. Table 26-2 contains a summary of the mean RQD, TCR and fracture frequency per meter (ff/m) as well as the minimum, maximum and mean Q' values that were developed from the core photograph logging.

Table 26-2: Rock Mass Quality Parameters by Deposit and Zone

Deposit	Zone	No. of Runs	Mean TCR (%)	Mean RQD (%)	Mean ff/m	Q' [A, B]		
						Min.	Mean	Max.
Fork	Hanging Wall	7	100	100	2.9	3.6	11.9	17.2
	Veins	20	100	99	2.4	5.6	12.1	18.8
	Footwall	7	99	97	5.0	1.3	13.1	18.8
Wedge	Hanging Wall	10	99	80	6.0	2.1	6.9	12.5
	Veins	32	100	89	3.5	4.2	13.9	37.5
	Footwall	12	100	97	2.9	5.5	9.5	18.8
Russet South	Hanging Wall	6	97	88	4.8	3.4	7.0	13.1
	Veins	25	99	78	7.4	0.6	7.6	18.8
	Footwall	6	99	86	4.8	2.0	5.4	12.3

[A] Q' is calculated by setting the Joint Water Factor (Jw) and Stress Reduction Factor (SRF) both equal to 1 in the Q equation.
Source: JDS (2019)

The Fork and Wedge deposits generally appear to have similar rock mass quality with mean Q' values between ranging of 12 and 14, respectively, classifying as 'Good' quality rock mass according to the Q classification system. The hanging walls and footwalls appear slightly lower Q' and more variable but still mostly 'Good' rock mass quality with some 'Fair' quality ground. Similar to the Austin and McVeigh deposits, very few brittle fault structures have been encountered in drilling or are anticipated within the deposits.

The Russet South deposit is dominantly Balmer basalt hosted with minor ultramafic hosted mineralization. The rock mass is typically of lower quality compared to Fork and Wedge. Mean Q' values ranged between 5 and 8 classifying as 'Fair' rock mass quality.

26.3.2 Stope Dimensions

Maximum stope dimensions were estimated using the Potvin (2001) method for the anticipated range of ground conditions and stope dimensions. The Trueman & Mawdesley (2003) 'Stable' line was then used as a check against the Potvin (2001) results. Empirical factors used in the calculation of stability numbers (N') were based on the following assumptions:

- Induced Stress Factor, A, was assumed to be 1.0 (least conservative) due to the relatively strong UCS typical to the Madsen main zones and the low horizontal stress anticipated for the shallow stoping depths (200 to 400 m below ground surface);
- Joint Orientation Factor, B, was set to 0.2 (most conservative) to 0.3 for the stope hanging walls and footwalls in anticipation of the dominant discontinuity orientation being sub-parallel to the mineralization; and,
- Gravity Factor, C, for stope walls, except for the footwall, were calculated based on dip angle using the following equation: C = 8 – 6 cos (face dip angle). The maximum C value of 8.0 was assumed for the footwall due to its favorable dip orientation.

For Wedge and Fork LH mining, the maximum level spacing was set at 20 m (floor to floor) and the maximum stable stope lengths and widths were estimated from the stability graph for the stope hanging walls and backs. A maximum hydraulic radius was then estimated from the stability graph for each stope wall and deposit as summarized in Table 26-3. Figure 26-1 contains output plots from a typical empirical stope stability analysis for Longitudinal LH stoping at the Fork deposit. Given the small cut and fill stope opening dimensions for Russet South and that the ground will be fully supported, the empirical stope design and ELOS charts were considered as a check but not relied upon for geotechnical design.

Table 26-3: Maximum Hydraulic Radii for Each Deposit and Stope Wall

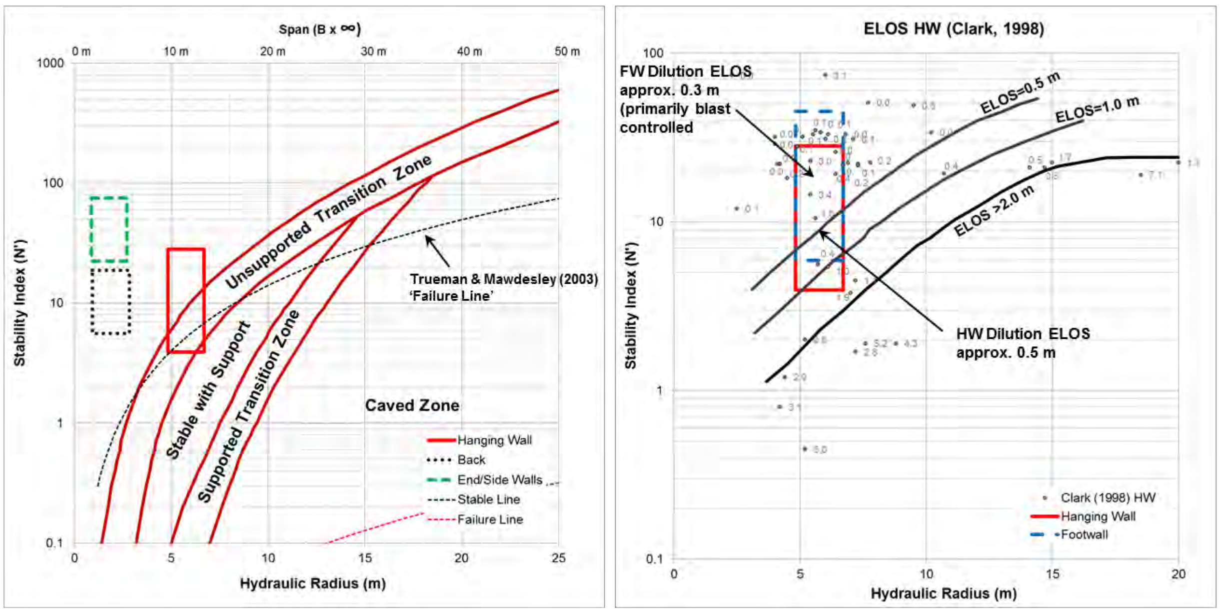
Zone	Stope Wall HR (m)			Comment
	Hanging Wall (along strike)	Back (flat)	End (perp. to strike)	
Fork	6.5	6	10	Controlled by HR of HW
Wedge	6	6	10.5	Controlled by HR of HW
Russet South	6	6	10.5	Controlled by HR of Back

Source: JDS (2019)

Figure 26-1: Example Empirical Stope Stability Analyses and Dilution Estimate (Fork Longitudinal LHOS)

Wall	Dip	Q'	Empirical Parameters			N'
			A (Stress)	B (Joint Orientation)	C (Dip Angle)	
HW	65	3.6	1	0.2	5.5	3.9
FW	65	3.7	1	0.2	8.0	5.9
Back	0	5.6	1	0.5	2.0	5.6
End/Side	90	5.6	1	0.5	8.0	22.4
HW	65	17.2	1	0.3	5.5	28.2
FW	65	18.8	1	0.3	8.0	45.1
Back	0	18.8	1	0.5	2.0	18.8
End/Side	90	18.8	1	0.5	8.0	75.2

Max. and Min. Ranges		
Width	Height	Length
7.0	24.2	30.0
2.0	14.2	30.0



Source: JDS (2019)

26.3.3 Estimates of Unplanned Dilution

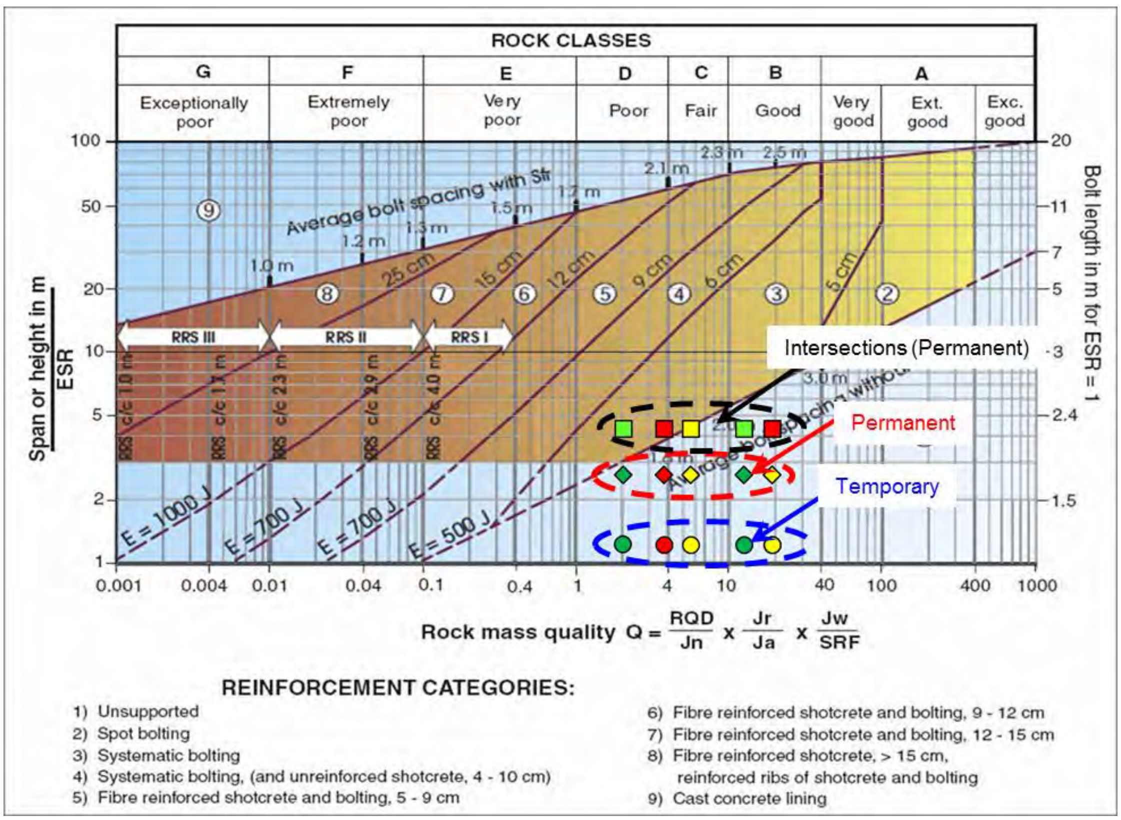
The potential for unplanned dilution was estimated for stope hanging walls and footwalls using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1998). The estimates are approximated in terms of the average thickness spread over the entire hanging wall or footwall. Figure 26-1 contains a typical ELOS plot for Longitudinal LH stoping at the Fork deposit. The following conclusions were made from analysis of the various ELOS graphs developed:

- An ELOS value of 0.5 m is estimated for unplanned dilution for the Fork and Wedge hanging walls;
- Unplanned dilution for the Fork and Wedge footwalls will be primarily blast controlled. An ELOS of 0.3 m can be used for estimation purposes; and
- Unplanned dilution for Russet South cut and fill will be primarily blast and excavation controlled. The ELOS methodology does not apply to cut and fill mining because the ground will be fully supported. The ELOS graph is not applicable to small hydraulic radii (less than approx.4 to 5).

26.3.4 Ground Support

Based on the range of anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support requirements were initially assessed according to the Barton & Grimstad (1994) criteria (Figure 26-2). The Q-system also accounts for the life and use of the opening (e.g. man-entry or equipment only) with the excavation support ratio (ESR) parameter. The ESR is used to adjust the design span, in order to obtain the equivalent span for use in the Q Support Diagram; in effect, it imposes a higher factor of safety on critical structures with long service lives (such as an underground nuclear power station with an ESR rating of 0.5 to 0.8) than on temporary tunnels (such as temporary mine workings with an ESR rating of 2 to 5). An ESR of 1.6 was used for all permanent and man-entry ore development with temporary, non-entry ore development assessed assuming a less conservative ESR of 3.

Figure 26-2: Barton's Q Ground Support Chart



Source: JDS (2019)

Based on the Barton & Grimstad (1994) criteria, most of the non-intersection development would require only spot bolting as a minimum; however, pattern bolting and welded wire mesh are recommended to control loose material for all development where personnel will enter. The ground support recommendations are as follows:

- Permanent Development (4.2 m W x 4.2 H):

  - o 1.8 m long #7 resin bolts on 1.2 m ring spacing and 1.2 m within the ring with 6-gauge welded wire mesh in back to within 1.5 m of floor.

- Temporary / Ore Development (4.2 m W x 4.2 H):
  - o 1.5 m long, Swellex Pm12 on 1.2 m ring spacing and 1.5 m within the ring with 6-gauge welded wire mesh in back to within 1.5 m of floor.

26.3.5 Mine Access and Development

26.3.5.1 Fork Portal

The Fork Portal, located at an elevation of 375 masl, will be the primary haulage portal and will include a laydown pad for run of mine rock, sea cans for storage and a first aid/lunchroom. The surface footprint will be approximately 8,800 m².

The portal excavation has been designed with a rock cut slope of approximately 3:1.5 V:H. Rock fill slopes have been set to approximately 1:1.5 V:H. A general arrangement of the portal is shown in Figure 26-3.

Figure 26-3: Fork Portal Location

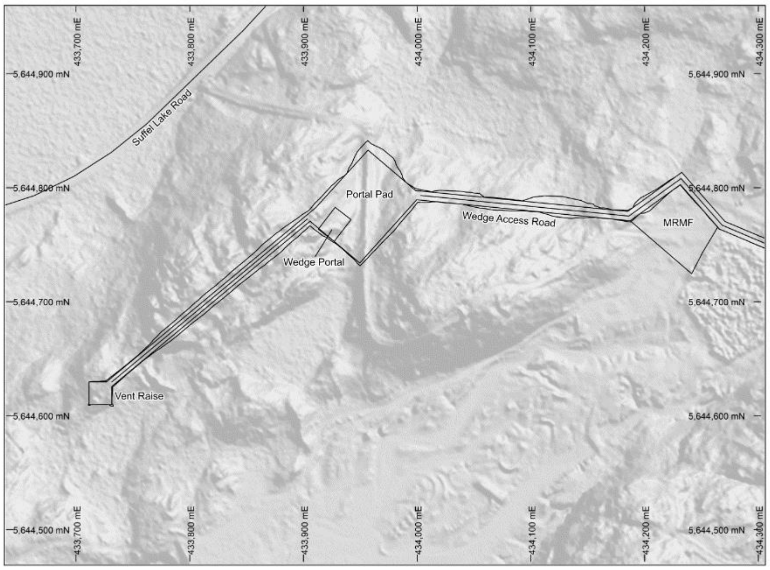
Source: JDS (2019)

26.3.5.2 Wedge Portal

The Wedge Portal, located at an elevation of 395 masl, will be the primary haulage portal and will include a laydown pad for run of mine rock, sea cans for storage and a first aid/lunchroom. The surface footprint will be approximately 4,900 m².

The portal excavation has been designed with a rock cut slope of approximately 3:1.5 V:H. Rock fill slopes have been set to approximately 1:1.5 V:H. A general arrangement of the portal is shown in Figure 26-4.

Figure 26-4: Wedge Portal Location



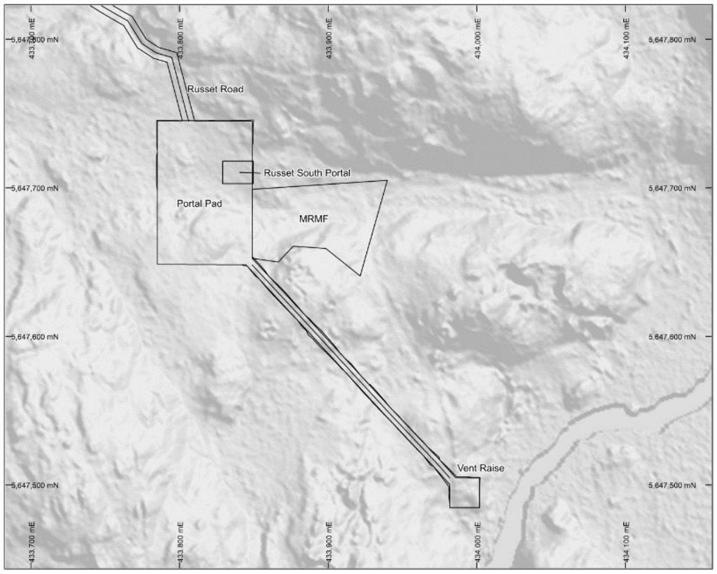
Source: JDS (2019)

26.3.5.3 Russet South Portal

The Russet South Portal, located at an elevation of 384 masl, will be the primary haulage portal and will include a laydown pad for mineralized stockpile, sea can storage and a first aid/ lunchroom. The surface footprint will be approximately 6,200 m².

The portal excavation has been designed with a rock cut of approximately 3:1.5 V:H. Rock fill slopes have been set to approximately 1:1.5 V:H. A general arrangement of the portal is shown in Figure 26-5.

Figure 26-5: Russet South Portal Location



Source: JDS (2019)

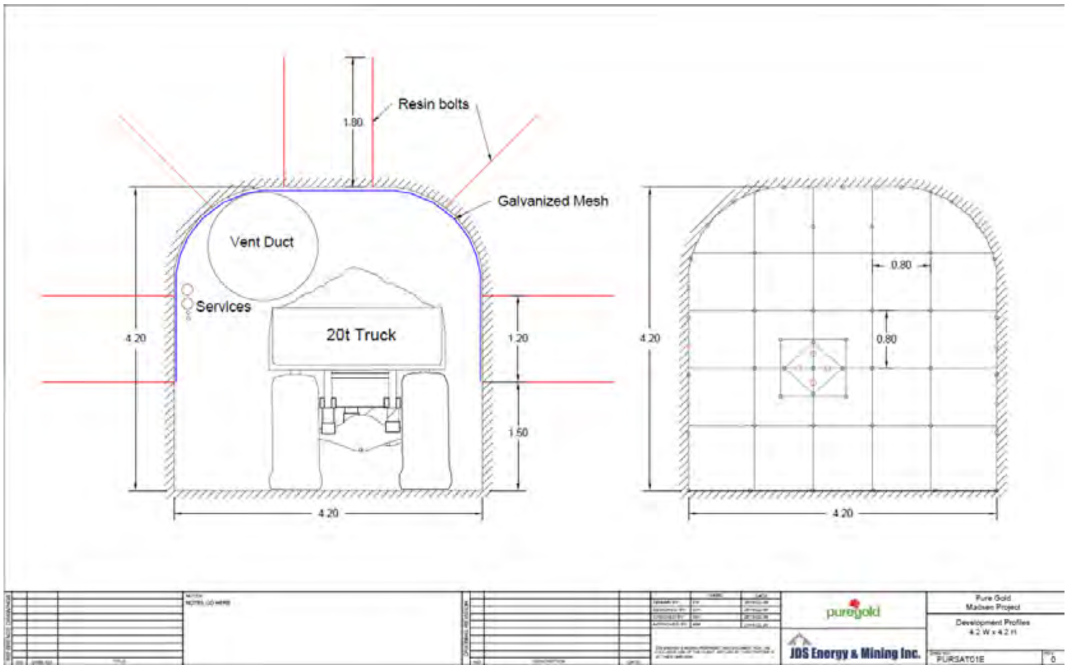
26.3.5.4 Fork Lateral Development

The Fork decline will be driven from surface (375 masl) to 170 L, a linear distance of 2,240 m. It will be driven at 4.2 m W x 4.2 m H and located in the footwall of the deposit. The decline is sized to accommodate all necessary services including ventilation, communication, power, water, and air lines. The decline connects all nine working levels of the mine and will be used for all haulage from the mine. It will also act as the mine's ventilation exhaust. The decline has been designed at a maximum gradient of 15% and a minimum centerline turning radius of 25 m. A general cross section of the mine development is shown in Figure 26-6 and Figure 26-7.

The working levels of the mine are spaced 20 m vertically, sill to sill. Each level will have a 4.2 m W x 4.2 m H footwall drive located at a minimum offset of 15 m from the deposit. Footwall drives will house the majority of services including remuck bays, ventilation raises, ancillary bays, electrical sub stations, and refuge bays.

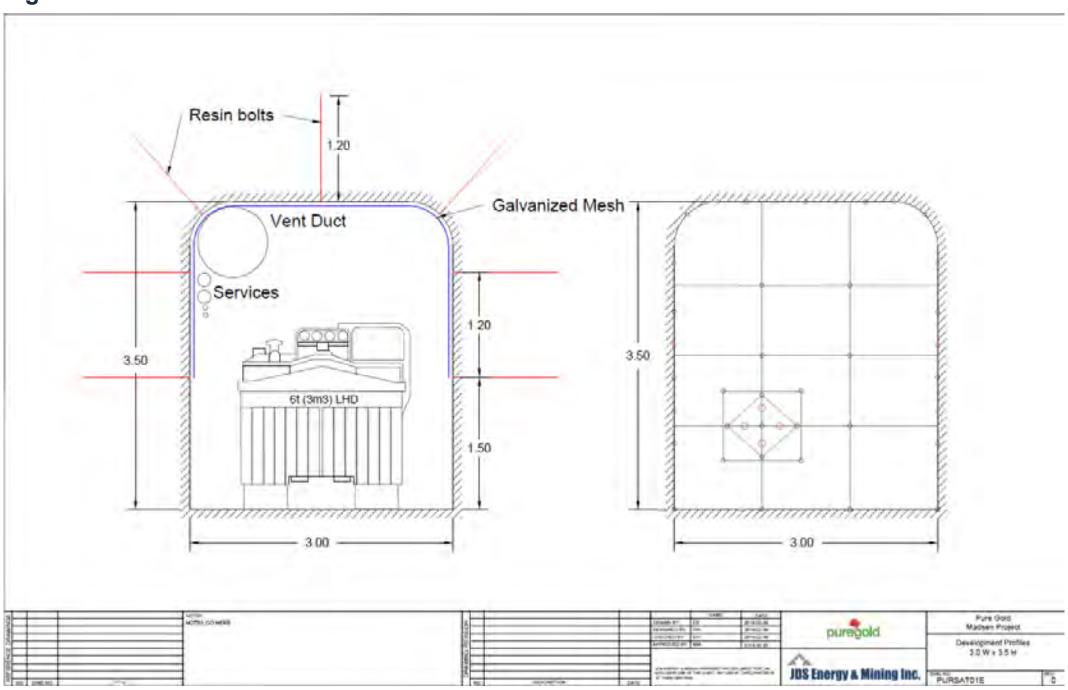
Attack drifts, 3 m W x 3.5 m H, will be driven off the footwall drive and are located strategically to access longhole drifts, 3 m W x 3.5 m H. LH drifts will be driven along the orebody at the top and bottom of the stoping blocks. LH drifts will also be driven for drilling, mineral extraction and RF placement. A plan view of a typical level is shown in Figure 26-8 and a 3D figure of planned development is shown in Figure 26-9. Development quantities are summarized in Table 26-4.

