NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project

Prepared for Minnova Corp. (TSXV: MCI)

General Location: PL Gold Mine, near Sherridon, Manitoba, Canada
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LIST OF ABBREVIATIONS

Abbreviation	Definition
ABGM	A and B Global Mining (Pty) Ltd
Au	gold
C$	Canadian dollar
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRM	certified reference material
DFO	Fisheries and Oceans Canada
EAL	Environment Act Licence
FS	feasibility study
g/t	grams per tonne
G&A	general and administrative
ha	hectare
km	kilometre
LCT	lithium–cesium–tantalum
m	metre
MDMER	Metal and Diamond Mining Effluent Regulations
ML065	Mining Lease 065 (also ML65)
MRE	Mineral Resource Estimate
MSFA	metallic screen fire assay
MSO	mineable stope optimizer
Mt	million tonnes
NAD83	North American Datum 1983
NAG	non-acid-generating
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects
NoA	Notice of Alteration
NSR	net smelter return
oz	troy ounce (31.10348 g)
PAG	potentially acid-generating
PEA	preliminary economic assessment
PL	PL Gold Mine / PL Property (formerly Puffy Lake)
Pr.Sci.Nat.	Professional Natural Scientist (SACNASP)
QA/QC	quality assurance / quality control
QP	Qualified Person
RPEEE	reasonable prospects for eventual economic extraction
SACNASP	South African Council for Natural Scientific Professions
t	tonne
TMA	tailings management area
Tpd	tonnes per day
TSXV	TSX Venture Exchange

Abbreviation	Definition
US$	United States dollar
UTM	Universal Transverse Mercator
WRD	waste rock dump

IMPORTANT NOTICE

This technical report was prepared for Minnova Corp. ("Minnova") by A and B Global Mining (Pty) Ltd ("ABGM") in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects and Form 43-101F1. The quality of information, conclusions and estimates contained herein is consistent with the scope of ABGM's services and is based on (i) information available at the time of preparation, (ii) data supplied by Minnova and other sources as described in this report, and (iii) the assumptions, conditions and qualifications set out in this report.

Minnova is permitted to file this report as a technical report with Canadian securities regulatory authorities pursuant to NI 43-101. 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.

This report contains forward-looking information, including the results of a preliminary economic assessment (PEA). Actual results may differ materially from those expressed or implied. The PEA is preliminary in nature and includes Inferred Mineral Resources; readers should refer to the cautionary statements in Item 1 and Item 22.

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1 EXECUTIVE SUMMARY

1.1 Introduction

A and B Global Mining (Pty) Ltd (ABGM) was commissioned by Minnova Corp. (Minnova or the Company; TSXV: MCI) to prepare this Technical Report, a preliminary economic assessment (PEA) of the restart of the PL Gold Mine (historically the Puffy Lake Mine) near Sherridon, Manitoba, Canada, as a combined contractor-mined open pit (OP) and underground (UG) operation feeding the refurbished and expanded PL process plant. The Technical Report has been prepared in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43-101F1. The Mineral Resources are those of the 2026 Mineral Resource Estimate (2026 MRE), effective September 10, 2026, reported using the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014) (CIM 2014 Definition Standards). The Technical Report supports the PEA results disclosed by Minnova in its news release of October 14, 2026. The Effective Date of the Technical Report is **September 10, 2026**.

Cautionary statement. The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. No Mineral Reserves have been estimated. Inferred Mineral Resources make up approximately 32.1% of the PEA plant feed tonnes and 24.6% of the contained gold. Most of the Inferred material is in the underground mine plan; Inferred makes up about 6.9% of the open pit plant feed tonnes.

1.2 Property Description, Ownership, Location & Access

The PL Gold Mine is in northwestern Manitoba, approximately 63 km northeast of Flin Flon and approximately 12 km southeast of the community of Sherridon. The PL deposit is on Mining Lease ML065, near 55°02’05” N latitude and 100°58’52” W longitude. Minnova reports that it holds a 100% interest in the PL Gold Mine.