Figure 26-6: 4.2 m W x 4.2 m H Profile



Source: JDS (2019)

Figure 26-7: 3.0 m W x 3.5 m H Profile



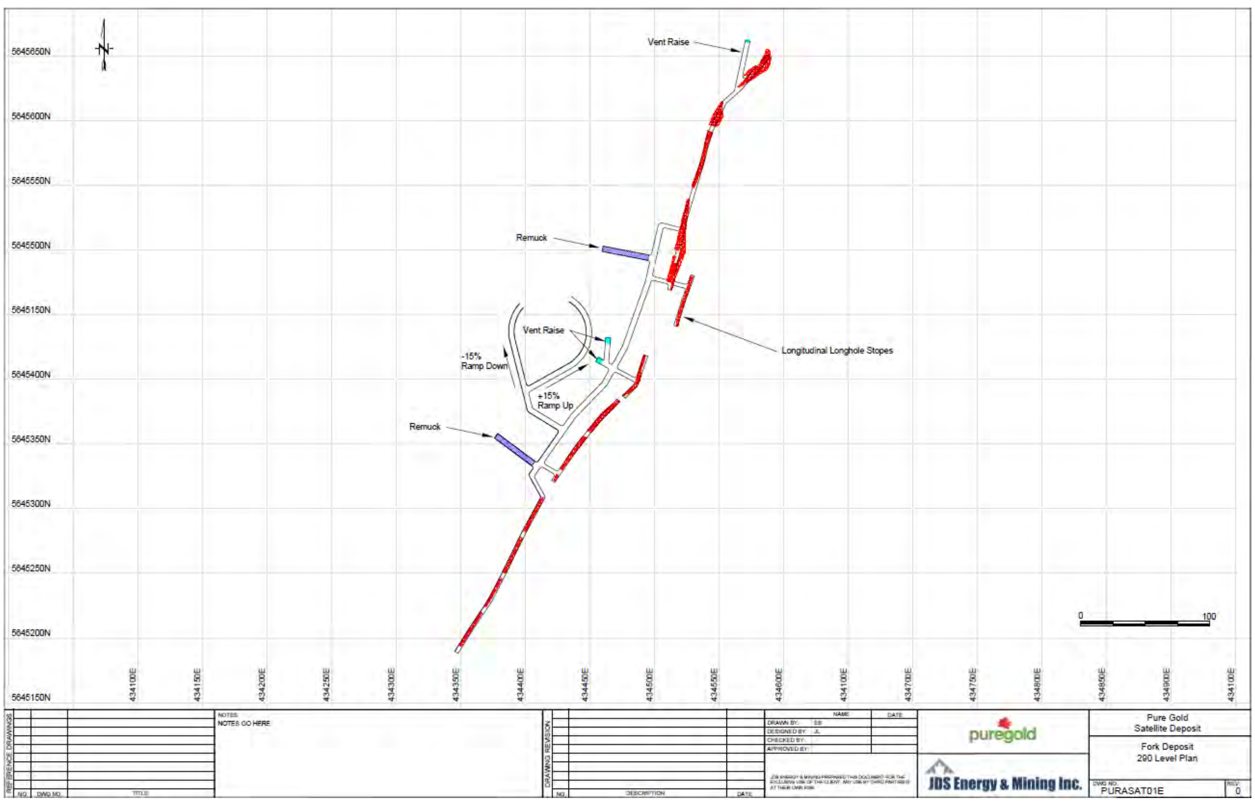
Source: JDS (2019)

Table 26-4: Fork Lateral Development Summary

Items	Units	Width	Height	Type	Total Planned
Main Ramp	m	4.2	4.2	CAPEX	2,241
Footwall Drive	m	4.2	4.2	CAPEX	1,279
Vent Drive	m	3.7	3.7	CAPEX	391
Muck Pass	m	3.7	3.7	CAPEX	107
Remuck	m	4.2	4.2	CAPEX	470
Sumps, Electrical stations, etc.	m	4.2	4.2	CAPEX	182
Attack Ramp	m	3	3.5	OPEX	1,963
Long-hole Drift	m	3	3.5	OPEX	2,655
Total	**m**				**9,288**

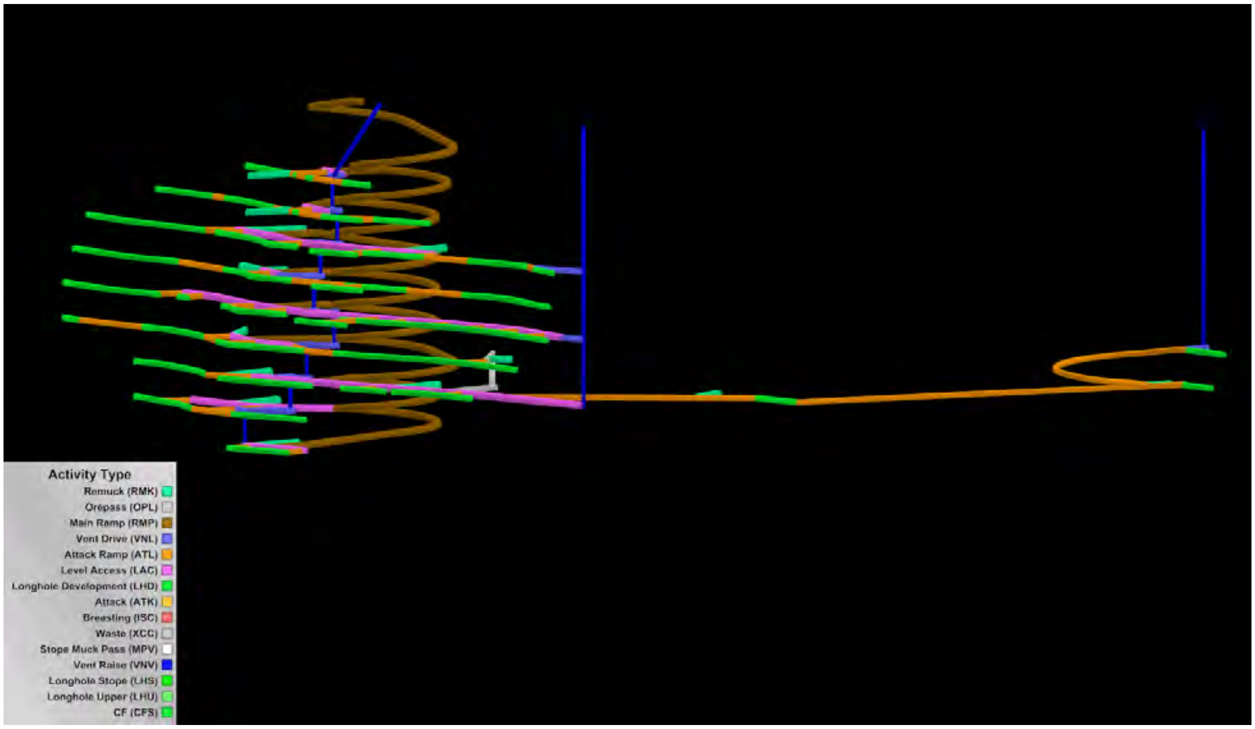
Source: JDS (2019)

Figure 26-8: Typical Fork Level Plan (290L Shown)



Source: JDS (2019)

Figure 26-9: Fork Development - Oblique View



Source: JDS (2019)

26.3.5.5 Wedge Lateral Development

The Wedge decline will be driven from surface (395 masl) to -40 L, a linear distance of 4,820 m. Wedge development quantities are summarized in Table 26-5. It will be driven at 4.2 m W x 4.2 m H and located in the footwall of the deposit. The decline is sized to accommodate all necessary services including ventilation, communication, power, water, and air lines. The decline connects all 22 working levels of the mine and will be used for all haulage from the mine. It will also act as the mine’s exhaust. The decline has been designed at a maximum gradient of 15% and a minimum turning radius of 25 m on centre. A general cross section of the mine development is shown in Figure 26-6 and Figure 26-7.

The working levels of the mine are spaced 20 m vertically, sill to sill. Each level will have a 4.2 m W x 4.2 m H footwall drive located at a minimum offset of 15 m from the deposit. Footwall drives will house the majority of services including remucks bays, ventilation raises, ancillary bays, electrical sub stations, and refuge bays.

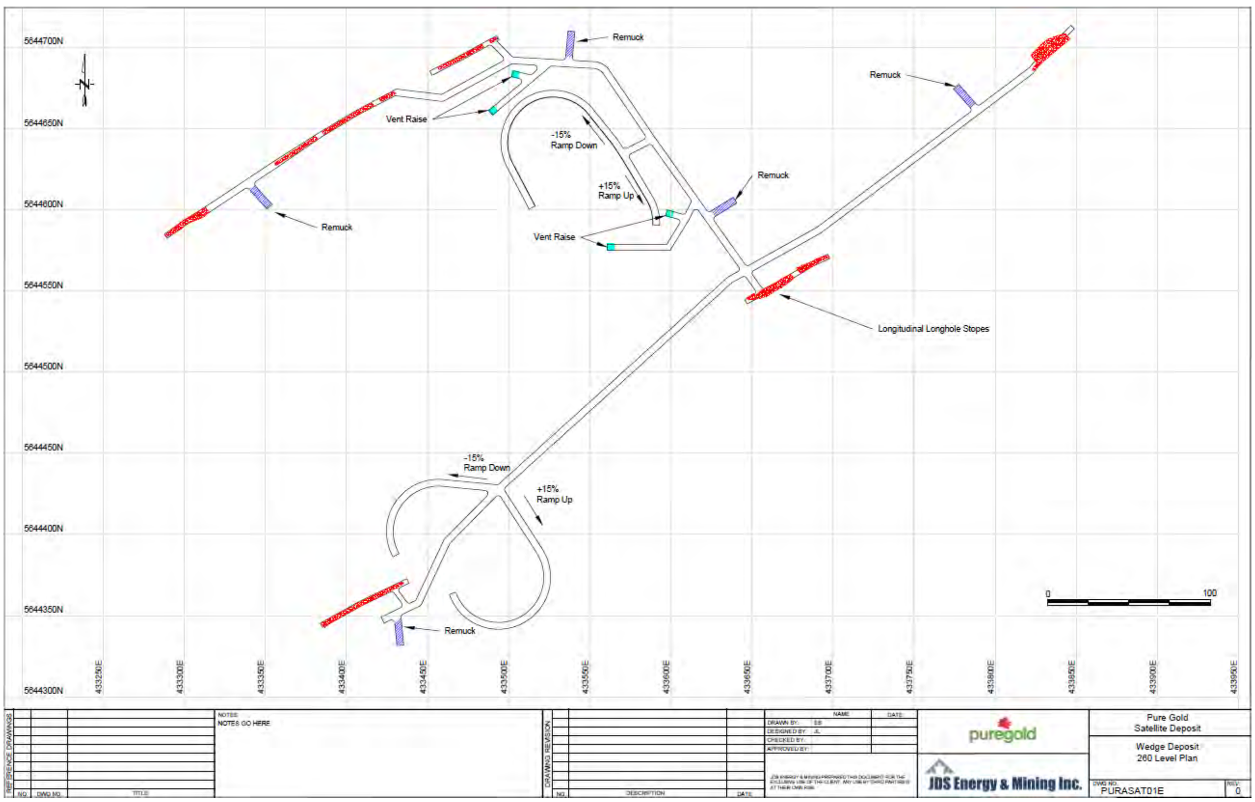
Attack drifts, 3 m W x 3.5 m H, will be driven off the footwall drive and are located strategically to access LH drifts, 3 m W x 3.5 m H. LH drifts are driven along the orebody at the top and bottom of the stoping blocks. A plan view of a typical level is shown in Figure 26-10, with a 3D view in Figure 26-11.

Table 26-5: Wedge Lateral Development Summary

Items	Units	Width	Height	Type	Total Planned
Main Ramp	m	4.2	4.2	CAPEX	4,820
Footwall Drive	m	4.2	4.2	CAPEX	4,628
Vent Drive	m	3.7	3.7	CAPEX	2,071
Remuck	m	4.2	4.2	CAPEX	670
Sumps, Electrical stations, etc.	m	4.2	4.2	CAPEX	405
Attack Ramp	m	3	3.5	OPEX	730
Long-hole Drift	m	3	3.5	OPEX	3,243
Total	**m**				**16,566**

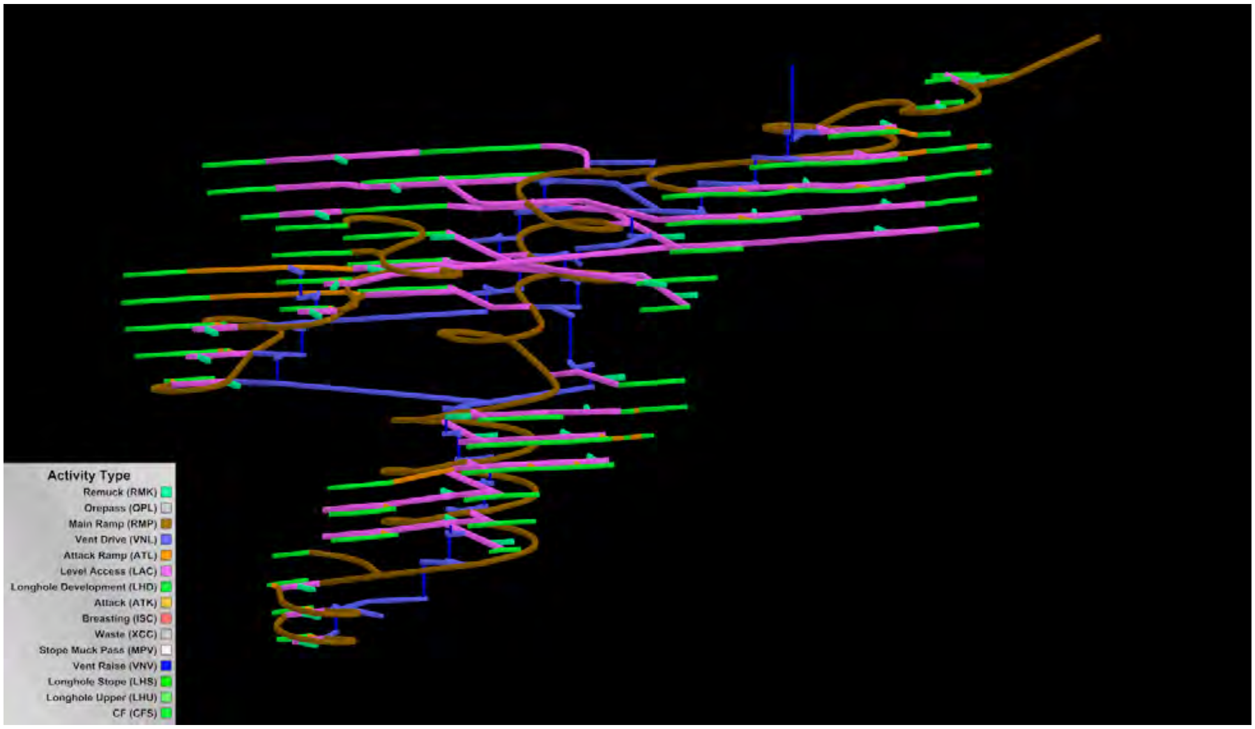
Source: JDS (2019)

Figure 26-10: Typical Wedge Level Plan (260L Shown)



Source: JDS (2019)

Figure 26-11: Wedge Development - Oblique View



Source: JDS (2019)

26.3.5.6 Russet South Lateral Development

The Russet South decline will be driven from surface (384 masl) to 202 L, a linear distance of 2,010 m. It will be driven at 4.2 m W x 4.2 m H and located in the footwall of the deposit. A development summary for Russet South is shown in Table 26-6.

The working levels of the mine are spaced approximately 13 m vertically, sill to sill. Each level will have a 4.2 m W x 4.2 m H footwall drive located at a minimum offset of 15 m from the deposit. Footwall drives will house the majority of services including remucks bays, ventilation raises, ancillary bays, electrical sub stations, and refuge bays.

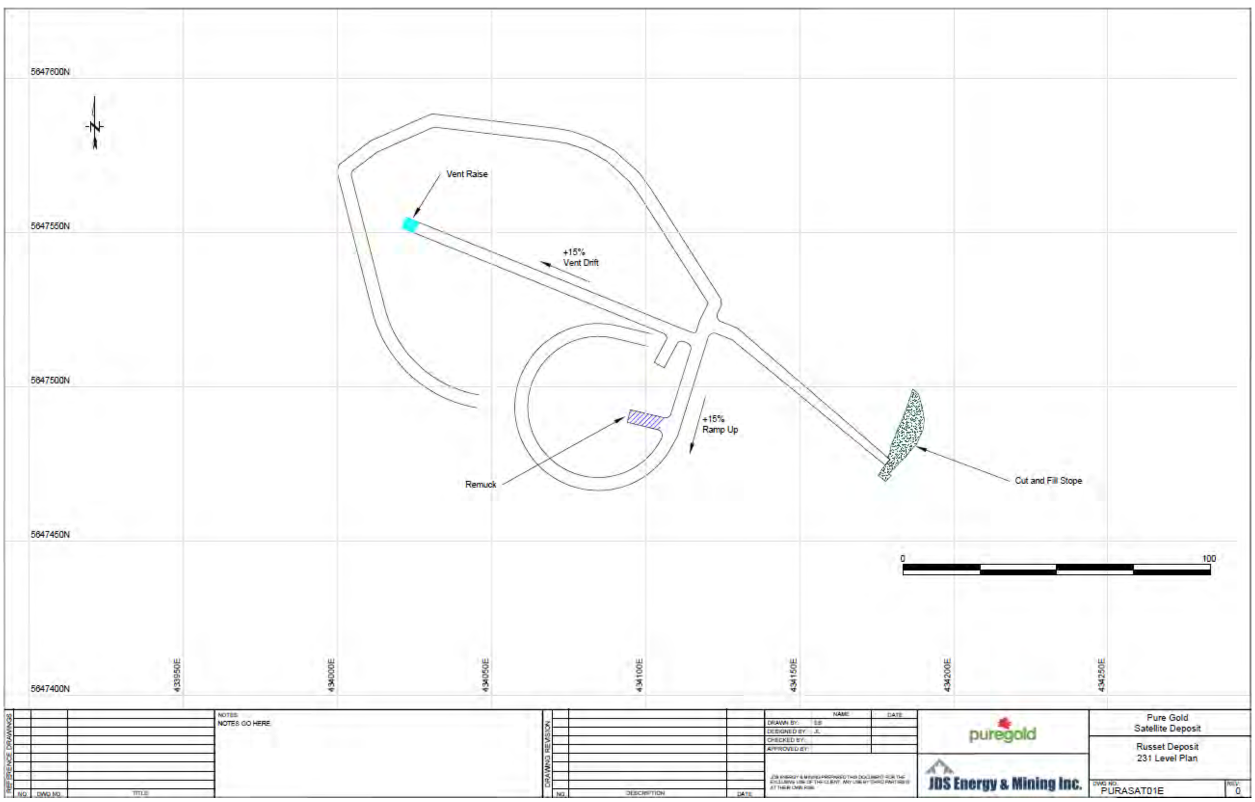
Off the footwall drive, an attack ramp drift will be driven, 2 m W x 2.4 m H. The attack ramp will be driven at -15% to access the MCF drives. A plan view of a typical level is shown in Figure 26-12 and a 3D view of development is shown in Figure 26-13.

Table 26-6: Russet South Lateral Development Summary

Items	Units	Width	Height	Type	Total Planned
Main Ramp	m	4.2	4.2	CAPEX	2,005
Footwall Drive	m	4.2	4.2	CAPEX	690
Vent Drive	m	3.7	3.7	CAPEX	354
Remuck	m	4.2	4.2	CAPEX	210
Sumps, Electrical stations, etc.	m	4.2	4.2	CAPEX	140
Un-mineralized Drift	m	2	2.4	OPEX	324
Breasting	m	2	2.4	OPEX	1,766
Attack Ramp	m	2	2.4	OPEX	830
Cut and Fill	m	4.1	2.4	OPEX	5,060
Total	**m**				**11,380**

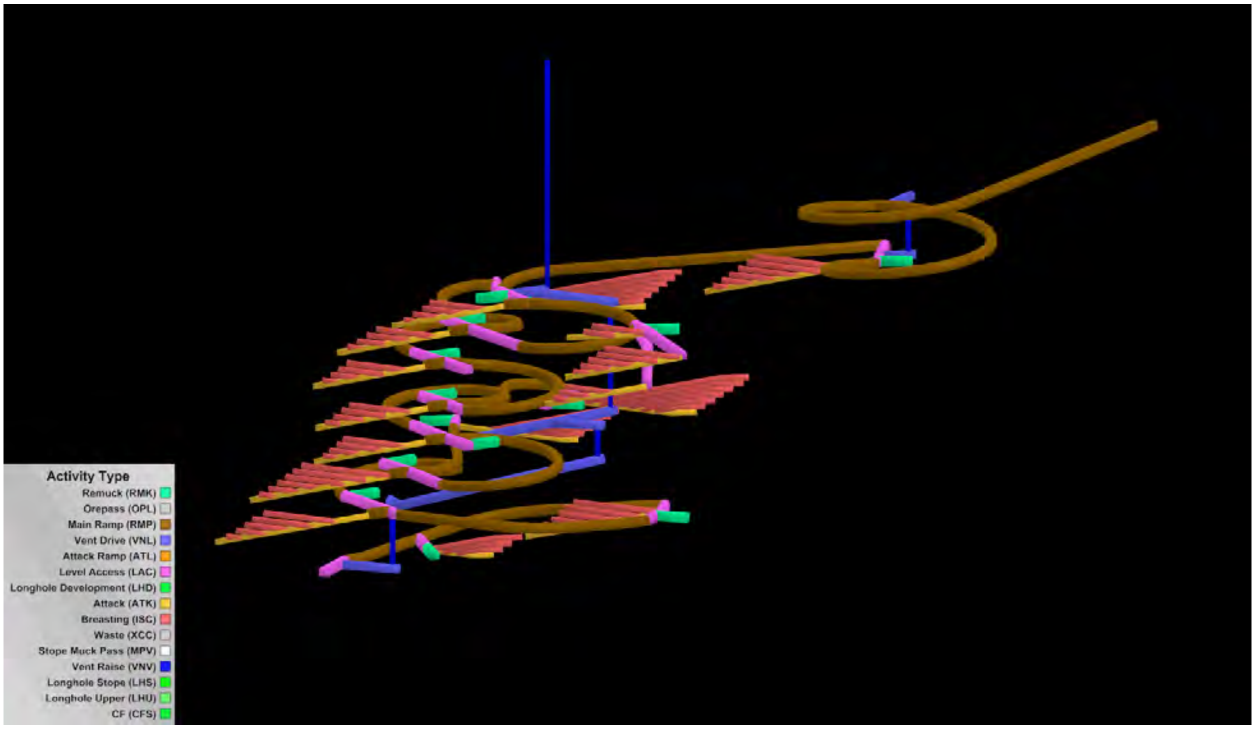
Source: JDS (2019)

Figure 26-12: Typical Russet South Level Plan (231L Shown)



Source: JDS (2019)

Figure 26-13: Russet South Development - Oblique View



Source: JDS (2019)

26.3.5.7 Fork Vertical Development

Vertical development includes one production pass, three ventilation raises (one fresh air and two exhaust), and multiple drop raises to develop a fresh air system and secondary escape way.

A 4 m L x 4 m W production pass will be driven from 230L to 210L using a LH drop raise.

The Fork fresh air raise (FFAR) will be driven from surface to 330L by 2.4 m diameter raise bore and will act as an as the primary fresh air raise and escape way. The raise bore will connect to a series of drop raises via ventilation drifts to enable fresh air to the bottom of the mine and a secondary escape way. Drop raises will be driven at 4 m L x 4 m W, in 20 m vertical segments from 170L to 330L.

The Fork return air raise 1 (FRAR1) will be driven from surface to 210 L by 2.4 m diameter raise bore and will act as an exhaust raise and secondary escape way.

The Fork return air raise 2 (FRAR2) will be driven from surface to 270 L by 2.4 m diameter raise bore and will act as an exhaust raise and secondary escape way.

A summary of vertical development is show in Table 26-7.

Table 26-7: Fork Vertical Development Summary

Items	Length (m)	Inclination (degrees)	Number of Sections	Method	Diameter / Width (m)	Length (m)
Fork Fresh Air Raise (FFAR)	67	80	1	Raise Bore	2.4	
Fork Return Air Raise 1 (FRAR1)	168	90	1	Raise Bore	2.4	
Fork Return Air Raise 2 (FRAR2)	131	90	1	Raise Bore	2.4	
Muck Pass	20	90	1	Drop Raise	4	4
Drop Raise	160	90	9	Drop Raise	4	4

Source: JDS (2019)

26.3.5.8 Wedge Vertical Development

Vertical development will be used to provide a ventilation raise, and a network of drop raises to develop a fresh air system and secondary escape way.

The Wedge fresh air raise (WFAR) will be driven from surface to 340L by 2.4 m diameter raise bore and will act as an as the primary fresh air raise and escape way. Drop raises will be driven at 4 m L x 4 m W, in approximately 20 m vertical segments from 340L to -40L.

A summary of vertical development is show in Table 26-8.

Table 26-8: Wedge Vertical Development Summary

Items	Length (m)	Inclination (degrees)	Number of Sections	Method	Diameter / Width (m)	Length (m)
Wedge Fresh Air Raise (WFAR)	60	90	1	Raise Bore	2.4	
Drop Raise	562	90	27	Drop Raise	4	4

Source: JDS (2019)

26.3.5.9 Russet South Vertical Development

The Russet South fresh air raise (RFAR) will be driven from surface to 305L by 2.4 m diameter raise bore and will act as an as the primary fresh air raise and escape way. The raise bore will connect to a series of drop raises via ventilation drifts to enable fresh air to the bottom of the mine and a secondary escape way. Drop raises will be driven at 4 m L x 4 m W, in approximately 20 m vertical segments from 305L to 203L.

A summary of vertical development is show in Table 26-9.

Table 26-9: Russet South Vertical Development Summary

Items	Length (m)	Inclination (degrees)	Number of Sections	Method	Diameter / Width (m)	Length (m)
Russet South Fresh Air Raise (RFAR)	100	90	1	Raise Bore	2.4	
Drop Raise	116	90	4	Drop Raise	4	4

Source: JDS (2019)

26.3.6 Mining Method

26.3.6.1 Fork Mining Method

The Fork deposit will be mined using longitudinal LH mining.

Stopes sills will be driven at 3 m W x 3.5 m H in 20 m vertical increments. Stopes will in general be a maximum of 30 m long (along strike). This produces a typical maximum exposed hanging wall and footwall of 30 m L x 23.5 m H. When the deposit has a strike greater than 30 m, a 5 m pillar will separate the stopes.

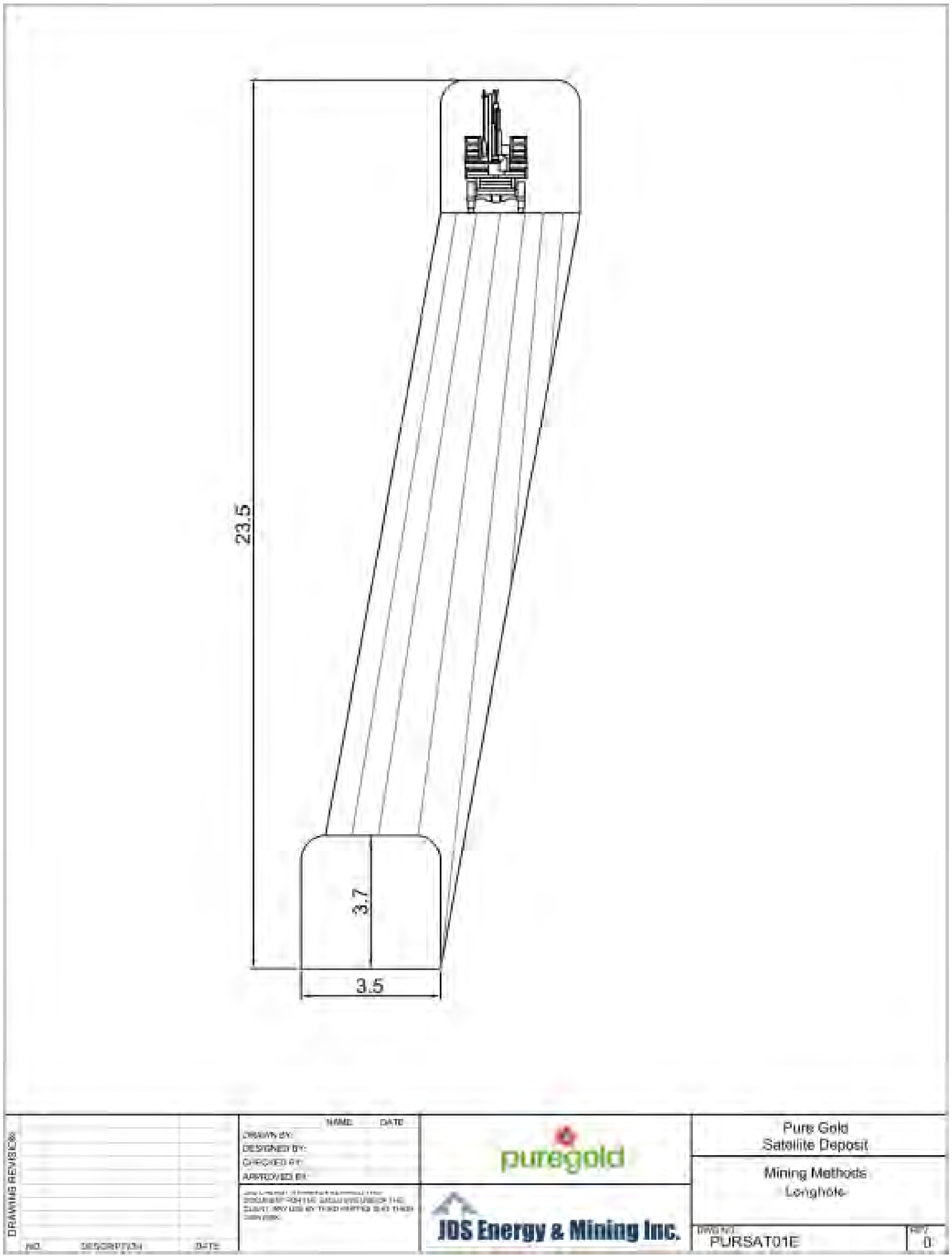
Typical production drilling will be completed using 89 mm diameter down holes, using drop raises, slots, and drill rings for stope development and mining. LHDs will be used to muck the stopes from the extraction sill. A typical ring pattern for longitudinal long-hole mining methods is show in Figure 26-14.