The PL Property is comprised of Mining Lease ML065 (378 ha) and 20 staked mining claims totalling 3,552 ha, and the contiguous Nokomis Property comprises 27 staked mining claims totalling 566 ha. The PL Property is subject to a sliding-scale net smelter return (NSR) royalty held by Franco-Nevada Corporation since November 1, 2013: 3.0% at gold prices at or above US$1,000/oz, 2.5% from US$750/oz to below US$1,000/oz, and 2.0% below US$750/oz. An underlying 2% NSR royalty applies to the Nokomis Property. Surface rights are held by the Crown.

The property benefits from significant existing infrastructure from past operations, including a processing plant on site with a nominal design capacity of 1,000 tonnes per day.

1.3 History

Gold mineralization at PL has been explored since 1953. Maverick Mountain Resources Ltd. and Pioneer Metals Corporation (Pioneer) defined the deposit between 1980 and 1987. Pioneer then built a 1,000 t/d processing plant and developed a ramp-access underground mine. Milling began in December 1987. The operation was suspended in March 1989 because it did not reach commercial production, and the plant was placed on care and maintenance in April 1989. Approximately 350,000 t were milled, and about 21,000 oz of gold were produced in 1988 and 8,500 oz in 1989. Minnova (then Auriga Gold Corp.) acquired the property from Pioneer in 2010 and has since drilled the deposit in several programs. These include the 2025 to 2026 infill program, which comprised more than 13,500 m in 132 holes. Of these, 126 holes (13,092 m) are in the database used for the 2026 MRE; the remaining six holes are not in the MRE database and will be incorporated in the planned pre-feasibility study (PFS).

Earlier estimates include the 2011 and 2014 P&E Mining Consultants estimates and the October 31, 2017 CSA Global Mineral Resource estimate (L. McGarry), which supported the 2018 Feasibility Study (2018 FS). These earlier estimates, and the 2018 FS Mineral Reserves, have been superseded by the 2026 MRE. They are historical and are not to be relied upon.

1.4 Geological Setting, Mineralization & Deposit Type

The PL deposit lies near the boundary between the Kisseynew and Flin Flon domains of the Paleoproterozoic Trans-Hudson Orogen. It is a structurally controlled, metamorphosed orogenic (lode) gold system. Gold occurs in moderately northeast-dipping shear zones that strike approximately N30°W and dip about 30° northeast. The shear zones are historically grouped as the Sherridon, Upper, Main and Lower zones and host quartz–arsenopyrite–pyrrhotite–pyrite veins. Mineralized shears are typically 1.3 m to 2.5 m wide. Drilling has traced the system to approximately 500 m vertical depth. The 2026 interpretation models 14 mineralized shear domains.

1.5 Exploration, Drilling and Data

The cleaned August 2026 project database contains 1,005 collars (858 related to PL) and approximately 146.7 km of drilling. This includes historical Pioneer/Maverick drilling (1980 to 1989), Auriga drilling (2010 to 2013) and Minnova drilling (2016 to 2026). The 2025 to 2026 program was focused on near-surface mineralization within the proposed open-pit resource area and was limited to approximately 200 m depth.

1.6 Sample Preparation, Analyses and Data Verification

Modern samples were analyzed at independent commercial laboratories (Accurassay, SGS, Activation Laboratories and SRC Laboratories) by fire assay with an atomic absorption finish and gravimetric finish for over-range samples. Certified reference materials, blanks and duplicates were inserted. N.R. King visited the site in April 2026. During the visit he reviewed core logging, sampling procedures, drill-data management and collar locations. In the QP's opinion, the database is adequate to support the estimation of Mineral Resources.

1.7 Mineral Processing and Metallurgical Testing

Historical testwork (Britton Research 1981, Lakefield Research 1984 and Coastech Research 1986) and plant records from December 1987 to May 1989 show that the mineralization responds to gravity concentration, flotation and cyanide leaching of the concentrates. The historical plant achieved an overall recovery to doré of about 83%, limited by poor plant availability, an unreliable jig and inadequate leach oxygenation. No metallurgical testwork has been completed since 1989. The PEA uses a design recovery of 90%, supported by the historical testwork (90% to more than 95%) and by planned improvements to the flowsheet: a centrifugal gravity concentrator with an intensive leach reactor, shear reactors and an oxygen supply in the leach, and improved regrinding. The flowsheet does not include ore sorting; ore sorting is recommended as testwork only.

1.8 Mineral Resource Estimate

The 2026 MRE was prepared by ABGM under the supervision of Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat. Gold grades were estimated by ordinary kriging into 10 m × 10 m × 1 m blocks within 14 interpreted shear domains, using 1 m composites, local means and outlier restriction. The volumes of historical workings were depleted. Nicholas Ross King is responsible for the geological model and the estimate. Reasonable prospects for eventual economic extraction (RPEEE) were demonstrated by an optimized open-pit shell and, below the pit, by underground Mineable Shape Optimiser (MSO) shapes. Ettienne Jaques Oosthuizen oversaw running of the pit optimization and the MSO, derived the cut-off grades and set the economic, cost and recovery parameters used for RPEEE. All figures assume a gold price of US$3,000/oz. Table 1-1 presents the Mineral Resources as at the Effective Date.

Table 1-1: PL Gold Mine Mineral Resource Estimate as at the Effective Date September 10, 2026

Mining method / cut-off	Category	Tonnes	Au (g/t)	Contained Au (oz)
Open pit (0.60 g/t Au)	Measured	1,288,360	5.35	221,575
	Indicated	289,750	4.99	46,516
	M & I	**1,578,110**	**5.28**	**268,091**
	Inferred	118,689	5.75	21,955
Underground (1.92 g/t Au; marginal 1.43 g/t Au)	Measured	1,343,678	5.06	218,603
	Indicated	835,239	4.26	114,464
	M & I	**2,178,916**	**4.75**	**333,066**
	Inferred	1,548,443	3.63	180,855
Total (open pit and underground)	Measured	2,632,038	5.20	440,178
	Indicated	1,124,989	4.45	160,980
	M & I	**3,757,026**	**4.98**	**601,158**
	Inferred	**1,667,132**	**3.78**	**202,810**

Notes to Table 1-1:

1. Mineral Resources are reported in accordance with the CIM 2014 Definition Standards. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

2. Open-pit Mineral Resources are reported at a cut-off grade of 0.60 g/t Au within an optimized pit shell. This full-cost cut-off is based on a mining cost of US$6.70/t for mineralized material (US$5.48/t for waste), a processing cost of US$26.81/t plus US$2.44/t haulage to the mill (US$29.25/t), a G&A cost of US$12.60/t and a reclamation cost of US$0.61/t (a total of US$49.16/t of mineralized material), a metallurgical recovery of 90%, a 3% royalty, a selling cost of US$31.68/oz and a gold price of US$3,000/oz (Table 14-11). The calculated full-cost cut-off of 0.59 g/t Au is rounded up to 0.60 g/t Au.

3. Underground Mineral Resources are reported within MSO shapes at a cut-off grade of 1.92 g/t Au. This break-even cut-off is based on a production mining (drift) cost of US$80.98/t, a mining development cost of US$41.16/t, a processing cost of US$26.81/t and a G&A cost of US$12.60/t (a total of US$161.54/t run-of-mine), a recovery of 90%, a 3% geological loss, a 3% royalty and a gold price of US$3,000/oz (Table 14-12). A marginal cut-off grade of 1.43 g/t Au, which excludes the mining development cost, is applied to material within and adjacent to the economic MSO shapes.

4. Mineral Resources are reported in situ and include a 3% geological loss. Mining dilution and mining recovery factors were used only to define the pit and stope limits. They have not been applied to the reported tonnes and grades.

5. A bulk density of 2.81 t/m³ was applied. Historical underground workings have been depleted.

6. Inferred Mineral Resources have a lower level of confidence than Measured and Indicated Mineral Resources and must not be converted to Mineral Reserves. They have not been added to the Measured and Indicated Mineral Resources. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

7. The Qualified Person for the Mineral Resource Estimate, including the geological model, domaining, block model, estimation and classification, is Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat. The Qualified Person for the open-pit optimization, the underground MSO constraints, the cut-off grades and the RPEEE parameters is Ettienne Jaques Oosthuizen, Principal Mining Engineer. The Effective Date is September 10, 2026.

8. The pit shell and MSO shapes are used only to demonstrate RPEEE. They are not Mineral Reserves and do not represent a mine plan.

9. Tonnes and ounces are rounded and totals may not sum exactly. One troy ounce equals 31.10348 g.

10. The 2017 CSA Global Mineral Resource estimate is superseded and is not relied upon.

1.9 Mineral Reserves

No Mineral Reserves have been estimated. The Mineral Reserves reported in the 2018 FS (effective October 31, 2017) are historical and not current. The PEA mine plan uses Measured, Indicated and Inferred Mineral Resources.