In situations where the final upper most stopes of the panel are at the top level of the mineralized block, stopes will be extracted using uphole drilling and inverse raises.

A mineralized block can span up to 6 mining levels. Stopes will be limited to mining one level, 20 metres sill to sill. After a stope has been extracted, it will be filled with RF, enabling mining of the following stopes. No RF will be placed in mined uphole stopes.

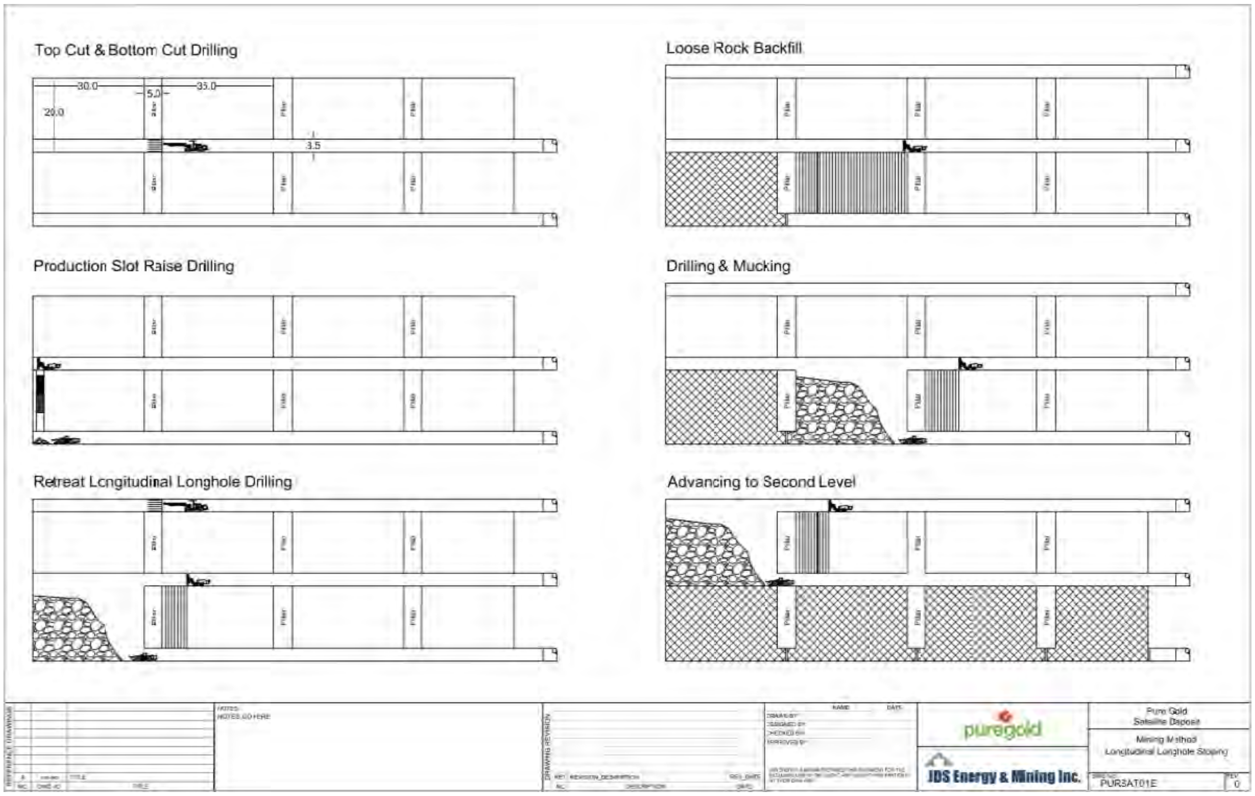
A general cross section of the longitudinal long-hole mining is show in Figure 26-15.

Figure 26-14: General Long-hole Cross Section



Source: JDS (2019)

Figure 26-15: Long Section Showing Mining Sequence for a Typical Block of Stopes



Source: JDS (2019)

26.3.6.2 Wedge Mining Method

The Wedge deposit will be mined using longitudinal LH mining in a similar manner as the Fork deposit with the same drift & stope sizes, sub-level interval and drill hole diameter.

A general cross section of the longitudinal long-hole mining is show in Figure 26-15.

26.3.6.3 Russet South Mining Method

The Russet South deposit will be mined using MCF with RF.

Attack drifts will be driven at 2.0 m W x 2.4 m H off the footwall drive and can access up to seven MCF cuts. Drifts will be driven at average width of 4.1 m W x 2.4 m H with a shanty back and a strike length to up 120 m.

MCF production will be done using single boom jumbos and 0.7 cubic meter capacity LHDs. Once a cut has been extracted, RF will fill the void before mining of the next level.

26.3.7 Mineral Inventory

Mining factors were applied for the selected mining methods and stope geometries to represent the anticipated dilution and recoveries throughout each deposit. Additional details on dilution estimates are described in Section 26.3.3.

26.3.7.1 Fork Dilution

Hangingwall and footwall dilution were calculated on the assumption of 0.5 m over break on the hanging wall and 0.3 m on the footwall. Backfill dilution was estimated at 0.2 m on the floor. The overall dilution is estimated at 34% and was assigned zero grade.

26.3.7.2 Wedge Dilution

Hanging and footwall dilution were calculated on the assumption of 0.5 m over break on the hanging wall and 0.3 m on the footwall. Backfill dilution was estimate at 0.2 m on the floor. The overall dilution is estimated at 27% with zero grade.

26.3.7.3 Russet South Dilution

Mining dilution was estimated at 5% plus 0.2 m backfill dilution on the floor.

26.3.7.4 Recovery

For each deposit, a 95% mining recovery was applied.

26.3.7.5 Mining Inventory

The total mining inventory by deposit is summarized in Table 26-10. The results are based upon preliminary mineable stope designs and incorporate the factors for recovery and dilution noted in Sections 26.3.7.1 to 26.3.7.4.

Table 26-10: Fork, Russet South, and Wedge Deposit Mining Inventory

Zone	Tonnes (kt)	Au (g/t)
Indicated	619	6.93
Fork	*200*	*4.63*
Wedge	*292*	*8.05*
Russet South	*127*	*7.95*
Inferred	444	5.67
Fork	*157*	*4.32*
Wedge	*239*	*6.39*
Russet South	*48*	*6.50*
Total Mine Plan	**1,063**	**6.40**

Source: JDS (2019)

26.3.7.6 Material Handling

26.3.7.6.1 Mineralized Material

Mucking will be carried out using 6.0 t and/or 1.3 t LHDs with remote tramming capabilities. The mineralized material will be trammed to a re-muck where it will be loaded into 20 t haul trucks. Loaded trucks will haul the material to surface up the decline. Once on surface at the individual portals, the mineralized material will be stockpiled. A fleet of highway dump trucks, loaded by a front-end loader, will transport the material from the portal stockpile to the Madsen processing plant.

26.3.7.6.2 Un-mineralized Development Rock

Un-mineralized development rock will be mucked using a using 6.0 t LHDs to a re-muck, then loaded onto 20 t haul trucks. Un-mineralized rock will be hauled to a stope void when possible. If no void is available, the rock will be hauled to surface and stockpiled at designated areas near each portal. When an empty stope requires more rockfill then what is produced from development, rock will be back hauled from surface to the desired level and dumped into a re-muck. From the re-muck, a 6.0 t and/or 1.3 t LHD will tram the material into the stope void. Excess un-mineralized mine rock will remain at the portal mine rock storage areas.

26.3.7.7 Mine Services

26.3.7.7.1 Fork Ventilation

Ventilation for the Fork mine will be managed through a series of internal raises, the surface portal and three raises collared on surface. Fan locations and loads are shown in Table 26-11. Fresh air will be forced down the FFAR and exhausted out the decline, or one of the two return air raises (FRAR1 and FRAR2). Air flow will be provided by an axial fan located at the FFAR.

The main fan is sized to accommodate peak airflow of 60 m³/s. The FFAR will provide fresh air to all the active levels of the mine. The return air from each level will exhaust through the decline, FRAR1 or FRAR2. The ventilation schematic during production is shown in Figure 26-16.

FFAR will comprise of a 2.4 m raise bore from surface to 330L. FFAR from 330L to 170L will comprise of a network of 4 m W x 4 m L drop raises and connected via ventilation drifts. RAR1 and RAR2 will be raise bored to a diameter of 2.4 m from surface to 210L and 270L respectively.

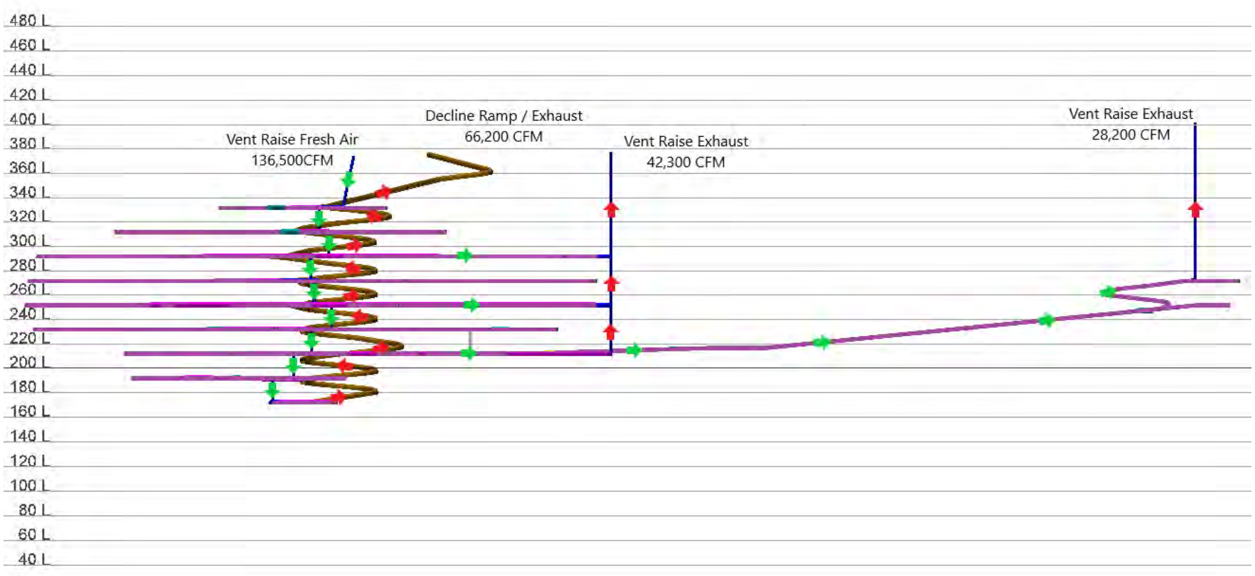
A minimum airflow of 5 m³/s will be provided to each level through regulators installed in FFAR access drifts. Smaller 56 kW auxiliary fans will be employed to provide fresh air to the face of each stope, capable of providing 30 m³/s of air flow through 1.1 m diameter collapsible ducts.

Table 26-11: Fork Fan Location and Duty Points

Fan	Location	Airflow (m³/s)	Pressure (kPA)	Fan Power (kW)
Main Intake	FFAR Collar	60	0.45	51
Level Development	Each Level	30	3	45
Stope Fan	Stope	30	3	45

Note: Fan power is calculated at an air density of 1.14 kg/m³
Source: JDS (2019)

Figure 26-16: Fork Mine Ventilation Section



Source: JDS (2019)

26.3.7.7.2 Wedge Ventilation

Ventilation for the Wedge mine will be managed through a series of internal raises, the surface portal and one raise collared on surface. Fan locations and loads are shown in Table 26-12. Fresh air will be forced down the WFAR and exhausted out the decline. Air flow will be provided by an axial fan located at the WFAR.

The main fan is sized to accommodate peak airflow of 92 m³/s. The WFAR will provide fresh air to all the active levels in the mine. The return air from each level will exhaust through the decline. A ventilation schematic during production is shown in Figure 26-17.

WFAR will comprise of a 2.4 m raise bore from surface to 340L. The WFAR from 340L to -40L will comprise of a network of 4 m W x 4 m L drop raises connected via ventilation drifts.

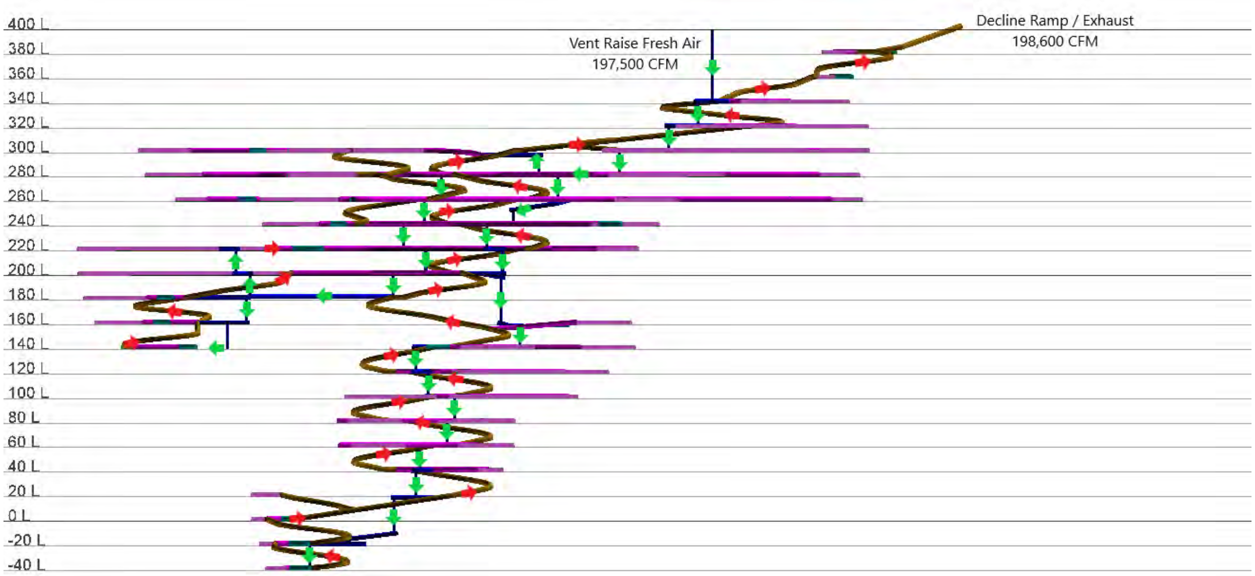
Fresh air will only be provided to production levels and controlled by regulators installed in WFAR access drifts. Smaller 56 kW auxiliary fans will be employed to provide fresh air to the face of each stope, capable of providing 30 m³/s of air flow through 1.1 m diameter collapsible ducts.

Table 26-12: Wedge Fan Location and Duty Points

Fan	Location	Airflow (m³/s)	Pressure (kPA)	Fan Power (kW)
Main Intake	WFAR Collar	92	1.2	149
Level Development	Each Level	30	3	45
Stope Fan	Stope	30	3	45

Note: Fan power is calculated at an air density of 1.14 km/m^3
Source: JDS (2019)

Figure 26-17: Wedge Mine Ventilation Section



Source: JDS (2019)

26.3.7.7.3 Russet South Ventilation

Ventilation for the Russet South mine will be managed through a series of internal raises, the surface portal and one raise collared on surface. Fan locations and loads are shown in Table 26-13. Fresh air will be forced down the RFAR and exhausted out the decline. Air flow will be provided by an axial fan located at the RFAR.

The main fan is sized to accommodate peak airflow of 60 m³/s. The RFAR will provide fresh air to all the active levels in the mine. The return air from each level will exhaust through the decline. A ventilation schematic during production is shown in Figure 26-18.

The RFAR will comprise of a 2.4 m raise bore from surface to 305L. The RFAR from 305L to 203L will comprise of a network of 4 m W x 4 m L drop raises connected via ventilation drifts.

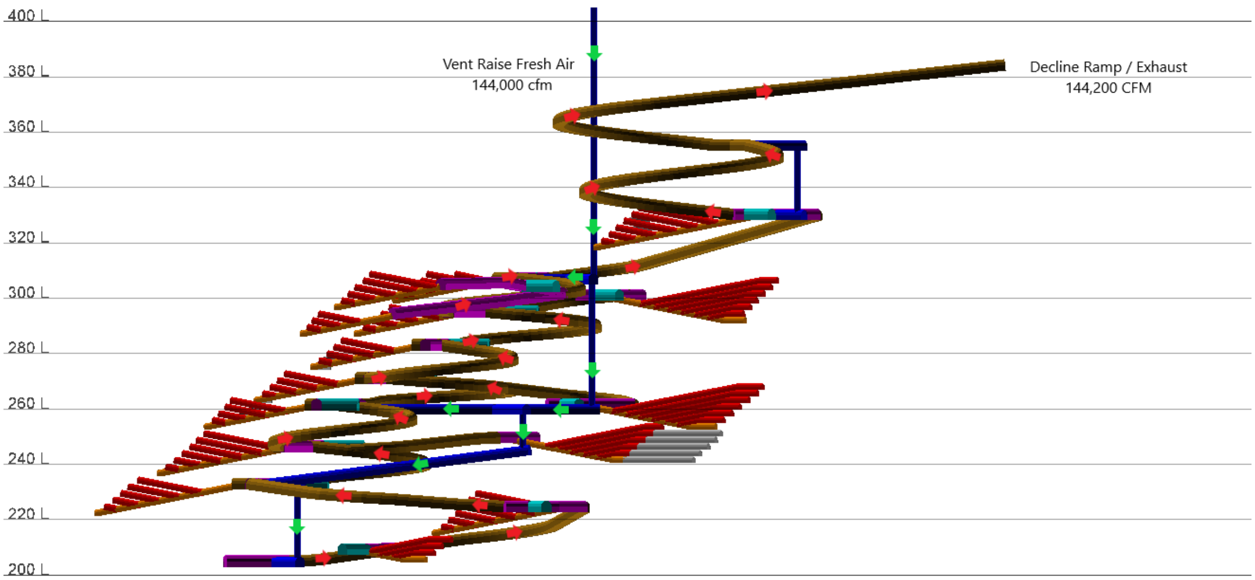
Fresh air will only be provided to production levels and controlled by regulators installed in RFAR access drifts. Smaller 56 kW and 30 kW auxiliary fans will be employed to provide fresh air to the face of each stope, providing 30 m³/s and 20 m³/s of air flow respectively, through 1.1 m diameter collapsible ducts.

Table 26-13: Russet South Fan Location and Duty Points

Fan	Location	Airflow (m³/s)	Pressure (kPA)	Fan Power (kW)
Main Intake	WFAR Collar	60	0.82	61
Level Development	Each Level	30	3	45
Stope Fan	Stope	20	3	24

Note: Fan power is calculated at an air density of 1.14 km/m^3
Source: JDS (2019)

Figure 26-18: Russet South Mine Ventilation Section



Source: JDS (2019)

26.3.8 Mine Dewatering

Dewatering activities at each mine will consist of a series of strategically placed water collection sumps and decant sumps from active working faces. The underground water will be pumped to the portal and stored in pressurized water tanks.

The underground dewatering system has been sized based on groundwater inflow information. In a 2018 study completed by Lorax, it was estimated that the Madsen Mine would have a steady state ground water inflow of 800 m³/day based on the amount exposed rock surface area. Groundwater inflow for each of the Fork, Russet South, and Wedge mines was determined by using the 800 m³/day and multiplying it by the ratio of exposed rock to the Madsen Mine. This resulted in groundwater inflows of 14 m³/day, 23 m³/day, and 9 m³/day respectively for the Fork, Russet South, and Wedge deposits.

For each mine, pumping will be conducted using a series of electric submersible pumps. The size, quantities and power requirements for each sump by mine is summarized in Table 26-14 to Table 26-16. The pump locations for each deposit are shown in Figure 26-19 to Figure 26-21.

Table 26-14: Fork Mine Pumping Summary

Primary Pumping Station	Level	Equivalent Pump	Discharge (inches)	# of Pumps	Power (kW)
Primary Collection Sump 1	170	LH8110	4	1	112
Primary Collection Sump 2	310	LH8110	4	1	112

Source: JDS (2019)

Table 26-15: Wedge Mine Pumping Summary

Primary Pumping Station	Level	Equivalent Pump	Discharge (inches)	# of Pumps	Power (kW)
Primary Collection Sump 1	-40	LH6110	4	1	112
Primary Collection Sump 2	140	LH6110	4	1	112
Primary Collection Sump 3	140	LH6110	4	1	112
Primary Collection Sump 4	280	LH6110	4	1	112

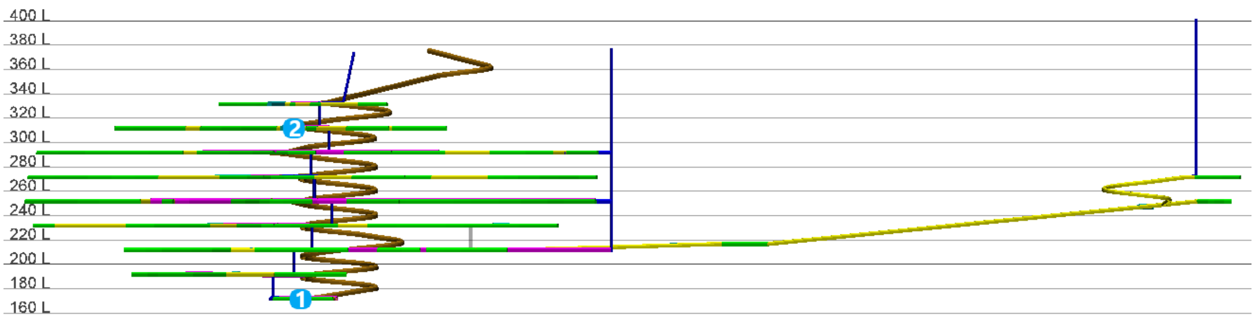
Source: JDS (2019)

Table 26-16: Russet South Mine Pumping Summary

Primary Pumping Station	Level	Equivalent Pump	Discharge (inches)	# of Pumps	Power (kW)
Primary Collection Sump 1	202	LH8110	4	1	112
Primary Collection Sump 2	305	LH8110	4	1	12

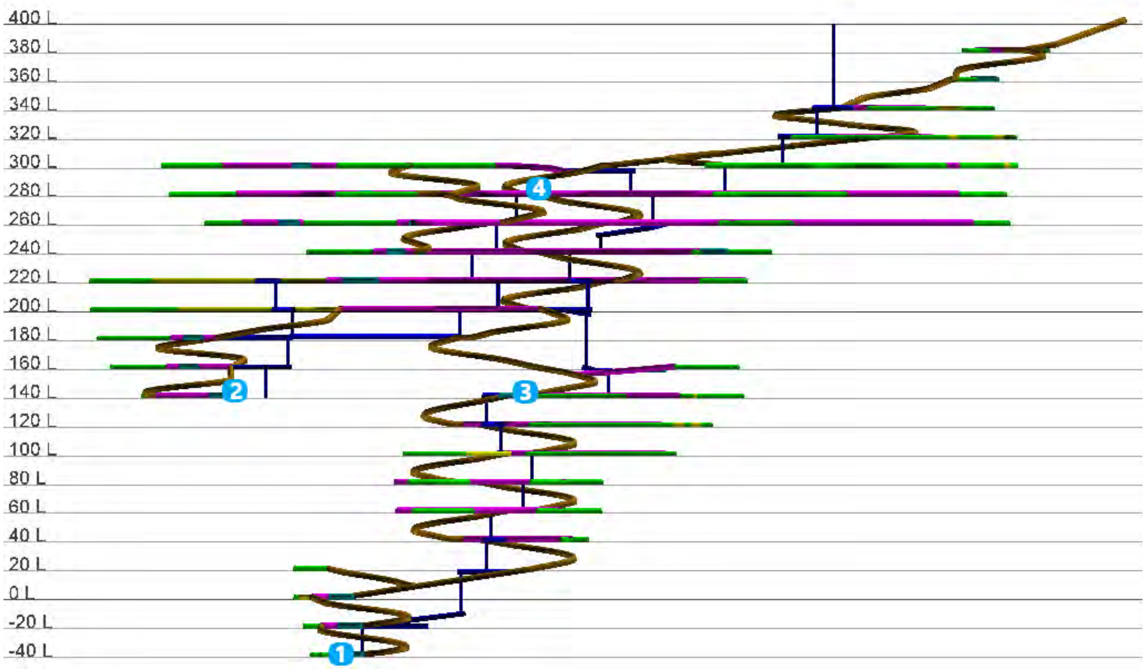
Source: JDS (2019)

Figure 26-19: Fork Mine Pumping Locations



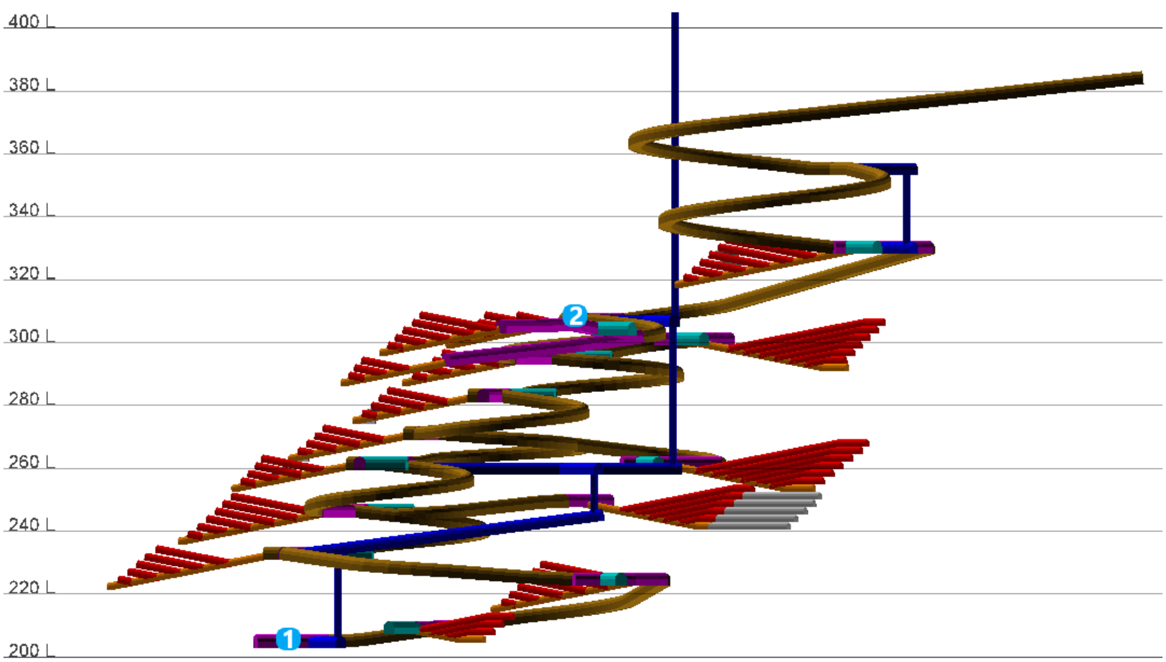
Source: JDS (2019)

Figure 26-20: Wedge Mining Pumping Locations



Source: JDS (2019)

Figure 26-21: Russet South Mine Pumping Locations



Source: JDS (2019)

26.3.9 Compressed Air and Water Supply

Compressed air will be supplied throughout the mine. The air will be provided from a surface compressor located at each of the Fork, Russet South, and Wedge Portals.

Water use for underground operations will be drawn from collected mine inflow and supplemented by nearby surface sources. The Russet South Mine will draw water from Russet Lake and the Fork and Wedge Mines will draw water from Flat Lake. Water collected from mine in-flow is expected to provide 70% of the operational water requirements, this includes a 50% water recycle rate. Water from the lakes is expected to provide the remaining 30% of the operational water requirements. Each mine will store water in pressurized water tanks located just inside the portal.

26.3.10 Electrical Distribution

Power will be supplied to each mine from the Madsen's power station at 4160 V via overhead lines to a step-down transformer located at the respective portals. For each mine, power will be distributed from the portal step-down transformer using 600 V service lines and bore-holes between levels.

26.3.11 Mine Personnel

The mine will require a full-time work force of mining, maintenance, services, technical and administrative personnel. Each mine will operate 365 d/a, 24, h/d, primarily through two – 12 hour shifts, allowing one hour for smoke clearing between shifts. Mine operations will consist of two different rotations:

- Four days on / four days off (4x4): Mine Operations, Maintenance, Construction Labour, Site Services (12 hour shifts); and
- Five days on / two days off (5x2): Engineering, Administrative, and Management (day shift only, 8 hour days).

During peak production the three mines will require a combined 217 people on site, including those on 5x2 rotations.

Staffing will be ramped up to full production requirements during the first year of operations. Certain production related positions are not expected to be necessary during preliminary mine development and construction such as ore control geologists, survey helpers, geotechnical technicians / samplers, Ventilation / Project Engineer.

Similarly, positions were reduced or eliminated during the winding down period of the final year of operation.

A break-down of the on-site the personnel requirements for ramp-up and production is presented in Table 26-17.

Table 26-17: On-site Personnel, Mine Operations

Position	Average	Pre-Prod	Production	Peak	Year 4
Mine General	19	19	19	19	19
Drill and Blast	35	31	36	44	32
Fork	*12*	*10*	*12*	*15*	*11*
Wedge	*12*	*10*	*12*	*15*	*11*
Russet South	*11*	*10*	*11*	*14*	*10*
Load and Haul	38	24	42	44	40
Fork	*13*	*8*	*14*	*15*	*14*
Wedge	*13*	*8*	*14*	*15*	*14*
Russet South	*12*	*8*	*13*	*14*	*12*
Support Services	49	56	47	56	28
Fork	*17*	*19*	*16*	*19*	*10*
Wedge	*17*	*19*	*16*	*19*	*10*
Russet South	*16*	*18*	*15*	*18*	*8*
Mine Maintenance	32	29	33	33	33
Fork	*11*	*10*	*11*	*11*	*11*
Wedge	*11*	*10*	*11*	*11*	*11*
Russet South	*11*	*9*	*11*	*11*	*11*
Technical Services	17	7	20	21	13
Total	190	166	197	217	165

Source: JDS (2019)

26.3.12 Mine Equipment

26.3.12.1 Mobile Equipment

Both diesel and electric equipment will be employed throughout the mines. The primary haulage fleet will consist of 20 t capacity articulated haul trucks and 6.0 t capacity LHDs. Development drilling will be conducted using two-boom jumbos. Long-hole drilling will be conducted using a Simba type long-hole. MCF mining will be conducted using a one-boom jumbo and mucked with a 1.3 t LHD.

Mineralized rock will be hauled to surface and stockpiled at its respective portal. On surface, the mineralized rock will be loaded by a front-end loader onto highway dump trucks and transported to the Madsen processing plant.

Equipment requirements were developed from first principles, based on the maximum annual duty hours for an individual piece of equipment, modified for mechanical availability and project utilization.

A list of the underground production and support equipment along with their respective factors used in the mine plan are shown in Table 26-18.

Table 26-18: Mine Mobile Equipment Summary

Equipment	Avg.	Peak	Average LOM Utilization (%)	Y-1	Y1	Y2	Y3	Y4
Jumbo 1-Boom	2	2	69%	0	2	2	2	1
Russet South	*2*	*2*	*69%*	*0*	*2*	*2*	*1*	*1*
Jumbo 2-Boom	3	4	45%	4	4	4	3	2
Fork	*2*	*2*	*72%*	*1*	*2*	*2*	*1*	*0*
Wedge	*2*	*3*	*66%*	*2*	*2*	*3*	*1*	*1*
Russet South	*1*	*2*	*22%*	*2*	*1*	*1*	*1*	*1*
LH Drilling	3	3	25%	3	3	3	3	3
Fork	*1*	*1*	*27%*	*1*	*1*	*1*	*1*	*1*
Wedge	*1*	*1*	*44%*	*1*	*1*	*1*	*1*	*1*
Russet South	*1*	*1*	*21%*	*1*	*1*	*1*	*1*	*1*
Bolting	3	3	30%	3	3	3	3	2
Fork	*1*	*1*	*44%*	*1*	*1*	*1*	*1*	*0*
Wedge	*1*	*2*	*51%*	*1*	*1*	*2*	*1*	*1*
Russet South	*1*	*1*	*12%*	*1*	*1*	*1*	*1*	*1*
LHD 6.0t	4	4	56%	3	4	4	4	4
Fork	*2*	*2*	*65%*	*1*	*2*	*1*	*2*	*2*
Wedge	*2*	*2*	*68%*	*1*	*2*	*2*	*2*	*2*
Russet South	*1*	*1*	*21%*	*1*	*1*	*1*	*1*	*1*
LHD 1.3t	2	2	68%	0	2	2	2	1
Russet South	*2*	*2*	*68%*	*0*	*2*	*2*	*2*	*1*
Trucking 20t	5	6	73%	3	6	6	6	5
Fork	*2*	*3*	*76%*	*1*	*3*	*2*	*3*	*3*
Wedge	*3*	*4*	*77%*	*1*	*4*	*4*	*4*	*3*
Russet South	*1*	*1*	*57%*	*1*	*1*	*1*	*1*	*1*

Note: The summation of equipment by mine can exceed total equipment on site, as equipment will be transferred to different mines as required.
Source: JDS (2019)

26.3.12.2 Fixed Plant Equipment

Fixed plant equipment in the mine plan includes ventilation fans, heaters, dewatering pumps, compressors, and electrical distribution systems. A list of fixed plant equipment is summarized in Table 26-19.

Table 26-19: Mine Fixed Plant Equipment Summary

Fixed Plant Equipment	Max # of Units
Ventilation	
Fork Axial Fan and Heater	1
Wedge Axial Fan and Heater	1
Russet South Axis Fan and Heater	1
Primary Dewatering Pumps	
150hp	4
100hp	4
Services	
13.8-4.2kV transformer (surface)	4
4.2kV underground substations	4
Surface compressor	4

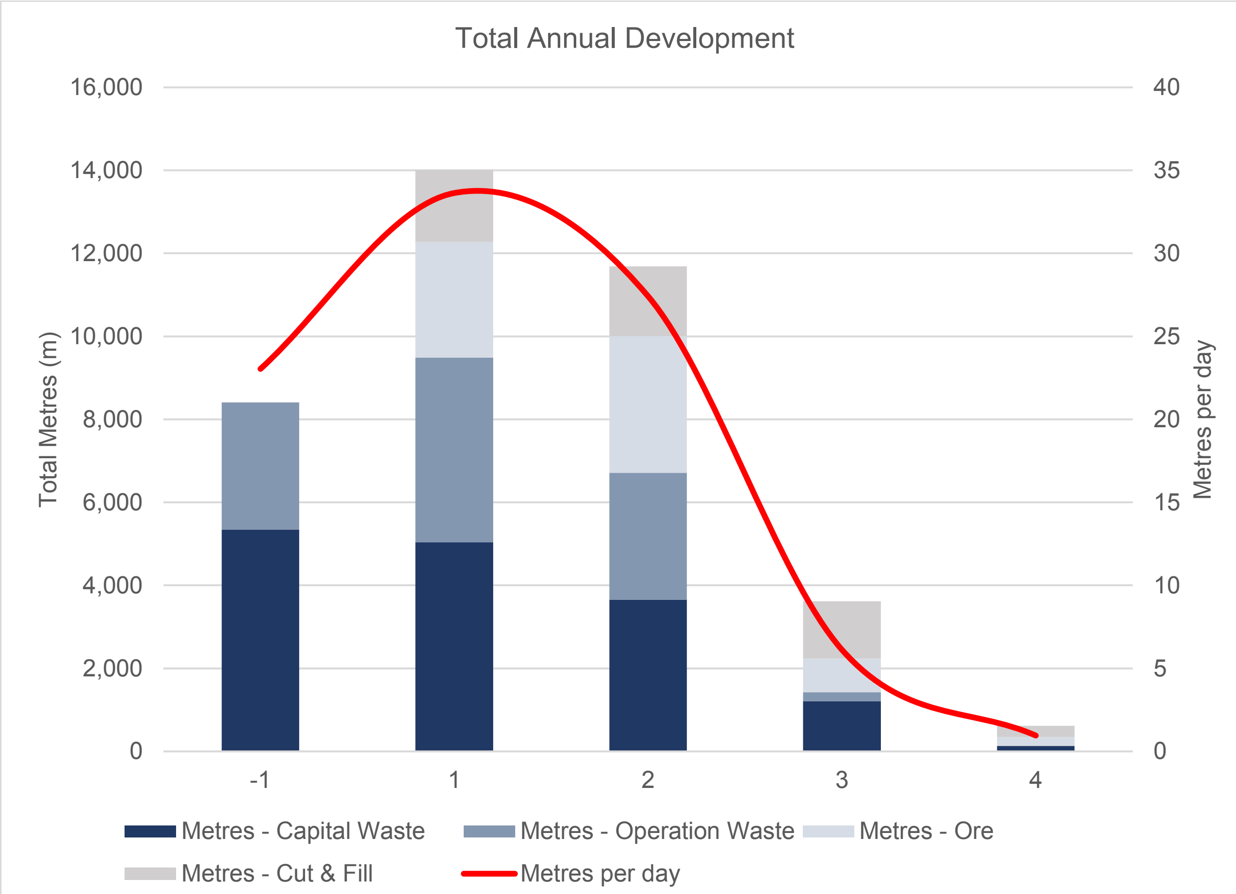
Source: JDS (2019)

26.3.13 Mine Schedules

26.3.13.1 Mine Development Schedule

The development schedule for each mine is based on creating sufficient working headings to maintain steady state production. The schedule has also been designed to provide secondary egress and positive ventilation flow throughout the mine prior to production. The development schedule is summarized in Figure 26-22.

Figure 26-22: Total Annual Development



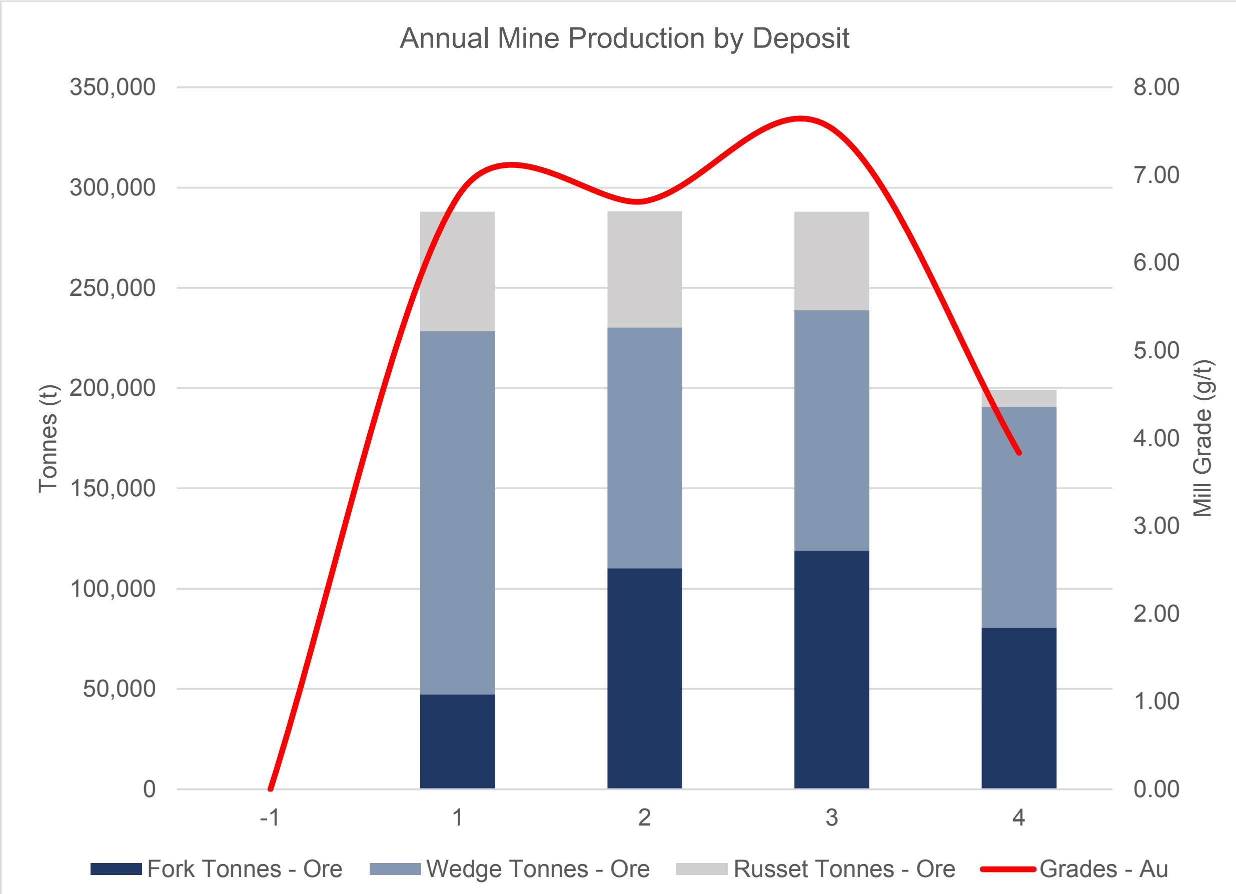
Source: JDS (2019)

26.3.13.2 Mine Production Schedule

Full steady state production is expected to commence in year one, at 288,000 t mined, this is a combined mining rate of all three mines. Between the three mines, steady state production is expected to continue throughout the mine life and end in Q4 of Year 4. Annual production of the three deposits is presented in Figure 26-23 and Table 26-20 and summarized as follows:

- Selecting higher grade stopes earlier in the mine life is limited by the mining methods selected, both LH and MCF mining methods require mining from the bottom up;
- The highest grades from the Fork and Wedge deposit are mined in Year 3, as for Russet, the highest grades are mined in Year 1;
- The portion of mineralized rock mined in each deposit varies by month; and
- Mining of the Russet South deposit ends in Q3 of Year 4, three months before Wedge and Fork.

Figure 26-23: Annual Mine Production by Deposit



Source: JDS (2019)

Table 26-20: Mining Summary

Parameter	Units	Totals	Year				
			-1	1	2	3	4
Mill Feed	**kt**	**1,063**	**-**	**288**	**288**	**288**	**199**
Fork	*kt*	*357*	-	*47*	*110*	*119*	*81*
Wedge	*kt*	*531*	-	*181*	*120*	*120*	*110*
Russet South	*kt*	*175*	-	*59*	*58*	*49*	*8*
Mining & Milling Rate	**t/d**	**800**	**-**	**800**	**800**	**800**	**830**
Fork	t/d	276	-	132	306	331	336
Wedge	t/d	407	-	503	334	333	459
Russet South	t/d	124	-	165	161	136	35
Diluted Grade	**g/t**	**6.40**	**-**	**6.75**	**6.70**	**7.53**	**3.83**

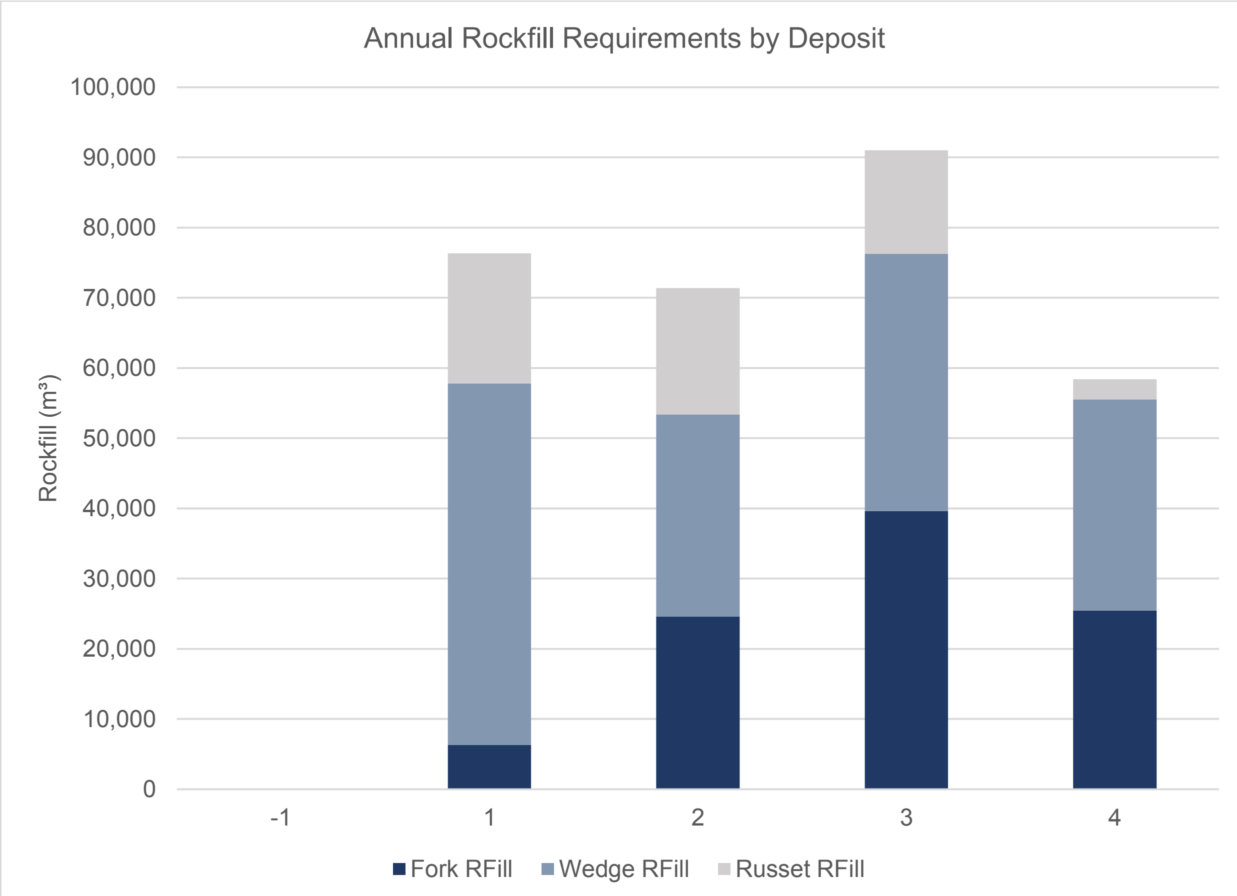
Parameter	Units	Totals	Year				
			-1	1	2	3	4
Fork	*g/t*	*4.50*	-	*4.83*	*4.25*	*5.10*	*3.73*
Wedge	*g/t*	*7.30*	-	*6.67*	*8.92*	*10.0*	*3.64*
Russet South	*g/t*	*7.55*	-	*8.53*	*6.76*	*7.36*	*7.25*
Mineralized Development	**km**	**7.1**	**-**	**2.8**	**3.3**	**0.8**	**0.2**
Lateral Capital Development	**km**	**20.5**	**7.7**	**7.2**	**4.9**	**0.7**	**0.0**
Lateral Operational Development	**km**	**4.5**	**0.6**	**2.0**	**1.5**	**0.3**	**0.1**
Vertical Development	**km**	**1.4**	**0.4**	**0.5**	**0.4**	**0.1**	**-**
Development Rate	**m/day**	**18**	**23**	**33**	**27**	**5**	**1**
Rock Fill Placed	**kt**	**672**	**-**	**173**	**161**	**206**	**132**

Source: JDS (2019)

26.3.13.3 Mine Backfill

Backfill distribution for all three mines are based on the mining sequence of both development and production tonnages. Backfill material will comprise solely of RF. The delivery schedule for RF for the three mines is summarized in Figure 26-24.

Figure 26-24: Annual Rockfill Schedule



Source: JDS (2019)

26.4 Recovery Methods

The recent metallurgical test program completed at Base Metallurgical Labs in Kamloops, BC (BL0354), summarized in Section 26.1.2, has demonstrated that gravity concentration followed by pre-oxidation, cyanide leach, carbon adsorption/desorption and electrowinning can yield an average overall gold extraction in the range of 96.5% to 98.5%. Results from this test program were used to confirm the corresponding process design criteria, mechanical equipment list and flowsheets developed for the Madsen deposit can be used for the Fork, Russet South, and Wedge deposits. No changes to the existing plant are required for the processing. The processing plant is described in Section 17.

The primary crushing plant will have a throughput of 800 t/d with an average blended head grade of 6.40 g/t Au. The crushing circuit will operate at an availability of 50%, resulting in an hourly throughput of 66.7 t/h. The milling, gravity, leach and CIP circuits will operate 24 hours per day, 365 days per year at an availability of 95%, resulting in an hourly throughput of 35.1 t/h. The carbon plant is expected to transfer 1 tonne of loaded carbon daily to the elution circuit to recover the gold to doré bars.

26.5 Project Infrastructure

The Fork, Russet South, and Wedge deposits will require the upgrading and/or construction of the following key infrastructure items:

- Additional storage capacity development of the TMF;
- Portal pads, mine rock stockpiles and access roads for each of the Fork, Russet South, and Wedge deposits;
- A fresh water supply and a water discharge pipeline for each deposit;
- Electrical overhead powerlines to each of the Fork, Russet South, and Wedge deposit portals with 4160 V distribution;
- Additional secured storage, fencing and security camera monitoring at each portal location;
- An office and personnel facility at the Russet South and Wedge portals; and
- Additional surface mobile equipment for personnel and equipment transport between the Madsen site and the Fork, Russet South, and Wedge development.

26.5.1.1 Site Layout

The Project site layout is shown in Figure 26-25.

The proposed site layout has been configured for optimal construction access and operational efficiency. The existing Madsen process and infrastructure facilities will be used with no upgrades necessary. The TMF dams will be further developed and raised to add storage capacity to the existing TMF. New access roads, portal pads and portal facilities will be installed to support the mining operations. Road construction will be minimized by using existing roads and development where possible.

Figure 26-25: Fork, Russet South, and Wedge Deposits Site Layout

Source: JDS (2019)

26.5.2 Access Roads

The Project will continue to use the site access from provincial highway ON-618 S during operations.

New roads and access will be built to access the Fork, Russet South, and Wedge portal locations. The existing Russet Road will be utilized to access the Russet South portal from the Madsen site. A short access road to connect the logging road and portal will be constructed. The Fork portal area will have a dedicated use access road built, approximately 600 m long linking to the Madsen portal area roads. Two maintenance access roads will connect from the Fork portal to the two Fork vent raise locations. The Wedge portal access will utilize the highway for 2 kilometers, to a new access road, approximately 550 m long, created by upgrading a legacy mining haul road.

26.5.3 Power Distribution

The existing substation at the Madsen site will be used to connect and supply power to the portal locations. It will not require any upgrades. Power will be distributed by new 4.16 kV overhead lines. A single line will connect to the Madsen substation and then branch out to supply each of the Fork, Russet South, and Wedge portal sites.

26.5.4 Process Plant

The processing plant on site will be used for the milling of the Fork, Russet South, and Wedge deposit ore. The crusher, ore bin, conveyors and process equipment in the mill will not require any modifications. Contract trucking will be used to haul the ore from the portals to the crusher.

26.5.5 Existing Madsen Facilities

The following facilities from the Madsen site will be used to support the mining operations of the Fork, Russet South, and Wedge deposits:

- Administration offices;
- Mine dry;
- Maintenance shop;
- Warehouse; and
- Fuel storage.

26.5.6 Mobile Equipment

The existing mobile equipment will be used during operations, and the following vehicles will be added for servicing the Fork, Russet South, and Wedge portal locations, shown in Table 26-21.

Table 26-21: Mobile Equipment to Service Fork, Russet South, and Wedge Locations

Equipment Required	Operating Description	Qty
Pickup Trucks ½ T	Operations	3
Pickup Truck ¾ T	Maintenance & Site Services	1
Semi-Tractor & Low-Boy Trailer	Transport mobile equipment between Madsen and Portals	1

Source: JDS (2019)

26.5.7 Portal Surface Infrastructure

26.5.7.1 Un-mineralized Mine Rock Stockpiles

Each of the portals will be cleared, stripped of organics, and levelled with un-mineralized mine rock to construct the pads as shown in Figure 26-26. The portal pads will be used as mine rock stockpiles during construction and operations. Initially un-mineralized rock will be stockpiled in Years -1, 1 and 2, and then reclaimed as the mines use it for backfill. Un-mineralized mine rock not used for backfill will be left on surface and contoured as the base for a closure cover. The Wedge mine will utilize a nearby legacy tailings area for placement and contouring of excess un-mineralized mine rock, not be utilized for underground backfill that could be used as a base for a closure cover.

26.5.7.2 Mine Water Supply

A fresh water supply will be installed for each of the three new portals, as well as a water discharge line and outlet. These will be at the local lakes, as shown in Figure 26-26. Each portal will utilize a pumping system from the lake intakes, and pump freshwater to the mine via installed piping. The system will be winterized, with insulated and heat traced piping, and de-icing bubblers at the intake pump stations. Power to the pumps will be from the portals, with distribution cables installed with the supply pipelines.

Mine water discharged from the portals will be from the decanted mine sumps, pumped to the discharge locations through winterized piping.

26.5.7.3 Portal Ancillary Facilities

Each of the portals will be secured with an access gate, fencing and security cameras monitored by the Madsen site security station.

The Russet South and Wedge Portal areas will be equipped with a modular office and personnel break facility. The Wedge office will also serve the Fork portal.

Lockable shipping containers will be used at each site for secured storage of consumables and parts to service the mines.

26.5.8 Tailings Management

The PEA design includes the ore from the Madsen Mine (Feasibility Study) as well as potentially mineable material from the Fork, Russet South, and Wedge deposits which will be mined simultaneously when mining at the Madsen Mine is complete after 12.3 years of operation. The PEA assumes mining a total of approximately 4.6 million tonnes (Mt) of mineralized material using underground mining methods and processing it over a 16-year mine life at a throughput of 800 t/d (PEA and Feasibility tonnages combined).

The Fork, Russet South, and Wedge deposits will be mined for 3.8 years of these years and will produce 1.1 Mt of tailings of this total. Tailings will not be used as backfill and all tailings will be deposited in the Madsen TMF. These additional tailings will be managed in the TMF by expanding the Polishing Pond storage capacity, modifying the tailings deposition plan and deferring the closure of Cell B until the end of mine operations.

Tailings will be deposited in the TMF in two separate phases. Phase 1 tailings deposition will be the same as the FS design. Phase 2 tailings deposition is modified to provide storage for the additional 1.1 Mt of tailings. The overall project general arrangement is shown on Figure 26-26 and shows the maximum operating footprint of each facility but does not represent any particular year of operation. The tailings deposition strategy for each phase is outlined below.

- Phase 1 (Year 1 – 6): Tailings deposition in Cell A; and
- Phase 2 (Year 7 – 16): Tailings deposition in Cell B, Cell C and Polishing Pond.

Phase 1 Tailings Deposition

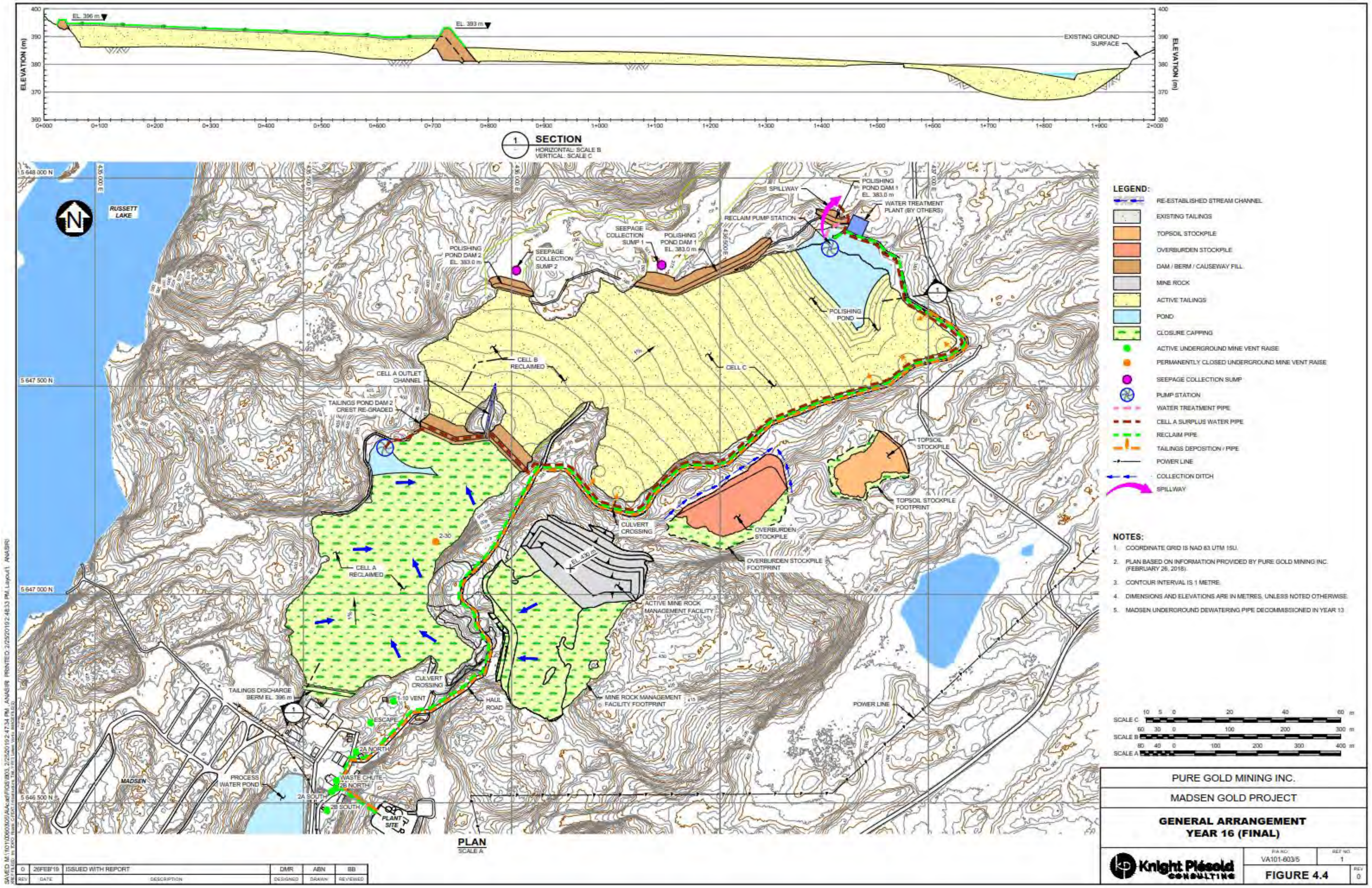
Processing and tailings production begins at the start of Year 1. Tailings containment will be provided in Cell A by raising the existing Tailings Pond Dams. Tailings will be deposited from the Tailings Pond Dams and from the east side of the facility to develop tailings beaches against the dams and control the location of the supernatant pond so that water can be efficiently reclaimed to the Polishing Pond. Tailings will be delivered to Cell A via a single tailings pipe, delivering whole mill tailings and overflow tailings from the underground backfill cyclone activities. Surplus water accumulation in Cell A will be pumped or gravity decanted directly to the Polishing Pond for storage prior to treatment and release. Cell A will reach its maximum capacity in Year 6 and will be closed and reclaimed during the following two years of operations. The TMF layout at the end of Year 6 is the same as the FS.

Phase 2 Tailings Deposition

Slurry tailings deposition will commence into the natural confinement of the Polishing Pond from Year 7 to Year 10. Tailings will be deposited at the northeast end of the Polishing Pond to begin filling the topographic depression and directing the supernatant pond towards the reclaim pump located at the north end of the facility. At the end of Year 8, Cell A will be fully closed with an engineered closure cover. The contributing and direct catchment areas for Cell A will now be classified as non-contact water and runoff from this area will be collected and pumped downstream of the TMF using the existing Cell Surplus Water pumping system.

Additional confinement in Cell B, Cell C and the Polishing Pond will be created by raising the existing Polishing Pond Dams in Year 10. The remainder of the Phase 2 tailings will be deposited in Cell B, Cell C and the Polishing Pond from Year 11 to Year 16. These cells will be regraded using selective tailings deposition as part of the closure objectives and will be fully closed with an engineered closure cover. The TMF layout at the end of operation in Year 16 is shown on Figure 26-26

Figure 26-26: TMF General Arrangement – Operations – Final



Source: KP (2019)

26.5.8.1 Polishing Pond Dams Expansion

Polishing Pond Dam 1 and Polishing Pond Dam 2 will be expanded and operated as free draining structures similar to the concept used at the Tailings Pond Dams. The dams will be raised using the downstream construction method. They will comprise zoned structures that will have a low permeability compacted tailings upstream zone with appropriate filter and transition zones to prevent downstream migration of tailings and fine material through the rockfill. A layer of non-woven geotextile and erosion protection material will be placed along the upstream embankment face directly on top of the compacted tailings zone for erosion protection.

The dams will be raised in one stage in Year 10 of operations. Polishing Pond Dam 1 will be raised to elevation 383 m and Polishing Pond Dam 2 will be raised to elevation 385 m. Seepage through the dams will be controlled by the upstream compacted sand, tailings beaches and directing the supernatant pond to the natural confinement in the Polishing Pond. Seepage collected in the drainage zones will flow to the Seepage Collection Sumps constructed downstream. The downstream shell zone designated Zone C will support the tailings sand and filter zones. The Zone C shell zone will be constructed using pervious Mine Rock.

Instrumentation will be installed in the Polishing Pond Dams and underlying foundations and monitored during construction and operations to assess performance and to identify any conditions that differ from those used during design and analysis. Amendments to the ongoing designs, operating strategies and/or remediation work can be implemented to respond to changing conditions, should the need arise.

26.5.8.2 Emergency Spillways

The operating basis for the TMF includes temporary storage of the Environmental Design Flood (EDF). This storage provision has been incorporated into the staging of Cell A for the Cell A catchment and within the Polishing Pond for the entire TMF catchment. Extreme inflows larger than the EDF exceeding the available storage capacity will discharge by way of emergency spillways constructed at Cell A and at the Polishing Pond. The primary objective of the emergency spillways will be to protect the integrity of the dams during an emergency and they are not intended to be used at any stage during operations. The spillways outlined in the Feasibility Study will be used for the TMF expansion until Year 10 of operation. The existing emergency spillway in the Polishing Pond will be raised to elevation 382 m as part of the Polishing Pond Dams expansion.

26.5.8.3 Mechanical Systems

The mechanical systems outlined in the Feasibility Study will be used for the remaining 3.8 years of operation. No change to the mechanical systems will be required.

26.5.9 Mine Rock Management

Un-mineralized mine rock material produced at the Madsen Mine will continue to be managed on surface in the MRMF as outlined in the Feasibility Study. Mine Rock from underground mining development at the Wedge, Russet South, and Fork deposits will be used directly in mine backfill with surplus mine rock stored on surface in stockpiles near the portal pad located at each deposit. A combined total of 1.0 Mt of un-mineralized mine rock will be produced at the Fork, Russet South, and Wedge deposits with the majority being managed in the underground mines. The surplus un-mineralized mine rock managed on surface will be deposited to fill natural topographic depressions near the portal sites where possible with an overall slope of 2.5H:1V for closure and reclamation. The total volume of un-mineralized mine rock remaining at each stockpile at closure is summarized in Table 26-22.

Table 26-22: Un-mineralized Mine Rock Stockpile at Closure

Deposit	Mine Rock (m^3)
Wedge	106,400
Russet South	19,700
Fork	18,500

Source: KP (2019)

26.5.10 Water Management

26.5.10.1 Water Management Strategy

The water management strategy for the PEA design follows the water management strategy outlined in the Feasibility Study. Sufficient capacity for mine operations, underground dewatering and storm storage will be maintained. The following water management strategies will be implemented throughout the pre-production and operations periods of the TMF.

- Reclaim water will be pumped directly from the Polishing Pond to the Process Plant over the 16-year mine life;
- Underground dewatering flows from the Madsen mine will be pumped directly to the Polishing Pond until the end of mine life in Year 12.3, with the exception of Pre-production Phase 1 when it will be discharged directly to the downstream environment;
- At the end of Year 8, Cell A will be fully closed with an engineered closure cover. The contributing and direct catchment areas for Cell A will now be classified as non-contact water and runoff from this area will be collected and pumped downstream of the TMF using the existing Cell Surplus Water pumping system;
- Underground dewatering from the Fork, Russet South, and Wedge mines will not be managed in the TMF;
- Site contact water will be managed in the Polishing Pond during operations;
- Excess water in the Polishing Pond will be pumped to the WTP, treated and released at the current effluent discharge point downstream of the TMF;
- Treated water can only be released from the WTP during periods when the receiving waterway is not frozen. Water releases are planned between March 15 and November 30 when conditions meet permit requirements; and
- Water treatment and release is required each year to maintain sufficient capacity in the Polishing Pond for the upcoming year of operations.

26.5.10.2 Water Balance

The water balance model developed for the Feasibility Study was updated to include the additional years of operation and the change in closure of Cell B from Year 8 in the Feasibility Study to the final closure scenario in the PEA. Underground dewatering flows to the TMF from the Madsen Mine are also terminated in Year 12.3. The results indicate the TMF will operate in a surplus during all phases of operations and under the full range of variable climatic conditions, including prolonged wet and dry cycles. Surplus water will be managed within the Polishing Pond prior to being treated and released to the downstream environment. The TMF water balance demonstrates that the water management strategy can be executed under the full range of variable climatic conditions available based on the inputs presented. This includes achieving the reclaim water requirements under prolonged dry climate cycles for the entire mine life and determining that there is sufficient operating capacity within the TMF to manage surplus water during prolonged wet climate cycles.

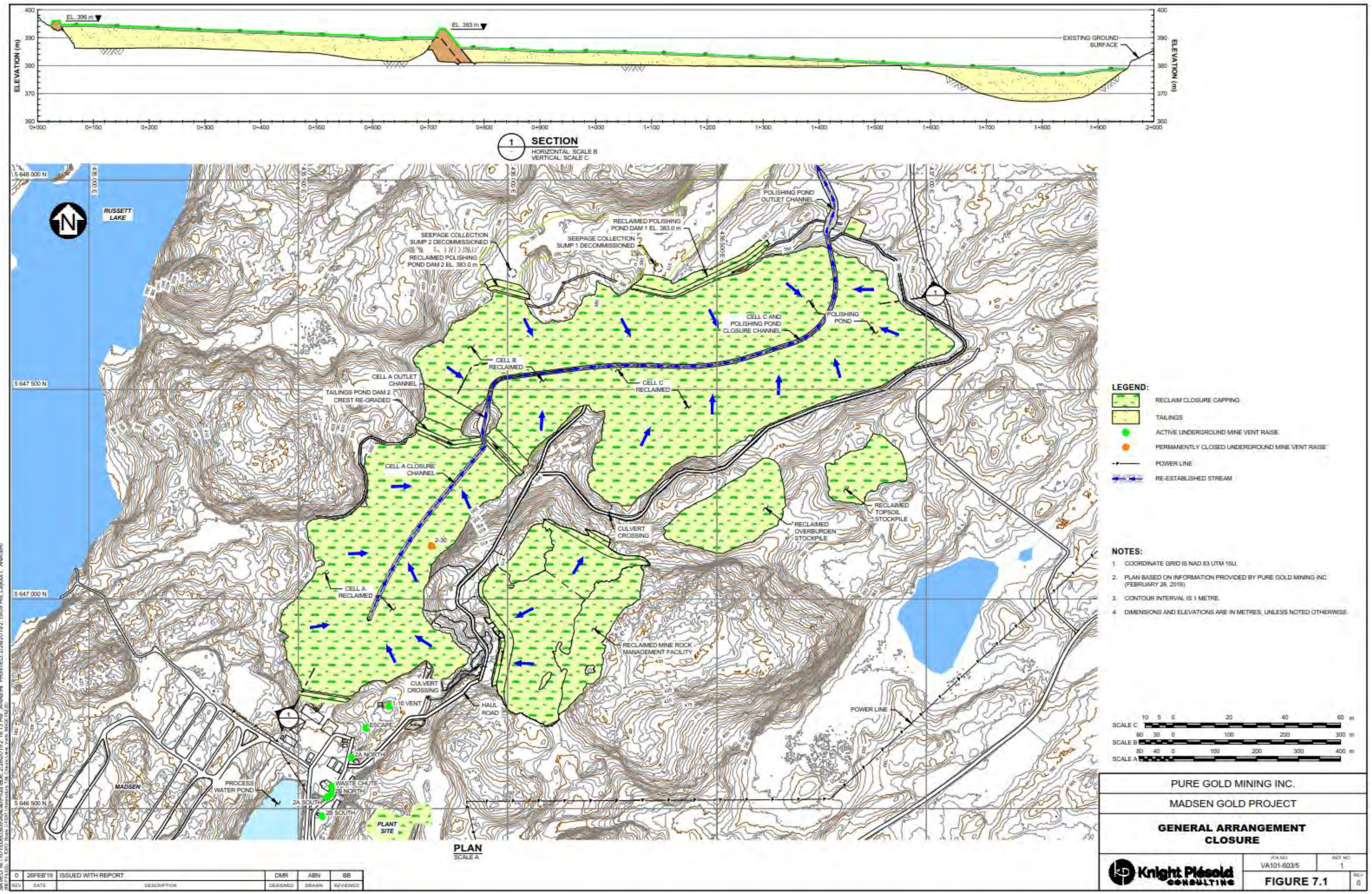
26.5.11 Closure and Reclamation

The closure design philosophy for the TMF remains the same as the FS involves removing all surface water ponds at the end of operations and covering the tailings surface with an engineered cover system, which naturally sheds non-contact run-off to the downstream environment. This closure plan would eliminate the need for long-term water treatment post closure. The objective of the closure and reclamation initiatives will be to eventually return the TMF site to a self-sustaining facility with pre-mining land capability. The closure plan regrades the TMF into a landform with a similar native species vegetation cover as the surrounding topography. The closure plan regrades the TMF into a landform with a native species vegetation cover to the surrounding topography. Closure channels will be constructed in Cell A, Cell B, Cell C and the Polishing Pond to direct flows downstream of the TMF. The emergency spillways constructed during operations will be upgraded to closure discharge channels to meet CDA Post-Closure Guidelines.

Reclamation of the MRMF will be completed at mine closure. The closure of the MRMF will typically include re-sloping of the face to a 2.5H:1V or flatter slope. The final facility bench crests will be smoothed or rounded to improve the long term erosional stability of the facility. The Mine Rock Stockpiles constructed at Fork, Russet South, and Wedge will be concurrently reclaimed where possible to meet closure objectives. Reclamation of the stockpiles will be conducted in conjunction with on-going geotechnical and environmental monitoring to assess slope stability, sediment control, and water quality objectives. Topsoil placement and revegetation of the slopes to return the facilities to pre-mining use.

The overall TMF general arrangement at closure is shown on Figure 26-27.

Figure 26-27 : TMF General Arrangement – Closure



Source: KP (2019)

26.6 Environment and Permitting

26.6.1 Regulatory Involvement

26.6.1.1 Environment Assessment

With development of the Preliminary Economic Assessment (PEA) resources it is possible that both the provincial and federal environmental assessment processes could be triggered, primarily due to new power line routing and the planned release of mine water to the Dome Creek catchment. The Dome Creek catchment has been impacted in the past from the historical Starratt Olsen mine which has been closed since 1956. In its current form, the PEA mine footprint is expected to increase by more than 50% but the daily tonnages will remain at 800 t/d and therefore could trigger a CEA Act environmental assessment. With potential advancement to Preliminary Feasibility these designs may be optimized and it may be possible to use alternative designs that could potentially avoid triggering environmental assessment. It is assumed that mineralized material from the PEA mine plan will be hauled to the then operating and permitted Madsen mill for processing and tailings will be managed in the Madsen TMF equipped with a water treatment plant.

26.6.1.2 Federal Regulatory Agencies

Federal agencies that may have regulatory involvement with the development of the PEA mine plan include Environment Canada, Natural Resources Canada, and Fisheries and Oceans Canada.

26.6.1.3 Provincial Regulatory Agencies

The Ontario Ministry of Energy, Northern Development and Mines, Ministry of Environment, Conservation and Parks, Ministry of Transportation as well as the Ministry of Natural Resources and Forestry, will each have key project development permit responsibilities.

26.6.1.4 Permit Requirements

Based on the current PEA design the following potential changes to the Madsen Gold Project permits may be required and new permits may be needed for the development of the PEA mine plan as outlined in Table 26-23.

Table 26-23: Potential Authorizations, Approvals and Permits Required

Permit / Approval	Trigger	Agency
Work Permits	Any work on water crossings (culvert installations, construction of a bridge, seasonal ice bridges, etc.)	MNRF
	Any work that involves upgrading of existing roads, building new roads or building trails on Crown Land	MNRF
	Beaver dam removal, collection of fish for scientific purposes, trapping of nuisance beavers	MNRF
Forest Resource License	Removal of merchantable timber on Crown Land. Will require review of forest harvest agreements.	MNRF
Approval	Installation/construction of water retaining structures / dams.	MNRF
Burning Permit	Burning of removed vegetation	MNRF

Permit / Approval	Trigger	Agency
Aggregate Permit / License	Removal of aggregate from pit or quarry (road construction)	MNRF
Authorization	Activities that may harm, harass or threaten a species listed under the Act or damage their habitat	MNRF
Environmental Compliance Approval (ECA) - Air and Noise	Discharge of airborne contaminant into receiver environment, including noise (milling, ventilation, emergency generator, aggregate crushing, screening and stockpiling, etc.)	MECP
ECA – Industrial Sewage	Industrial Sewage Works Environmental Compliance Approval (ECA) requirement for discharge of mine wastewater and domestic sewage to receiver environments	MECP
Water Well Installation	Well drilling for water supply or groundwater monitoring, in accordance with O. Reg. 903 Ontario Water Resources Act	MECP
Permit To Take Water (>50,000 L)	Taking more than 50,000 L per day	MECP
Entrance Permit	Requirement for a new or upgraded road entrance onto a provincial highway	Ministry of Transportation (MTO)
Encroachment Permit	Activities within 45 m of the highway may be controlled for safety considerations	MTO
Closure Plan	Mine development	ENDM
Notice of Material Change	Change to a Closure Plan	ENDM
Notice of Project Status	Mine development	ENDM
Fish Habitat Authorization	Work on water crossings or work near fish habitat	DFO
EC Damage or Danger Permit	Perform a regulated activity that harms migratory birds or an area frequented by migratory birds	Environment Canada (EC)
DFO Approval	Any work for crossing a navigable water body that may interfere substantially with navigation	Transport Canada, DFO

Source: Pure Gold (2019)

26.6.2 Baseline Studies

In order to develop the PEA mine plan it will be necessary to undertake baseline scientific studies, predictions and forward modelling with an expanded footprint beyond those completed for the Madsen FS. These include:

- Species at Risk;
- Dome Creek catchment water quality;
- Dome Creek hydrology;
- Ground water;

- Mine rock geochemistry;
- Dome Creek aquatic biology; and
- Air, noise and vibration.

26.6.2.1 Species at Risk

In November 2018 MNRF provided an updated list of SAR in the Red Lake District (Table 26-24). Although only Myotis bat species have been observed on the Madsen site to date, habitat at the site may be suitable for some of these species. FRi has been contracted to assess the suitability of the habitat as well as undertake baseline surveys.

Table 26-24: List of Species at Risk Observed in the Red Lake District and their Associated Status Under the Ontario Endangered Species Act

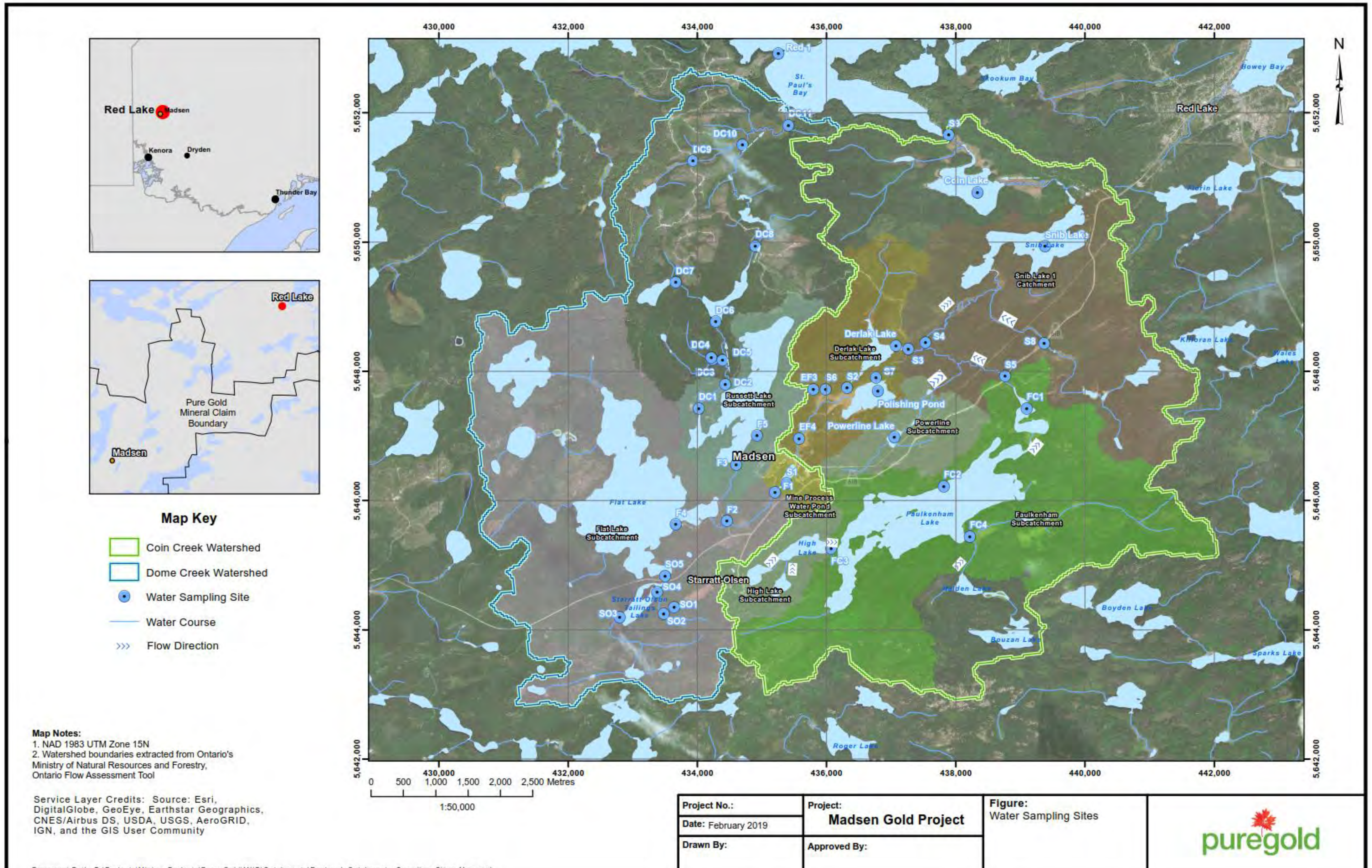
Common Name	Scientific Name	ESA Status
Birds		
Bank Swallow	*Riparia riparia*	Threatened
Barn Swallow	*Hirundo rustica*	Threatened
Eastern Whip-poor-will	*Antrostomus vociferus*	Threatened
American White Pelican	*Pelecanus erythrorhynchos*	Threatened
Horned Grebe	*Podiceps auritus*	Special Concern
Common Nighthawk	*Chordeiles minor*	Special Concern
Olive-sided Flycatcher	*Contopus cooperi*	Special Concern
Canada Warbler	*Cardellina canadensis*	Special Concern
Bald Eagle	*Haliaeetus leucocephalus*	Special Concern
Evening Grosbeak	*Coccothraustes vespertinus*	Special Concern
Yellow Rail	*Coturnicops noveboracensis*	Special Concern
Short-eared owl	*Asio flammeus*	Special Concern
Mammals		
American Badger	*Taxidea taxus*	Endangered
Northern Myotis	*Myotis septentrionalis*	Endangered
Little Brown Myotis	*Myotis lucifugus*	Endangered
Wolverine	*Gulo gulo*	Threatened
Woodland Caribou	*Rangifer tarandus*	Threatened
Turtles		
Snapping Turtle	*Chelydra serpentina*	Special Concern
Fish		
Lake Sturgeon (Northwestern Ontario population)	*Acipenser fulvescens*	Threatened
Lake Sturgeon (Southern Hudson Bay-James Bay population)	*Acipenser fulvescens*	Special Concern

Source: Pure Gold (2019)

26.6.2.2 Dome Creek Catchment

The PEA mine development is planned within the Dome Creek Catchment which includes the historical Starratt Olsen Mine and TMF, TMF catchment pond, Flat Lake, Russet Lake and Dome Creek and associated unnamed lakes to Red Lake. All mineral processing and tailings management is planned within the Coin Creek catchment (Madsen mill and TMF). It is important to also note that both Flat Lake and Russet Lake are used for recreational fishing and that the Madsen Community Water Supply is drawn from Russet Lake. In addition, it is currently planned to use Faulkenham Lake (Coin Creek catchment) as the receiving water body when dewatering the mine workings from the Wedge and Fork Deposits and Dome Creek when dewatering the Russet South mine workings. The catchment and potential monitoring sites are shown in Figure 26-28.

Figure 26-28: Dome Creek Catchment and Potential Water Monitoring Sites



Source: Pure Gold (2019)

26.6.3 Consultation

Pure Gold has committed to engagement and consultation with local First Nations, Metis Nation of Ontario, provincial and federal governments, the public, and stakeholders throughout all stages of Project planning, regulatory review, and construction. The intent is to provide all interested parties with opportunities to learn about the Project, identify issues, and provide input with the goal of positively enhancing Project planning and development.

Pure Gold recognises the importance of full and open discussion of the issues and options associated with the development of the Project and the related concerns those individuals or communities may have in relation to the activities. In light of this, Pure Gold will maintain open and honest communications with local communities and individual stakeholders throughout all stages of the Project. Pure Gold intends to 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.

26.6.3.1 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 26-29).

The Ontario Ministry of Energy, Northern Development and Mines (ENDM) has advised Pure Gold 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 ENDM will undertake and fulfill the Crown's Duty to Consult. Currently, the primary role of Pure Gold with First Nations is to ensure that appropriate information sharing occurs (Figure 26-30).

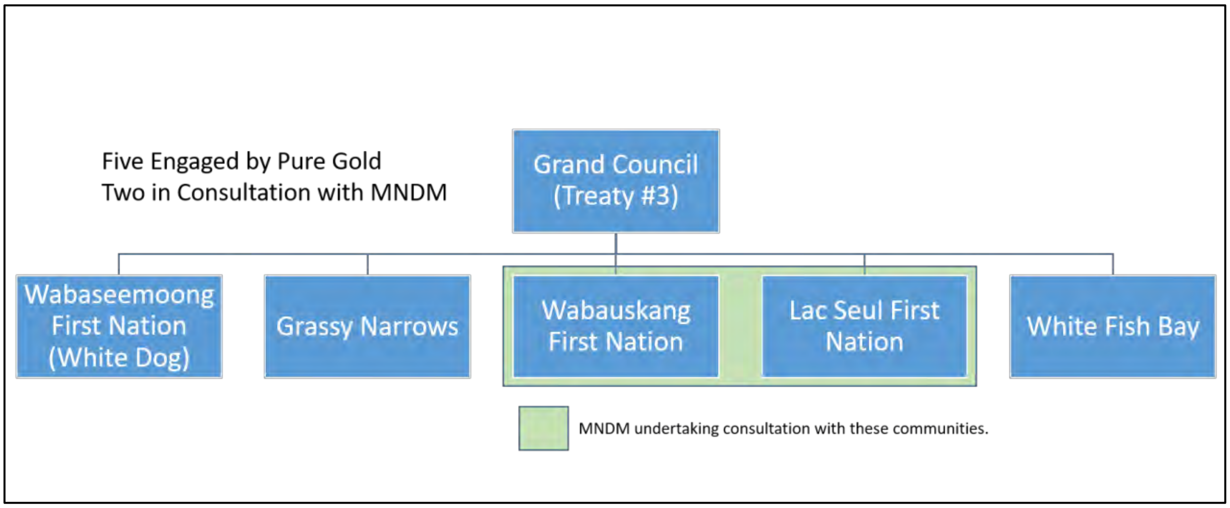
Lac Seul and Wabauskang, two communities which Pure Gold is required to consult with, have come together on a Shared Territory Protocol for resource projects that are of mutual interest. This allows Pure Gold to consult openly with both parties functioning as one. Pure Gold and the communities are working towards a Project Agreement. The 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. Pure Gold and the First Nations have agreed terms and are working toward finalizing the agreement.

Pure Gold retains a record of all communications and discussions and to date all interactions have been mutually positive and supportive.

Figure 26-29: Treaty 3 First Nations

Source: Pure Gold (2019)

Figure 26-30: Treaty 3 First Nations Engaged by Pure Gold



Source: Pure Gold (2019)

26.6.3.2 Community Considerations

Pure Gold has continued ongoing communication with Red Lake Municipality and the various regulators who have an interest in the Project. Pure Gold has also developed a Consultation Plan that forms the basis of their ongoing plans for community, regulator and First Nations Consultation. Currently, Pure Gold's plan to reopen the Madsen Mine is largely seen as a benefit to the local community, which has suffered economically from the decline in mining over recent years. It is expected that a mine life extension such as that described in the PEA will be viewed in the same light.

26.6.3.3 Regulator Considerations

Pure Gold has engaged with regulators and established good working relationships. Regulators from, CEAA, MECP, MoL, ENDM, MNRF and Red Lake Municipality have been involved with Pure Gold at the mine or in evaluation of proposed reopening activities at the Madsen Mine. It is anticipated that this process will be continued for the PEA mine plan.

26.7 Capital Cost Estimate

LOM Project capital costs total $101 M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the Fork, Russet South, and Wedge deposits to an 800 t/d mining production rate. Initial capital costs total $57 M and are expended over a 12-month pre-production period on engineering, construction and commissioning activities;
- Sustaining Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $42 M and are expended in operating Years 1 through 4; and
- Closure Costs – includes all costs related to the closure, reclamation, and salvage value, post operations. Closure costs total $1.2 M (after salvage credits), and are incurred in Year 5.

The capital cost estimate was compiled using a combination of quotations, database costs, and factors; the overall cost estimate was benchmarked against similar operations. Table 26-25 presents the capital estimate summary for initial and sustaining capital costs in Q1 2019 dollars with no escalation.

Table 26-25: Capital Cost Summary

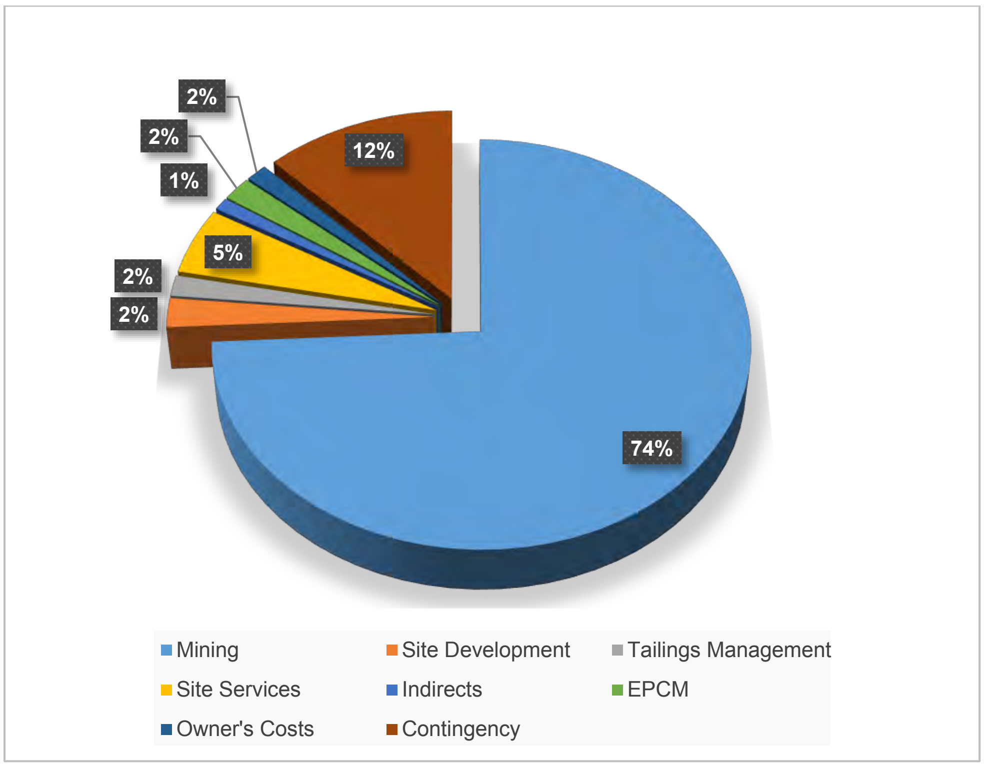
WBS AREA	WBS DESCRIPTION	Pre-Production Cost ($M)	Sustaining Cost ($M)	Project Total Cost ($M)
1000	**Mining**	**42.5**	**42.1**	**84.6**
1100	Underground Mobile Equipment	6.6	8.6	15.2
1200	Underground Infrastructure	5.0	2.0	7.0
1300	Capital Development	21.5	31.5	53.0
1400	Capitalized Production	9.4	-	9.4
2000	**Site Development**	**1.3**	**-**	**1.3**
2100	Bulk Earthworks (Pads)	0.4	-	0.4
2200	Site Roads	0.9	-	0.9
4000	**Tailings Management**	**0.9**	**-**	**0.9**
4100	Tailings Management Facility	0.8	-	0.8
5000	**Site Services**	**3.0**	**-**	**3.0**
5100	Power Supply & Distribution,	0.9	-	0.9
5200	Ancillary Fork, Russet South, and Wedge Surface Facilities	0.4	-	0.4
5300	Surface Mobile Equipment	0.5	-	0.5
5400	IT & Communications	0.1	-	0.1
5500	Water Supply and Distribution	1.1	-	1.1
6000	**Closure**	**-**	**1.4**	**1.4**
6100	TMF WRMF Closure	-	0.6	0.6
6400	Fork, Russet South, and Wedge Portals Closure	-	0.8	0.8
7000	**Project Indirects**	**0.6**	**-**	**0.6**
7100	Contracted Support Services	0.2	-	0.2
7200	Temporary Facilities & Utilities	0.1	-	0.1
7300	Contractor Indirects	0.2	-	0.2
7400	Logistics & Freight	0.1	-	0.1
7500	Start Up & Commissioning	0.0	-	0.0
8000	**EPCM**	**1.1**	**-**	**1.1**
8100	Engineering & Procurement	0.5	-	0.5
8200	Construction & Project Management	0.6	-	0.6
9000	**Owner's Costs**	**0.8**	**-**	**0.8**
9100	Contingency (Direct + Indirect + Owner's)	0.8	-	0.8

WBS AREA	WBS DESCRIPTION	Pre-Production Cost ($M)	Sustaining Cost ($M)	Project Total Cost ($M)
10000	**Contingency**	**7.1**	**2.1**	**7.1**
10100	Contingency (Direct + Indirect + Owner's)	7.1	-	7.1
11000	**Salvage**	**-**	**-0.2**	**-0.2**
11100	Salvage	-	-0.2	-0.2
	TOTAL CAPEX	57.2	43.3	100.6

Source: JDS (2019)

Figure 26-31 and Figure 26-32 present the capital cost distribution for the pre-production and sustaining phases. As typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development.

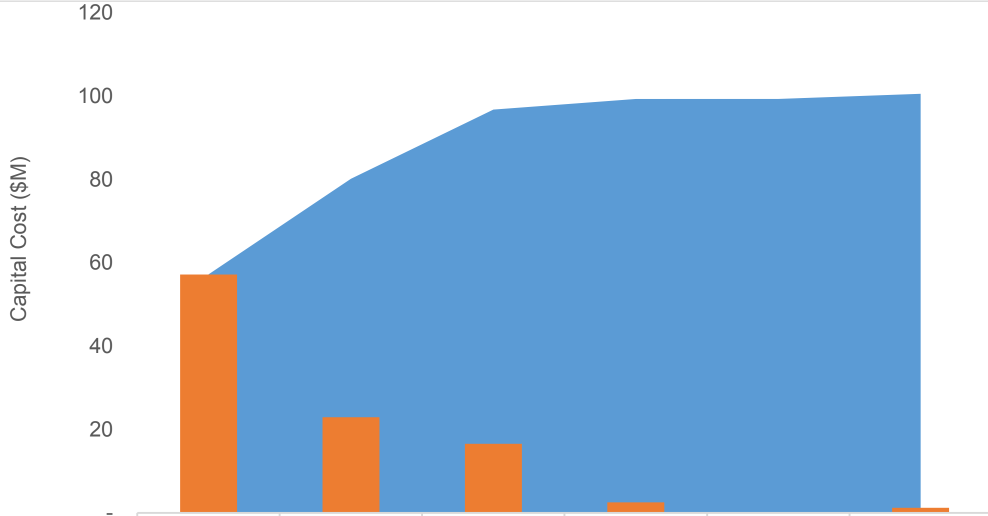
Figure 26-31: Distribution of Initial Capital Costs



Source: JDS (2019)

All capital costs for the PEA have been distributed against the development schedule to support the economic cash flow model. Figure 26-32 presents an annual life of mine capital cost profile including closure years (Years 4-5).

Figure 26-32: Capital Cost Profile



	Y-1	Y1	Y2	Y3	Y4	Y5
Cumulative Capital Cost	57.2	80.2	96.8	99.3	99.3	100.6
Annual Capital Cost	57.2	22.9	16.6	2.5	0.0	1.2

Source: JDS (2019)

26.7.1 Key Assumptions

The following assumptions were made during the development of the capital estimates:

- Underground mine development activities will be performed by the Owner's work force; and
- All surface construction (including earthworks) will be performed by contractors.

26.7.2 Key Estimate Parameters

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-20% / +30%). The overall project's definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is February 2019. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate and
- Currency: All capital costs are expressed in Canadian Dollars ($). Portions of the estimates in US Dollars (US$) were converted to Canadian Dollars at a rate of C$1.00 : US$0.75

26.7.3 Underground Mine CAPEX

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases, and similar mines in Eastern Canada. Table 26-26 summarizes the combined capital cost estimates for the three mines.

Table 26-26: Mining CAPEX Summary

Description	Unit	Initial ($M)	Sustaining ($M)	Total ($M)
UG Mobile Equipment	$M	6.6	8.6	15.2
UG Infrastructure	$M	5.0	2.0	7.0
Capital Lateral Development	$M	20.0	28.1	48.1
Capital Vertical Development	$M	1.5	3.4	4.9
Capital Period OPEX	$M	9.4	0.0	9.4
Total	**$M**	**42.5**	**42.1**	**84.6**

Source: JDS (2019)

26.7.3.1 Mobile Equipment and Replacement

Underground mining equipment quantities and costs were determined through buildup of the mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities. Mobile equipment for the three mines will either come from the Madsen Mine or purchased as required. Equipment from the Madsen Mine includes estimated engine hours at time of transfer, and used to predict equipment rebuilds and replacements. The totals include the total purchase and replacement value of the fleet. No leasing options have been placed on the equipment.

26.7.3.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database cost were used for the major infrastructure components. Allowances have been made for miscellaneous items, such as refuge station, and water supply. Capital expenditure for items such as initial PPE and radios have not been included as it is assumed that personal will be transferred from the Madsen Mine to the Fork, Russet South, and Wedge Deposits, hence the capital cost for these items were included in the Madsen Mine Feasibility Study and only sustaining cost for these items are included in the PEA study. Acquisition of underground infrastructure is timed to support the mine plan requirements.

26.7.3.3 Lateral and Vertical Capital Development

The majority of lateral development for the mines has been capitalized. Underground infrastructure, with the expectation of cross cuts or access drifts into mineralization, have been considered capital projects. These items account for 64% of all lateral development in the mines. The remaining 36% of development is captured in the OPEX.

Additionally, 100% of the vertical development and associated costs are considered CAPEX.

26.7.3.4 Capitalized Operating Cost

Capitalized production costs are defined as mine operating expenses (operating development, mine maintenance, and mine general cost) incurred prior to and during commissioning and ceasing at the commencement of commercial operations and generation of project revenues. They are included as pre-production capital costs. Once mill feed is processed, these costs transition to operating expenses.

The PEA operating cost estimates were formed on a similar basis to those described in Section 22. Capitalized product costs are included in the asset value of the mine development and are depreciated over the life of the Fork, Russet South, and Wedge Deposits within the financial model.

26.7.4 Site Development and Infrastructure Costs

Site development and infrastructure costs include Fork, Russet South, and Wedge portals site development, access roads, power and water infrastructure, and additional tailings management facility storage capacity construction. These cost estimates are primarily based on a cost database or recently quoted costs, with factors applied for minor cost elements.

Table 26-27 presents a summary basis of estimate for the various construction components of the site infrastructure estimates.

Table 26-27: Surface Construction Basis of Estimate

Commodity	Basis
Access Roads	Material take-offs developed from preliminary 3D model Database unit rates for earthworks.
Bulk Earthworks	Material take-offs for surface works developed from preliminary 3D model. Database unit rates for earthworks.
Concrete	Material take-offs measured in neat quantities and quoted rates from multiple, local subcontractors.
Pre-Engineered Buildings, modular buildings and warehouses.	Buildings costed with recent database quotations from similar projects, inclusive of lighting, grounding, electrical, and plumbing systems.
Mechanical Equipment	Database rates were used for the supply and install of mechanical equipment.
Piping	Material take-offs for major pipelines based on 2d drawings and factored. Database rates were used for the supply and install of piping.
Electrical and Instrumentation	Major electrical equipment list prepared for mine facilities, and power distribution estimated from 2 d drawings. Major equipment and cabling utilizing database rates based on recent quotations for other projects.
Power Transmission Line and Major Sub-stations	Quantities developed based on general arrangements and site layouts. A combination of quoted and database costs applied from similar projects utilizing an experienced electrical contractor in the region.

Source: JDS (2019)

The process plant and existing site infrastructure will be used as described in the FS without additional costs incurred.

26.7.4.1 Surface Construction Sustaining Capital

There are no surface sustaining capital requirements, due primarily to the short duration of this project.

26.7.4.2 Indirect Costs

Indirect costs are classified as costs not directly accountable to a specific cost object. Table 26-28 presents the subjects and basis for the indirect costs within the capital estimate.

Table 26-28: Indirect Costs

Commodity	Basis
On Site Contract Services	Construction equipment based on estimated durations and database rates from recent quotations factored for similar work.
Contractor Field Indirects	Estimated by first principles, and including the following items:
	Time based cost allowance for general construction site services (temporary power, heating and hoarding, contractor support, etc.) applied against the surface construction schedule
	Construction offices and temp facilities
	General Services including Surveying and Quality Assurance
	Communications
	Contractor facilities and related cost
	Construction team facilities, fuel
Freight and Logistics	Factored (6%) for freight and logistics related to the materials and equipment required. Factor excludes mining equipment as prices quoted include delivery to site.
Capital Spares	Based factored (2%) of equipment.
Start-up and Commissioning	An allowance for contractor assistance and testing of the power distribution infrastructure.
First Fills	Based on requirements determined by engineering and database pricing.

Source: JDS (2019)

26.7.4.3 EPCM

Estimates for engineering and procurement were based on deliverables for the mining engineering, and factored for the surface construction activities. A mine engineering team and duration was estimated by JDS for the detailed mine engineering and procurement. The surface construction costs were factored at 7% for the engineering and procurement activities for surface infrastructure.

The staffing plan was built up based on the development schedule for construction management, project controls, and contract administration. Costs are based on a simple EPCM execution strategy to accomplish the surface works construction in 4 months. A schedule of rates was applied against a staffing plan aligned with the construction duration period.

26.7.4.4 Owner Cost

Owner costs are capitalized in the initial capital costs during the construction phase. Owner costs are for a 4 month period prior to the start of operations. Any Owner costs prior to this are assumed to be within the Owner approved budget expenses and are considered sunk costs.

The Owner costs consist of:

- Pre-Production G&A and Site Services Labour; and
- Pre-production G&A expenses.

26.7.4.5 Closure Cost Estimate

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine. Activities for closure are only for the items constructed for the Fork, Russet South, and Wedge deposits, including:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMF and MRMF;
- Removal of all fixed underground equipment;
- Closure of the underground mine portals and ventilation raises;
- Power transmission line and substation removal; and
- Re-vegetation and seeding.

The majority of closure costs are incurred immediately following completion of operations (Year 5).

26.7.4.6 Cost Contingency

Contingency has been applied to the estimate determined by the estimate class of the sub-categories. An overall contingency of 15% was applied to the pre-production CAPEX, which resulted in approximately 13% of direct, indirect, and Owner costs. Where additional contingency was deemed necessary, based on the level of confidence for that sub-category, a growth allowance was added to those specific items. The overall contingency is applied after the growth allowance has been added.

26.7.4.7 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in Project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;

- Any Project sunk costs (studies, exploration programs, etc.);
- Closure bonding; and
- Escalation cost.

26.8 Operating Cost Estimate

The operating cost (OPEX) estimates are based on combination of experiential judgement, reference projects, budgetary quotes and factors as appropriate with a PEA study. Cost are a build-up of the summations of all three mines.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies.

All operating costs are in Canadian Dollars. No allowance for inflation has been applied. No operating cost contingency provision has been included in the estimate.

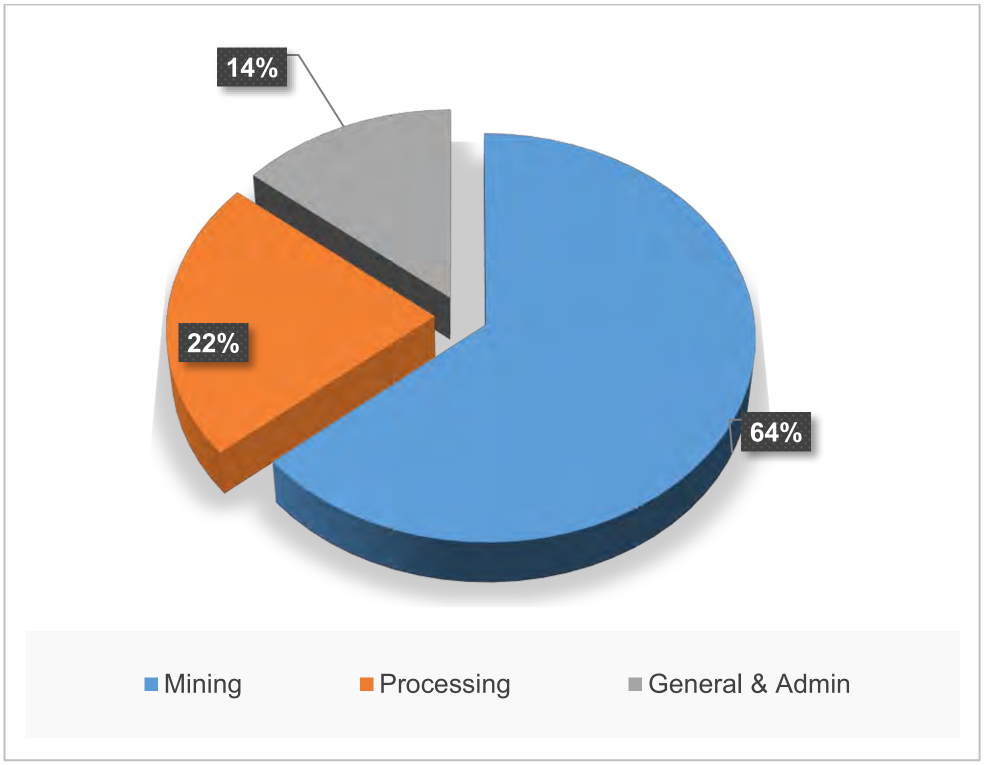
Total LOM operating cost amount to $153.7 M or an average unit cost of $144.60 per mined tonne. The LOM costs are summarized in Table 26-29 and Figure 26-33.

Table 26-29: LOM Total Operating Cost Estimates

Description	Avg Annual (M$)	$/t processed	LOM (M$)
UG Mining	26.7	92.30	98.1
Processing	9.2	32.00	34.0
G&A	5.9	20.30	21.6
Total Operating Costs	**42.8**	**144.60**	**153.7**

Source: JDS (2019)

Figure 26-33: Operating Cost Distribution



Source: JDS (2019)

The main operating cost component assumptions are shown in Table 26-30.

Table 26-30: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.100
Overall Power Consumption (all facilities)	kWh/t processed	9.87
Diesel Cost (delivered)	$/litre	1.072
LOM Average Operations Workforce	employees	260

Source: JDS (2019)

All operating costs for the PEA have been estimated against the development schedule to support the economic cash flow model.

26.8.1 Mine Operating Costs

The total combined mine operating cost between all three mines is broken out by cost centre in Table 26-31.

Table 26-31: LOM Mining Operating Costs Estimates

Description	Total ($M)	Average Unit Cost ($/t milled)
OPEX Development	17.2	16.20
OPEX Cut and Fill	10.8	10.20
OPEX Longhole	27.4	25.72
OPEX Uppers	3.2	3.00
Mine Maintenance	17.4	16.34
Mine General	22.2	20.84
Total	**98.1**	**92.30**

Source: JDS (2019)

26.8.2 Processing Operating Cost

Processing operating costs were estimated to include all gold and silver recovery activities to produce unrefined gold doré on-site. The crushing and process plants are designed for a throughput of 800 t/d for the 4 year mine life of the deposits. Labour rates and benefit loadings are based on information supplied by JDS and Pure Gold. All reagent cost estimates are detailed in Section 22.4.4. The process operating costs are summarized in Table 26-32.

Table 26-32: Process Operating Costs

Processing Category	Unit Cost (CAD$/t processed)	LOM Cost (CADM$)
Labour	16.41	17.5
Power	5.57	5.93
Maintenance and Consumables	10.02	10.65
Total Processing Costs	**32.00**	**34.03**

Source: JDS (2019)

26.8.3 General and Administration Cost

General and administration costs are summarized in Table 26-33.

Table 26-33: LOM General and Administration Operating Cost Estimates

Area	Average $ M/year	Life of Mine $ M	$/t processed
G&A Labour	2.3	9.8	8.7
G&A On-Site Operations	2.9	11.7	11.0
Employee Travel	0.2	0.9	0.7
Total G&A	5.4	22.4	20.3

Source: JDS (2019)

The G&A labour, services and expenses have been prepared as described in Section 22.5.

26.9 Economic Analysis

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

An engineering economic model was developed to estimate annual cash flows and sensitivities of the PEA. Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. Tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 26.3.17 and Section 26.3.18 of this report (presented in 2019 dollars). The economic analysis has been run with no inflation (constant dollar basis).

26.9.1 Summary of Results

The summary of the mine plan and payable metals produced is outlined in Table 26-34.

Table 26-34: Life of Mine (LOM) Summary

Parameter	Unit	Value
Mine Life	Years	3.7
Resource Mined	kt	1,063
Gold (Au) Grade	g/t	6.40
Processing Rate	ktpd	0.8
Gold (Au) Payable	koz	210
	koz/a	57

Source: JDS (2019)

Other economic factors include the following:

- Discount rate of 5%;
- Nominal 2019 dollars;

- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing / incoming payment;
- Working capital calculated as one month of operating costs (mining, processing, site services and G&A) in Year 1;
- Results are presented on 100% ownership;
- No management fees or financing costs (equity fund-raising was assumed); and
- The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 26-35 outlines the metal prices and exchange rate assumptions used in the economic analysis. The gold price selected was based on the 9-month average as of December 2018 and is in line with recently released comparable Technical Reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the Project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 26-35: Metal Prices and Exchange Rates

Parameter	Unit	Value
Gold (Au) Price	US$/oz	1,275
FX Rate	CDN$:US$	0.75

Source: JDS (2019)

26.9.2 Assumptions

Mine revenue is derived from the sale of doré bars into the international marketplace. No contractual arrangements for refining currently exist. Table 26-36 indicates the NSR parameters that were used in the economic analysis.

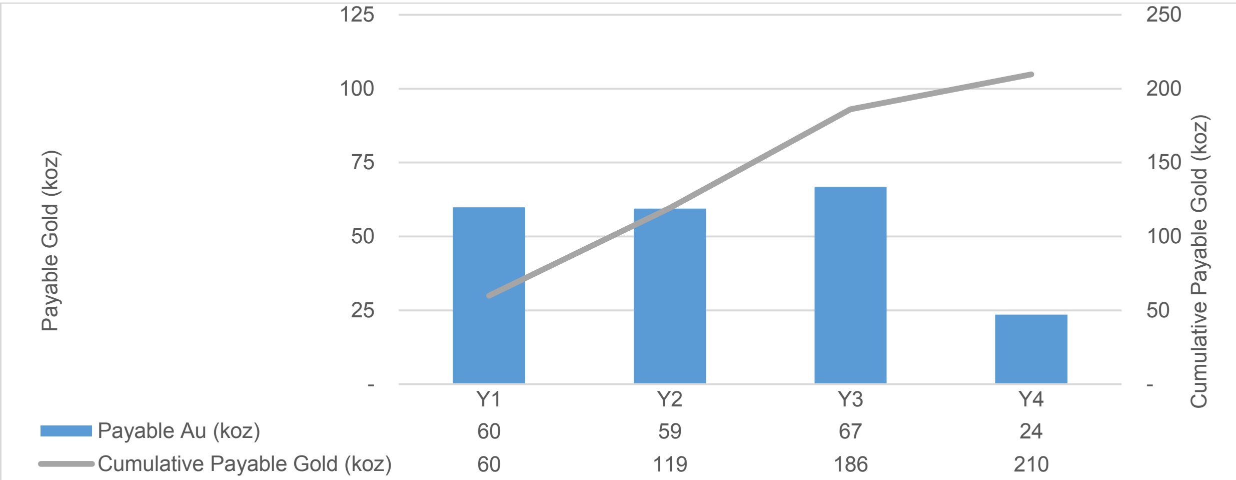
Table 26-36: NSR Parameters

Parameter	Unit	Value
Gold (Au) Recovery	%	95.8
Gold (Au) Payable	%	99.97
Gold (Au) Refining Charge	US$/pay oz	0.38
Transportation	US$/pay oz	1.35
Royalties – Fork and Wedge Deposits	%	-
Royalties – Russet South Deposits (to total CDN$2M)	%	2.0

Source: JDS (2019)

Figure 26-34 shows the grade and the amount of gold (Au) recovered during the mine life. A total of 210 koz of gold is projected to be produced over the life of mine.

Figure 26-34: LOM Payable Gold



Source: JDS (2019)

26.9.3 Taxes

The PEA has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the potential Project economics. A tax model was prepared by Wentworth Taylor, an independent tax consultant, and reviewed by JDS and Pure Gold personnel. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- Federal Income Tax: 15%;
- Ontario Mineral Tax: 10%;
- Capital cost allowance applied on units of production basis and at specific rates in the tax act; and

Total taxes for the Project amount to $32 M.

26.9.4 Royalties

The Mineral Resource at the Russet South deposit evaluated in the PEA is subject to a 2% NSR royalty capped at maximum of $2 M. No royalties are applicable to the resources at the Fork and Wedge deposits

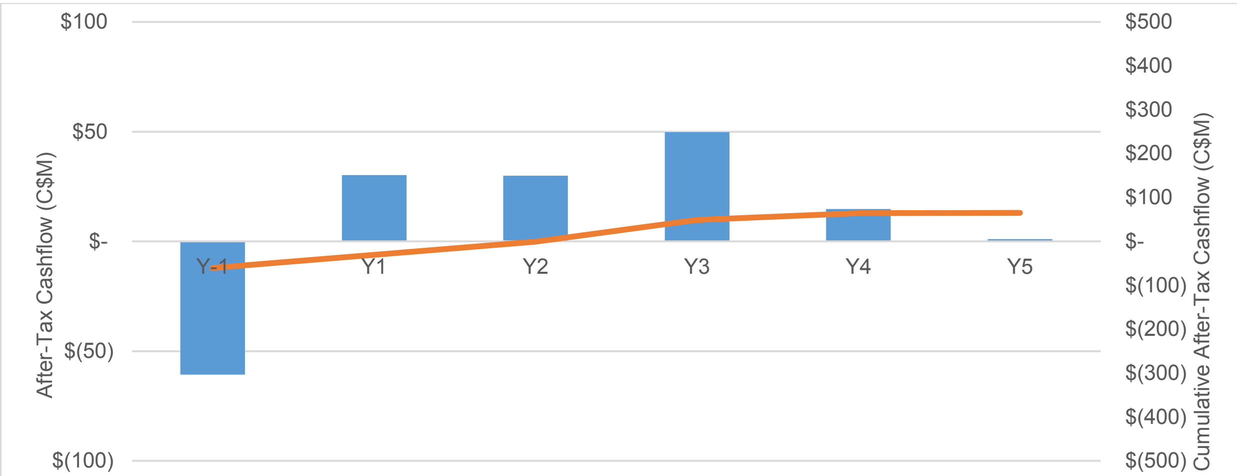
26.9.5 Results

The Madsen PEA has a post-tax IRR of 39% and a net present value using a 5% discount rate ($NPV_{5\%}$) of $51 M. Figure 26-35 shows the projected cash flows, and Table 26-37 summarizes the economic results of the PEA.

The post-tax break-even gold price is approximately US$916/oz, based on the LOM plan presented herein. This is the gold price at which the Project NPV @ 0% discount rate is zero.

The life of mine all-in sustaining cost (AISC) is US$712/oz. The straight AISC cost is calculated by adding the refining, transport, royalty, operating, and sustaining and closure costs together and dividing by the total payable ounces of gold.

Figure 26-35: Annual After-Tax Cash Flow



Source: JDS (2019)

Table 26-37: Summary of Results

Summary of Results	Unit	Value
AISC*	US$/oz	712
Capital Costs		
Pre-Production Capital	M$	50
Pre-Production Contingency	M$	7
Total Pre-Production Capital	**M$**	**57**
Sustaining and Closure Capital	M$	43
Sustaining and Closure Contingency	M$	0
Total Sustaining and Closure Capital	M$	43
Total Capital Costs Incl. Contingency	**M$**	**101**
Working Capital	M$	3
Pre-Tax Cash Flow	LOM M$	100
	M$/a	27
Taxes	LOM M$	32
After-Tax Cash Flow	LOM M$	68
	M$/a	19
Economic Results		
Pre-Tax $NPV_{5\%}$	**M$**	**79**
Pre-Tax IRR	**%**	**57**
Pre-Tax Payback	**Years**	**1.6**
After-Tax $NPV_{5\%}$	**M$**	**51**
After-Tax IRR	**%**	**39**
After-Tax Payback	**Years**	**2.0**

*All-in Sustaining Cost is calculated as: (Refining & shipping costs + royalties+ operating costs + sustaining and closure capital) / payable gold ounces.

Source: JDS (2019)

26.9.6 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -15% to +15%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM. For instance, the metal prices were evaluated at a +/- 15% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

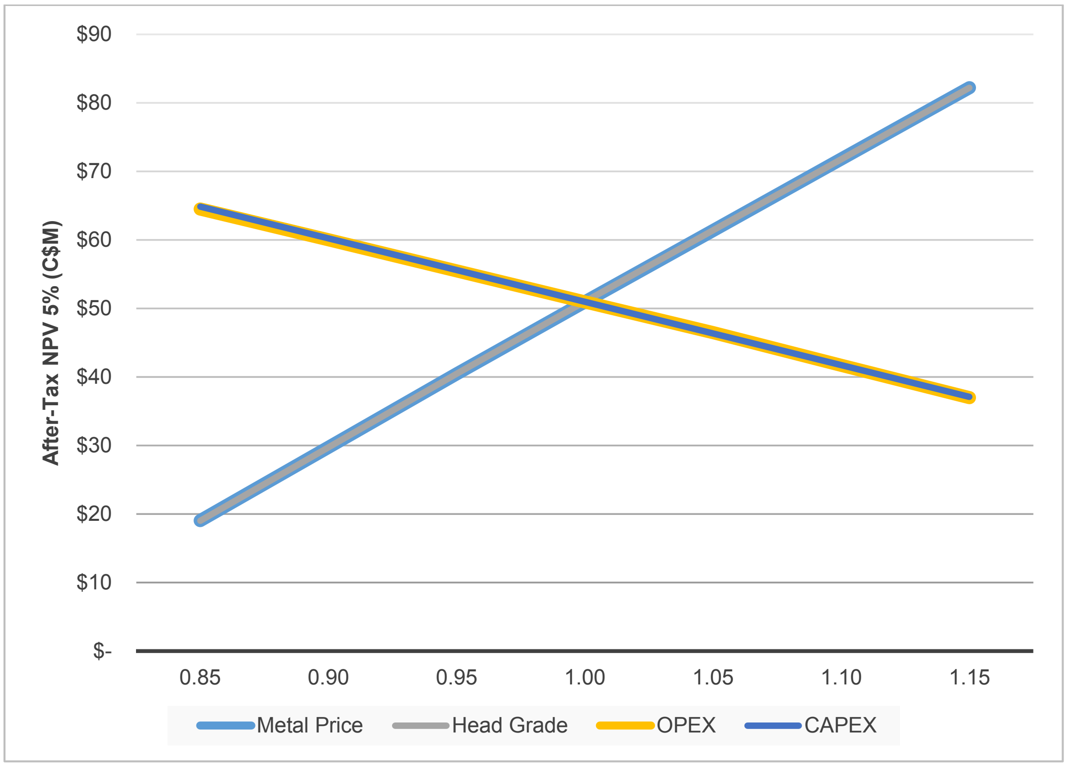
The analysis revealed that the PEA is most sensitive to metal prices and head grade. The PEA showed the least sensitivity to capital costs. Table 26-38 and Figure 26-36 show the results of the sensitivity tests, while Table 26-39 shows the NPV at various discount rates.

Table 26-38: Pre-Tax and After-Tax Sensitivity Results on NPV @ 5%

Variable	After-Tax $NPV_{5\%}$ (M$)			Pre-Tax $NPV_{5\%}$ (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Metal Price	19	51	82	33	79	124
Mill Head Grade	19	51	82	33	79	124
OPEX	65	51	37	98	79	59
CAPEX	65	51	37	93	79	65

Source: JDS (2019)

Figure 26-36: Post-Tax $NPV_{5\%}$ Sensitivity



Source: JDS (2019)

Table 26-39: Project NPV at Various Discount Rates

Discount Rate (%)	Pre-Tax NPV (M$)	Post-Tax NPV (M$)
0	100	68
5	79	51
6	75	48
7	72	45
8	68	43
10	62	38
12	56	33

Source: JDS (2019)

The economic cash flow model for the Project is illustrated in Table 26-40.

Table 26-40: PEA Economic Model

Item	Unit	Pre-Production	Production Total	Life of Mine Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7
KEY PARAMETERS													
Metal Prices													
Gold (Au)	US$/oz	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275	1,275
Exchange Rates													
F/X	USD:CAD	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
MINING													
Production													
Resource Mined	ktonnes	-	1,063	1,063	-	-	288	288	288	199	-	-	-
Waste Mined	ktonnes	351	645	996	-	351	352	252	40	2	-	-	-
Total Mined	ktonnes	351	1,709	2,059	-	351	640	540	328	201	-	-	-
Operating Days	days	365	1,344	1,709	-	365	365	365	365	249	-	-	-
Average Mining Rate	tpd	-	791	622	-	-	789	789	789	800	-	-	-
MINERAL PROCESSING													
Resource Processed	ktonnes	-	1,063	1,063			288	288	288	199	-	-	-
Head Grade - Au	g/t	-	6.40	6.40			6.75	6.70	7.53	3.83	-	-	-
Contained - Au	koz	-	219	219			62	62	70	25	-	-	-
Fork - Tonnes	*ktonnes*	-	*357*	*357*			*47*	*110*	*119*	*81*	-	-	-
Fork - Grade	*g/t*	-	*4.50*	*4.50*			*4.83*	*4.25*	*5.10*	*3.73*	-	-	-
Fork - Ounces	*koz*	-	*52*	*52*			*7*	*15*	*20*	*10*	-	-	-
Wedge - Tonnes	*ktonnes*	-	*531*	*531*			*181*	*120*	*120*	*110*	-	-	-
Wedge - Grade	*g/t*	-	*7.30*	*7.30*			*6.67*	*8.92*	*10.01*	*3.64*	-	-	-
Wedge - Ounces	*koz*	-	*125*	*125*			*39*	*34*	*39*	*13*	-	-	-
Russet - Tonnes	*ktonnes*	-	*175*	*175*			*59*	*58*	*49*	*8*	-	-	-
Russet - Grade	*g/t*	-	*7.55*	*7.55*			*8.53*	*6.76*	*7.36*	*7.25*	-	-	-
Russet - Ounces	*koz*	-	*42*	*42*			*16*	*13*	*12*	*2*	-	-	-
Operating Days	days	-	1,344	1,344			365	365	365	249	-	-	-
Average Plant Throughput	tpd	-	791	791			789	789	789	800	-	-	-
RECOVERED METAL													
Au Recovery	%	0.0%	95.8%	95.8%	0.0%	0.0%	95.8%	95.8%	95.8%	95.8%	0.0%	0.0%	0.0%
Au Recovered	koz	-	210	210	-	-	60	59	67	24	-	-	-
SALES & NSR													
Payable Metals													
Au Payable	%	0.00%	99.97%	99.97%	0.00%	0.00%	99.97%	99.97%	99.97%	99.97%	0.00%	0.00%	0.00%
Au Payable	koz	-	210	210	-	-	60	59	67	24	-	-	-
Payable Value	US$M	-	267.2	267.2	-	-	76.3	75.8	85.1	30.0	-	-	-
Payable Value	C$M	-	356.3	356.3	-	-	101.7	101.1	113.5	40.0	-	-	-
Refining Charges													
Au Refining Unit Charge	CAD$/oz recovered	-	0.50	0.50	-	-	0.50	0.50	0.50	0.50	-	-	-
Au Refining Costs	US$M	-	0.1	0.1	-	-	0.0	0.0	0.0	0.0	-	-	-
Au Refining Costs	C$M	-	0.1	0.1	-	-	0.0	0.0	0.0	0.0	-	-	-
Transport Charges													
Shipment Unit Cost	US$/shipment	-	6,800	6,800	-	-	6,800	6,800	6,800	6,800	-	-	-
Shipments	#	-	44	44	-	-	12	12	12	8	-	-	-
Additional cost by value	US$/US$1000 value	-	0.09	0.09	-	-	0.09	0.09	0.09	0.09	-	-	-
Additional cost per by weight	US$/kg	-	6.50	6.50	-	-	6.50	6.50	6.50	6.50	-	-	-
Transportation Cost	US$M	-	0.4	0.4	-	-	0.1	0.1	0.1	0.1	-	-	-
Transportation Cost	C$M	-	0.5	0.5	-	-	0.1	0.1	0.1	0.1	-	-	-
Net Smelter Return (before Royalties)													
Net Smelter Return	**US$M**	**-**	**266.8**	**266.8**	**-**	**-**	**76.2**	**75.7**	**85.0**	**29.9**	**-**	**-**	**-**
(before Royalties)	**C$M**	**-**	**355.7**	**355.7**	**-**	**-**	**101.6**	**100.9**	**113.3**	**39.9**	**-**	**-**	**-**

Item	Unit	Pre-Production	Production Total	Life of Mine Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Royalties													
Royalty Rate - Canhorn Mining (Russet) - max C$1.0M	% of NSR	0.0%	1.0%	1.0%	0.0%	0.0%	1.0%	1.0%	1.0%	1.0%	0.0%	0.0%	0.0%
Royalty Rate - Franco Nevada (Russet) - max C$1.0M	% of NSR	0.0%	1.0%	1.0%	0.0%	0.0%	1.0%	1.0%	1.0%	1.0%	0.0%	0.0%	0.0%
Applicable to % of Gold (Russet)	%	0.0%	19.4%	19.4%	0.0%	0.0%	26.1%	20.3%	16.7%	8.0%	0.0%	0.0%	0.0%
Royalties Paid	US$M	-	1.0	1.0	-	-	0.4	0.3	0.3	0.0	-	-	-
Royalties Paid	C$M	-	1.4	1.4	-	-	0.5	0.4	0.4	0.1	-	-	-
Net Smelter Return (after Royalties)													
Net Smelter Return (after Royalties)	**US$M**	**-**	**265.8**	**265.8**	**-**	**-**	**75.8**	**75.4**	**84.7**	**29.9**	**-**	**-**	**-**
	US/t	-	249.91	249.91	-	-	263.16	261.59	294.17	149.90	-	-	-
	C$M	**-**	**354.3**	**354.3**	**-**	**-**	**101.1**	**100.5**	**113.0**	**39.8**	**-**	**-**	**-**
	C$/t	-	333.22	333.22	-	-	350.88	348.78	392.23	199.86	-	-	-
OPERATING COSTS													
Mining	C$/t resource mined	-	92.29	92.29	-	-	92.11	94.99	97.46	81.16	-	-	-
	C$M	-	98.1	98.1	-	-	26.5	27.4	28.1	16.2	-	-	-
Re-handle	C$/t re-handled												
	C$M												
Equipment Lease	C$/t processed												
	C$M												
Processing	C$/t processed	-	32.00	32.00	-	-	31.64	31.63	31.64	33.60	-	-	-
	C$M	-	34.0	34.0	-	-	9.1	9.1	9.1	6.7	-	-	-
General & Administration	C$/t processed	-	20.27	20.27	-	-	19.38	19.69	19.87	22.98	-	-	-
	C$M	-	21.6	21.6	-	-	5.6	5.7	5.7	4.6	-	-	-
Total Operating Costs	**C$/t processed**	**-**	**144.56**	**144.56**	**-**	**-**	**143.13**	**146.31**	**148.97**	**137.74**	**-**	**-**	**-**
	C$M	**-**	**153.7**	**153.7**	**-**	**-**	**41.2**	**42.2**	**42.9**	**27.4**	**-**	**-**	**-**
INCOME													
Net Operating Cash Flow	**C$M**	**-**	**200.6**	**200.6**	**-**	**-**	**59.8**	**58.3**	**70.1**	**12.4**	**-**	**-**	**-**
	C$/t processed	**-**	**188.65**	**188.65**	**-**	**-**	**207.74**	**202.47**	**243.26**	**62.12**	**-**	**-**	**-**
CAPITAL COSTS													
Mining	C$M	42.5	42.1	84.6	-	42.5	22.9	16.6	2.5	0.0	-	-	-
Site Development	C$M	1.3	-	1.3	-	1.3	-	-	-	-	-	-	-
Mineral Processing	C$M	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Management	C$M	0.9	-	0.9	-	0.9	-	-	-	-	-	-	-
Site Services	C$M	3.0	-	3.0	-	3.0	-	-	-	-	-	-	-
Closure	C$M	-	1.4	1.4	-	-	-	-	-	-	1.4	-	-
Salvage	C$M	-	- 0.2	- 0.2	-	-	-	-	-	-	- 0.2	-	-
Indirects	C$M	0.6	-	0.6	-	0.6	-	-	-	-	-	-	-
EPCM	C$M	1.1	-	1.1	-	1.1	-	-	-	-	-	-	-
Owner's Costs	C$M	0.8	-	0.8	-	0.8	-	-	-	-	-	-	-
Subtotal - Pre-Contingency	**C$M**	**50.2**	**43.3**	**93.5**	**-**	**50.2**	**22.9**	**16.6**	**2.5**	**0.0**	**1.2**	**-**	**-**
Contingency %	%	14.1%	0.0%	7.6%	0.0%	14.1%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Contingency	C$M	7.1	-	7.1	-	7.1	-	-	-	-	-	-	-
Total Capital Costs	**C$M**	**57.2**	**43.3**	**100.6**	**-**	**57.2**	**22.9**	**16.6**	**2.5**	**0.0**	**1.2**	**-**	**-**
	C$/t processed	**-**	**40.74**	**94.57**	**-**	**-**	**79.63**	**57.62**	**8.84**	**0.03**	**-**	**-**	**-**
WORKING CAPITAL													
Working Capital & Reserves Accounts													
Working Capital	C$M	3.4	- 3.4	-		3.4				- 3.4			
CASH FLOWS													
Pre-Tax													
Net Pre-Tax Free Cash Flow	**C$M**	**- 60.7**	**160.7**	**100.0**	**-**	**- 60.7**	**36.9**	**41.7**	**67.5**	**15.8**	**- 1.2**	**-**	**-**
Cumulative Pre-Tax Free Cash Flow	**C$M**			**100.0**	**-**	**- 60.7**	**- 23.8**	**18.0**	**85.5**	**101.3**	**100.0**	**100.0**	**100.0**
Post-Tax													
Income Taxes	C$M	-	23.4	23.4	-	-	6.7	9.2	12.6	0.3	- 2.2	- 2.5	- 0.6
Ontario Mining Tax	C$M	-	8.5	8.5	-	-	-	2.7	5.1	0.7	-	-	-
Net After-Tax Free Cash Flow	**C$M**	**- 60.7**	**128.9**	**68.2**	**-**	**- 60.7**	**30.2**	**29.9**	**49.8**	**14.8**	**1.0**	**2.5**	**0.6**
Cumulative After-Tax Free Cash Flow	**C$M**			**68.2**	**-**	**- 60.7**	**- 30.5**	**- 0.6**	**49.3**	**64.0**	**65.0**	**67.5**	**68.2**

Source: JDS (2019)

26.10 Conclusions

It's the conclusion of the QPs that the Fork, Russet South, and Wedge Deposit PEA summarized in this technical report section contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

The Fork, Russet South, and Wedge Deposits contains substantial gold resources that can be mined by underground methods and recovered with the processing plant defined in the Feasibility Study.

Based on the assumptions used for this preliminary evaluation, the project is considered to be economic and should proceed to the pre-feasibility (PFS).

There is also a likelihood of improving the project economics by identifying additional mineral resources within the deposits that may justify increased mine production and/or extend the mine life. Further study and/or design work may identify additional opportunities to improve project economics.

26.10.1 Risks and Opportunities

The most significant risks are summarized below:

- **Surface Geotechnical Conditions** - There is a potential for additional foundation improvement measures to be required at the Tailings Pond Dams to meet the stability Factors of Safety. There is a potential for additional overburden excavation at the critical stability footprint area of the MRMF which could lead to additional associated costs. This risk can be managed by completing additional geotechnical investigations and studies on those areas identified in the FS.
- **Underground Water Flow** – There a potential that the predicted water inflows at the three PEA mines are higher than expected. This risk can be managed by adding additional dewatering capacity, albeit at an increased operating cost.

The most significant opportunities are summarized below:

- **Scheduling** – Mining the Fork, Russet South, and Wedge deposits earlier in the project life, while the Madsen mine is in operation could provide additional mill feed above 800 t/d and could potentially lead to an expanded operation resulting in improved economics.
- **Power Unit Cost** - Other comparable Hydro One customers in the region have been able to achieve significant reductions in their electricity costs by closely managing their electrical load by applying for Northern Industrial Rebate Program (NIER), reduction in Global Adjustment (GA) Payments and Shifting Peak Demand to Off-Peak Hours. These three mechanism could reduce the unit power rate and ultimately operating costs.
- **Resource Conversion & Expansion** - Through additional exploration drilling, there is potential opportunity to convert Inferred Resources to Indicated Resources and to discover additional mineralized zones. If successful this could increase production rates or extend mine life. There is no certainty that all or any part of the Inferred Resources will be converted to Indicated Mineral Resources or converted into Mineral Reserves.

- **Ore Sorting** - The deposit may be amenable to ore sorting. If successful, ore sorting would remove external and planned dilution increasing the grade to the process plant. Further physical properties study is required.

26.10.2 Recommendations

It is recommended that the Fork, Russet South, and Wedge deposits proceed to the Pre-feasibility Study stage in line with Pure Gold's desire to advance the project. It is also recommended that environmental work and permitting continue as needed to support Pure Gold's project development plans and the work programs defined below:

It is estimated that a Pre-feasibility Study and supporting field work would cost approximately $4.8 M. A breakdown of the key components of the next study phase is as follows in Table 26-41.

Table 26-41: Cost Estimate to Advance to PFS Stage

Item	Estimated Cost ($M)	Comment
Additional Resource Drilling	2.0	Conversion of Indicated to Measured resources. Drilling will include holes combined for resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.5	Comminution, variability testing, tailings dewatering, concentrate filtration, mineralogy, minor element analysis.
Geochemistry	0.5	Acid Base Accounting (ABA) tests and humidity cell testing to determine acid generating potential of rock and tailings.
Waste & Water Site Investigation	0.1	Site investigation drilling, sampling and lab testing.
Geotechnical, Hydrology & Hydrogeology	0.5	Drilling, sampling, logging, test pitting, lab tests, etc.
Engineering	0.7	PFS-level mine and infrastructure design, cost estimation, scheduling & economic analysis.
Environment	0.5	Baseline investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology.
Total	***4.8***	Excludes corporate overheads and future permitting activities.

Source: JDS (2019)

27 Interpretations and Conclusions

27.1 Risks

It is the conclusion of the Qualified Persons that the Feasibility Study summarized in this technical report contains adequate detail and information to support a feasibility level analysis. Standard industry practices, equipment and design methods were used in this Feasibility Study and except for those outlined in this section, the report authors are unaware of any unusual or significant risks or uncertainties that would affect project reliability or confidence based on the data and information made available.

For these reasons, the path going forward should continue to focus on obtaining permits to re-start the operation, while concurrently advancing key activities that will reduce and de-risk the project execution schedule.

Risk is present in any development project. Feasibility level engineering formulates design and engineering solutions to reduce that risk common to every mining project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, environmental and social impact, political risks, schedule and cost overruns, and labour sourcing. JDS is of the opinion that these risks have been clearly identified and mitigation measures have been considered.

Table 27-1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches.

Table 27-1: Main Project Risks

Risk	Explanation / Potential Impact	Possible Risk Mitigation
Surface Geotechnical Conditions	Geotechnical studies were carried out to investigate the foundation conditions in the TMF and MRMF. There is a potential for additional foundation improvement measures to be carried out at the Tailings Pond Dams to meet the required stability Factors of Safety. There is a potential for additional overburden excavation at the critical stability footprint area of the MRMF which could lead to additional associated costs.	This risk can be managed by completing additional geotechnical investigations and studies on those areas identified in the FS.
Rock Stress	Greater than anticipated stress impacts on stopes in McVeigh/Austin LH areas. Local stress concentrations may be higher than anticipated near historical workings.	Localized destress drilling and blasting as well as further numerical modeling.
8 Zone Ground Conditions	More significant ground squeezing than anticipated in 8 Zone. Squeezing or closure around drifts may be more dramatic or frequent than anticipated requiring slower mining rates and increased ground support and maintenance requirements.	Additional geotechnical drilling and laboratory testing should be carried out as soon as possible within the 8 Zone hanging wall and mineralized zone.
Process Plant Refurbishment	The existing mill may require more rework and refurbishment of electrical and plumbing systems. Until the mill is re-energized, there is some uncertainty with regards to the mill electrical components including the motors, and valves and piping. This would require additional equipment, material and installation labour to bring the mill into functional capacity.	Cost allowances have been made for refurbishing the mill motors and pumps, and some allowances have been made in the installation costs. Additional inspections and testing prior to construction and re-starting.
Power Load & Timing of Line Expansion	Adequate electricity supply from Hydro One is a concern in Northwestern Ontario and while capacity study completed by Hydro One shows adequate power will be available for the initial years, the capacity of 7MW is very close to the estimated operating demand load. The future upgrade of the Pickle lake section of transmission line by 2022, and additional power capacity required for the hoisting remains an area of uncertainty for the Madsen Gold Project. Additional power from other sources, such as generators, could add additional operating costs.	Additional power supply from generators to mitigate power shortages.
Shaft Conditions	Blockages and/or conditions in the shaft are worse than anticipated which could increase rehabilitation costs and/or delay getting the shaft operational.	Delay in shaft hoisting would be mitigated by increased truck haulage albeit at a higher operating cost. An increased dewatering rate would allow additional time for assessment and mitigation/rehabilitation.
Existing Backfill Failure	Saturated backfill from existing stopes could become unstable and fail. Backfill failures pose a health and safety risk, could increase operating costs to clean-up and cause mine schedule delays.	Increased dewatering ahead of when existing levels are required to allow old stopes to drain/dry. Assessments with drones and/or probe holes from safe areas well in advance of accessing or mining activities.
Underground Water Inflow Rates	Higher than anticipated underground water inflow rates could increase operating costs and cause mine schedule delays.	Add additional pumping capacity.
Ore Body Complexity	The complexity of the ore body could potentially lead to increased mining dilution. Grade control and proper mining execution will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs.	Definition drilling in advance of mining to assist stope design. Evaluate the potential for ore sorting.
Tailings and Un-mineralized Mine Rock Geochemistry	Tailings and un-mineralized mine rock are assumed to be Non-Acid Generating (NAG) due to testwork completed to date and past operations. Additional testwork may be required prior to detail engineering in order to confirm this assumption. If classification is changed to Potentially-Acid Generating (PAG), then increased CAPEX and OPEX may result to mitigate.	Geochemistry testing costs are included in the project budget and should be carried out prior to detail engineering.
Ability to Attract Experienced / Trained Local Labour	The ability to attract and retain competent, experienced local labour is a key success factor for the Project.	The early search for trained labour as well as competitive salaries and benefits identify, attract and retain critical local personnel.
	High turnover or the lack of appropriate technical staff at the Project could result in difficulties meeting Project goals.	Further opportunity may exist in hiring trained labour from other Red Lake mines.

Source: JDS (2019)

27.2 Opportunities

This Feasibility Study has highlighted several opportunities to increase mine profitability and project economics and reduce identified risks. These are summarized in Table 27-2.

Table 27-2: Main Project Opportunities

Opportunity	Explanation	Potential Benefit
Underground Water Management	Flows from the Madsen Underground Mine are currently being management in the TMF all year round. This results in a large accumulation of water in the TMF, particularly during periods of water release restrictions. A WTP with a large design treatment rate is therefore required to manage the surplus water in the TMF.	Managing the Madsen Mine Underground Dewatering flows outside the TMF would considerably reduce the yearly surplus volume of water to be treated and reduce the design treatment rate of the WTP. This would lead to large savings in Project Capital Costs. Additional water quality studies would need to be carried out to investigate the possibility of releasing the underground flows directly to Derlak Lake.
Power Unit Cost	Other Hydro One customers have been able to achieve significant reductions in their electricity costs by closely managing their electrical load. There are three mechanisms that Pure Gold could use to significantly reduce costs to the 4.5 cents per kWhr range: 1. Apply for the Northern Industrial Rebate Program (NIER) This ENDM program can provide a two cent per kilowatt hour rebate on the program participant's electricity bill. 2. Reduction in Global Adjustment (GA) Payments - This can be achieved by reducing demand during the 5 highest provincial power consumption peaks for the year. Normally these peaks occur during hot summer days due to air conditioning demand in Southern Ontario 3. Shifting Peak Demand to Off-Peak Hours. By reducing a site's peak demand during the period of 7AM to 7PM on weekdays (except holidays) a customer can reduce the Network Service Charge portion of the monthly Delivery Charge (which represents approximately 46%). Note this may be difficult, since they can only run crusher during day shift.	Decreased operating costs.
Underground Fleet Optimization	Fleet requirements remain high from years 1 through to 5, investigate possibility of lowering fleet requirements.	Decreased CAPEX and OPEX.
Mining Lower Grade/Marginal Material	Lower grade material was not included in the mine plan and could be potentially added to the plan during mining.	Increased material mined and processed.
Include Buffer Zone Material	Known high grade material has been modelled in the 5m buffer zone. Including this would result in an increase in stopes.	Increased Mineral Reserves.
Resource Conversion	Through additional infill diamond drilling, there is an opportunity to increase the Indicated Resource Category from Inferred.	Additional gold ounces in the Indicated Resource resulting in additional gold ounces in the Mineral Reserves, which has potential to improve economic results.
Resource Expansion	With additional diamond drilling, there is an opportunity to discover and develop additional zones and extend known zones to increase resources.	Additional gold ounces in the Measured and Indicated Resource resulting in additional gold ounces in the Mining Reserves, which has potential to improve economic results.
Ore Sorting	The deposit may be amenable to ore sorting. If successful, ore sorting would remove external and planned dilution increasing the grade to the process plant.	If ore sorting is implemented in the process design it will result in opportunities to reduce plant size and/or allow for more flexibility in mining method by mitigating potential increased dilution.
Used Equipment	There is an opportunity to source used equipment for the process plant or mine.	Potential reduction in initial capital costs.

Source: JDS (2019)

28 Recommendations

Based on the Feasibility Study results, it's recommended to advance the Project to construction and development, and then production. The recommended development path is to continue efforts in obtaining the required environmental permits to start construction and continue dewatering the underground workings in advance of mine development, while concurrently advancing key activities that will reduce and de-risk the project execution schedule. Associated project risks are deemed manageable; and identified opportunities can provide enhanced economic value.

The project exhibits robust economics with the assumed gold price, currency exchange rates, and consumables pricing. Value engineering and recommended fieldwork should be advanced in conjunction with preparation of permit amendments/applications to de-risk the construction schedule and minimize costs.

From the identified project risks and opportunities, the following activities are noted as critical actions that have the potential to strengthen the project and further reduce risk:

- Complete additional infill diamond drilling to convert more of the Inferred Mineral Resource into Measured and Indicated resources;
- Continue underground dewatering;
- Confirm TMF and MRMF foundation design parameters with additional site investigation, complete TMF foundation liquefaction assessment and optimize the water balance and water management strategy;
- Develop a full closure plan for the TMF and MRMF based on the final design configuration;
- Conduct initial test work on the viability of employing ore sorting technology as a method of rejecting low grade plant feed and increasing head grade in the process plant; and
- Investigate potential for purchasing used equipment to reduce project capital costs.

Table 28-1 provides an estimate for each of the activities recommended above.

Table 28-1: Estimate of Recommended Post-FS Activities

Item	Approximate Cost ($M)
Additional infill drilling	2.0
Continue underground dewatering	0.5
TMF and MRMF site investigation and engineering	0.6
Closure plan development	0.1
Conduct ore sorting test work	0.2
Investigate used equipment	0.1
Total	***3.5***

Source: JDS (2019)

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30 Units of Measure, Abbreviations and Acronyms

Symbol / Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
ACK	apparent coherent kimberlite
ALT	active layer thickness
ALT	active layer thickness
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
C$	dollar (Canadian)
Ca	calcium
cfm	cubic feet per minute
CHP	combined heat and power plant
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
cP	centipoise
Cr	chromium
Cu	copper
d	day
d/a	days per year (annum)

Symbol / Abbreviation	Description
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DGPS	differential global positioning system
DMS	dense media separation
dmt	dry metric ton
DWT	dead weight tonnes
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
EWR	enhanced winter road
FEL	front-end loader
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GPa	gigapascal
gpm	gallons per minute (us)
GSC	geological survey of Canada
GTZ	glacial terrain zone
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
ha	hectare
HG	high grade

Symbol / Abbreviation	Description
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
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
INAC	Indigenous and Northern Affairs Canada
IOL	Inuit owned land
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
kg	kilogram
kg	kilogram
kg/h	kilograms per hour
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
KIM	kimberlitic indicator mineral
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LG	low grade

Symbol / Abbreviation	Description
LGM	last glacial maximum
LOM	life of mine
m	metre
M	million
m/min	metres per minute
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
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
MBP	melt-bearing pyroclasts
mbs	metres below surface
mbsl	metres below 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
mo	month
MPa	megapascal
MRMF	Mine Rock Management Facility
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt

Symbol / Abbreviation	Description
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
Nm^3/h	normal cubic metres per hour
NQ	drill core diameter of 47.6 mm
NRC	natural resources Canada
OP	open pit
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PEA	preliminary economic assessment
PFS	preliminary feasibility study
PGE	platinum group elements
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RC	reverse circulation
RIA	regional Inuit associations
RMR	rock mass rating
ROM	run of mine
rpm	revolutions per minute
RQD	rock quality designation
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SFD	size frequency distribution
SG	specific gravity
st/kg	stones per kilogram
st/t	stones per metric tonne

Symbol / Abbreviation	Description
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year
t/d	tonnes per day
t/h	tonnes per hour
TCR	total core recovery
TFFE	target for further exploration
TMF	tailings management facility
tph	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VEC	valued ecosystem components
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
wk	week
wmt	wet metric ton
µm	microns
µm	micrometre

Scientific Notation	Number Equivalent
1.0E+00	1
1.0E+01	10
1.0E+02	100
1.0E+03	1,000
1.0E+04	10,000
1.0E+05	100,000
1.0E+06	1,000,000
1.0E+07	10,000,000
1.0E+09	1,000,000,000
1.0E+10	10,000,000,000

Appendix A – Qualified Person Forms

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CERTIFICATE OF AUTHOR

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario” with an effective date of February 5, 2019 (the “Technical Report”) prepared for Pure Gold Mining Inc.;

2. I am currently employed as Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

   I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over thirteen years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and technical report writing for mining projects worldwide;

   I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I visited the Madsen Gold Project site on December 7, 2017;

5. I am responsible for Sections 1.1, 1.2, 1.7, 1.8, 1.11-1.15, 2-3, 4.3, 4.4, 12.7, 15, 16 (except 16.3, 16.8), 19-21, 22 (except 22.4), 24, 26 (except 26.2, 26.3.1, 26.4, 26.5, 26.8.2), 27-30 of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Michael Makarenko, P. Eng."

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CERTIFICATE OF AUTHOR

I, Michael Levy, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario" with an effective date of February 5, 2019 (the "Technical Report") prepared for Pure Gold Mining Inc.;
2. I am currently employed as Geotechnical Engineering Manager with JDS Energy & Mining Inc. with an office at Suite 100 – 14143 Denver West Parkway, Golden, Colorado, 80401;
3. I am a Professional Civil Engineer (P.Eng. #2692) registered with the Association of Professional Engineers Yukon and Colorado (P.E. #40268). I am a current member of the Society for Mining, Metallurgy & Exploration (SME) and Metallurgical Engineers (SME) and the American Society of Civil Engineers (ASCE).

   I hold a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since 1999 and have been involved in a numerous mining and civil geotechnical projects across the Americas;
4. I have visited the project on February 7-9, 2017 and March 14-15, 2018;
5. I have had no prior involvement with the property that is the subject of this Technical Report;
6. I am responsible for sections 16.3 and 26.3.1 of the Technical Report;
7. I have read the definition of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association, and past relevant experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
8. I am independent of the Issuer and related companies as defined in Section 1.5 of NI 43-101;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1; and,
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Michael Levy, P. Eng."

Michael Levy, P. Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Kelly McLeod, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario" with an effective date of February 5, 2019 (the "Technical Report") prepared for Pure Gold Mining Inc.;

2. I am currently employed as Process Engineer with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 12 years consulting in the mining industry in metallurgy and process design engineering;

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I visited the Madsen Gold Project site on December;

6. I am responsible for Sections 1.5, 1.9, 12.6, 13, 17, 22.4, 26.2, 26.4, 26.8.2 of the report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have not had prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Kelly McLeod, P. Eng."

Kelly McLeod, P. Eng.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Ruane, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario” with an effective date of February 5, 2019 (the “Technical Report”) prepared for Pure Gold Mining Inc.;
2. I am currently employed as a Senior Engineer with Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of the National University of Ireland, Galway with a B.E. in Civil Engineering (2010). I am a graduate of the University of Strathclyde and the University of Glasgow with a M.Sc. in Geotechnics (2011). I have practiced my profession continuously since 2011. My experience includes tailings and waste and water management for mine developments in Canada, the US, Mexico and Europe.
4. I am registered as a Professional Engineer in good standing with Engineers and Geoscientists of British Columbia in the area of civil engineering (No. 42894).
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Madsen Gold Project site on December 7, 2017 and July 16, 2018;
7. I am responsible for Sections 12.8, 18.9-12, 26.5.8-11 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no involvement with the property that is the subject of this Technical Report;
10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) “Daniel Ruane, P. Eng.”

Daniel Ruane, P. Eng.

Equity Exploration Consultants Ltd.
1510 – 250 Howe Street,
Vancouver, B.C.
V5N 4S2

CERTIFICATE OF AUTHOR

I, Darcy E.L. Baker, P.Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario” with an effective date of February 5, 2019 (the “Technical Report”) prepared for Pure Gold Mining Inc.;
2. I am a consulting geologist and President of Equity Exploration Consultants Ltd., a mining exploration management and consulting company with offices at 1510 – 250 Howe Street, Vancouver, British Columbia, V6C 3R8;
3. I am a graduate of Dalhousie University with an Honours B.Sc. in Geology (1997) and a graduate of the University of Newcastle, Australia with a Ph.D. in Geology (2003). I have practiced my profession continuously since 2003;

   I have worked managing exploration programs focused on identifying and delineating epithermal, porphyry, VMS, orogenic gold, IOCG and other deposits in Alaska, British Columbia, Finland, Mexico, Nevada, Nunavut, Ontario, Quebec and Yukon. Prior to launching a career in mineral exploration, I completed a Ph.D. research project studying the timing and structural relations of orogenic gold deposits in the Archean Pilbara Craton of Western Australia;

   I have completed geological mapping, geochemical sampling, drill targeting, geological modelling, review of historical hard copy plan maps and diamond drill logs and examination of historical and recent drill core at the Madsen Gold Project;

   I am a Registered Professional Geologist in British Columbia (#33448) and Ontario (#2746);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. Since 2014, I have visited the Madsen Gold Project numerous times and my most recent property visit was February 11–15, 2019;
5. I am responsible for Sections 1.3, 1.4, 4 (except 4.3, 4.4), 5-11, 12.1-12.5 and 25 of this report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
7. Prior to 2014, I have had no involvement with the property that is the subject of this Technical Report;
8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and
9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Darcy E.L. Baker, P. Geo."

Darcy E.L. Baker, Ph.D., P. Geo.

GINTO CONSULTING INC.

333 West 17th Street
North Vancouver, B.C.
Canada V7M 1V9
(604) 374-1629

CERTIFICATE OF AUTHOR

I, Marc Jutras, P. Eng., M.A.Sc., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario” with an effective date of February 5, 2019 (the “Technical Report”) prepared for Pure Gold Mining Inc.;

2. I am currently employed as Principal, Mineral Resources with Ginto Consulting Inc. with an office at 333 West 17th Street, North Vancouver, British Columbia, V7M 1V9;

3. I am a graduate of the University of Québec in Chicoutimi in 1983, and hold a Bachelor’s degree in Geological Engineering. I am also a graduate of the Ecole Polytechnique of Montréal in 1989, and hold a Master’s degree of Applied Sciences in Geostatistics;

   Since 1984, I have worked continuously in the field of mineral resource estimation of numerous international exploration projects and mining operations. I have been involved in the evaluation of mineral resources at various levels: early to advanced exploration projects, preliminary studies, preliminary economic assessments, prefeasibility studies, feasibility studies and technical due diligence reviews;

   I am a Registered Professional Engineer with the Engineers and Geoscientists British Columbia (license # 24598) and Engineers and Geoscientists Newfoundland and Labrador (license # 09029). I am also a Registered Engineer with the Quebec Order of Engineers (license # 38380).

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have completed a site inspection of the Madsen Gold property on August 30, 2017. At that time the core logging and sample preparation facilities were visited, as well as the SGS assaying laboratory. Drill hole core of the high-grade and low-grade mineralized zones from the Austin, South Austin, McVeigh, and 8 Zone areas were examined. An underground visit of the McVeigh area was performed from the ramp access down to the 1 level. A brief review of some of the historical sections and plans was also carried out during this visit. Overall, the site inspection was satisfactory;

5. I am responsible for Sections 1.6 and 14 of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Marc Jutras, P. Eng., M.A.Sc."

Marc Jutras, P. Eng., M.A.Sc.

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Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300
jdsmining.ca

CERTIFICATE OF AUTHOR

I, Richard Boehnke, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario" with an effective date of February 5, 2019 (the "Technical Report") prepared for Pure Gold Mining Inc.;

2. I am currently employed as Senior Engineer with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

   I am a Registered Professional Engineer in British Colombia (#30486). I am a graduate of the University of Manitoba with a B.Sc. in Industrial Engineering, 1989. I have practiced my profession continuously since 1989;

   I have worked in technical, operations and management positions in industrial facilities in Canada for over 15 years, and have been working in mining related business for over 12 years. I have been an independent consultant for eight years and have performed project management, field engineering, engineering management, cost estimation, construction management, technical due diligence reviews and technical report writing for mining projects worldwide;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer, vendor, property and related companies applying all of the tests in Section 1.5 of NI 43-101;

3. I visited the Madsen Gold Project site on April 17-19, 2018;

4. I am responsible for Sections 1.10, 18 (except 18.9-18.12), 23, 26.5 (except 26.5.8-11) of this Technical Report;

5. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6. I have had no prior involvement with the property that is the subject of this Technical Report;

7. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

8. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

(Original signed and sealed) "Richard Boehnke, P. Eng."

Richard Boehnke, P. Eng.

CERTIFICATE OF AUTHOR

I, David Stone, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Madsen Gold Project Technical Report, Feasibility Study for the Madsen Deposit and Preliminary Economic Assessment for the Fork, Russet South and Wedge Deposits, Red Lake, Ontario" with an effective date of February 5, 2019 (the "Technical Report") prepared for Pure Gold Mining Inc.
2. I am currently employed as President of MineFill Services, Inc., that is a Washington, USA, domiciled Corporation.
3. I am a graduate of the University of British Columbia with a B.Ap.Sc in Geological Engineering, a Ph.D. in Civil Engineering from Queen's University at Kingston, Ontario, Canada, and an MBA from Queen's University at Kingston, Ontario, Canada.
4. I have practiced my profession for over 30 years and have considerable experience in the preparation of engineering and financial studies for base metal and precious metal projects, including Preliminary Economic Assessments, Preliminary Feasibility Studies and Feasibility Studies.
5. I am a licensed Professional Engineer in Ontario (PEO #90549718) and I am licensed as a Professional Engineer in a number of other Canadian and US jurisdictions.
6. 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 reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
7. I have not visited the subject property.
8. I am responsible for the report content related to backfilling and the backfill plant (portions of Section 16.8)
9. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property.
11. I have read NI 43-101 and NI 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the Effective Date of the Technical Report (February 5, 2019), to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Effective Date: February 5, 2019
Signing Date: March 21, 2019

//Original signed and sealed by David Stone, P.Eng. //

David Stone, P.Eng.