1.10 Mining Methods

The PEA mine plan combines contractor-mined open pits (OP) in 2028 to 2030 with a contractor-mined underground (UG) operation from 2029, accessed through a new portal in the north-east pit (373,663.6 E, 6,100,247.1 N, 292.0 m RL) (Item 16). The OP mines 1.756 million tonnes (Mt) of plant feed at 4.68 g/t Au with 36.91 Mt of waste (strip ratio 21.0). The UG mine supplies 4.743 Mt at 2.80 g/t Au with 3.415 Mt of waste development, and becomes the sole source of plant feed from 2031. The mine plan is supported by a final geotechnical study (R. Lousteau, 2026), a VENTSIM ventilation study (Umoya, 2026) and a dewatering assessment (Water Hunters, October 19, 2026).

The PEA cash flow model carries no mine-air heating or backfill cost. Waste heat from the gensets is the planned primary source of mine-air heating, and the underground mining method, including options that require no backfill or only partial backfill, will be assessed in trade-off studies at the PFS stage.

1.11 Production Schedule

Total plant feed is 6.499 Mt at 3.30 g/t Au, containing 691 thousand ounces (koz) of gold. At 90% recovery the project produces 621.5 koz of gold over ten years (2028 to 2037), with peak production of 91.6 koz in 2029.

1.12 Recovery Methods

The existing PL mill will be refurbished to treat 1,000 tonnes per day (t/d) and then expanded to 2,000 t/d. The flowsheet comprises crushing, grinding, gravity concentration with intensive leaching, flotation, and cyanide leaching of the flotation and gravity concentrates, with carbon-in-pulp recovery and doré production. Flotation tailings are deposited in the Ragged Lake tailings management area and the leach residue in a new lined sulphide tailings facility. The PEA uses a recovery of 90%. The QP's plant ramp-up basis (tonnage and recovery ramp-up) is assessed as a sensitivity and is not part of the base case.

1.13 Project Infrastructure

The project uses the existing PL site: the mill buildings, the Ragged Lake tailings management area, the airstrip and access road from Sherridon. Underground access is through a new portal in the north-east pit (373,663.6 E, 6,100,247.1 N, 292.0 m RL). New infrastructure comprises liquefied natural gas (LNG)-fuelled gensets for the first 36 months until the grid connection to Sherridon is restored, a temporary 70-person camp, a 525 kL fuel farm, a lined sulphide tailings facility, two waste rock dumps (WRDs), a south-west WRD on ML065 and a north WRD on claim MIT-045, a NAG waste rock embankment on perimeter B of the Ragged Lake TMA (crest

352.4 m), and UG ventilation shafts, together with the truck shop, LNG/CNG pad and power plant, contractor yard, effluent treatment plant and discharge, security gate, explosives and detonator magazines and 25 kV line shown on Figure 18-1. Manitoba Hydro has advised that restoring supply to Sherridon may take about two years (up to three years) and that the capacity then available may be about 3 MW to 3.5 MW, below the project demand, so the grid power assumed in the model from Month 37 (2031) is not assured (Item 18).

1.14 Market Studies and Contracts

Gold doré is sold to a refinery at the spot price. The PEA uses a base-case gold price of US$3,000/oz, with US$3,500/oz and US$4,000/oz as upside cases. The PL Property is subject to a sliding-scale net smelter return (NSR) royalty held by Franco-Nevada Corporation: 3.0% at gold prices at or above US$1,000/oz, 2.5% from US$750/oz to below US$1,000/oz, and 2.0% below US$750/oz. The cash flow model applies a flat 3% of gross revenue, consistent with the filed MRE cut-off calculation. At the base-case gold price the 3.0% rate applies, and applying it to gross revenue rather than NSR is a slightly conservative simplification. No offtake, streaming or sales agreements are in place, and no LNG supply contract has been signed.

1.15 Environmental Studies, Permitting and Social or Community Impact

The existing Environment Act Licence 1207E relates to the historical underground operation and the existing plant. Open pit mining and the expansion to 2,000 t/d require a Notice of Alteration to the licence, and the use of Ragged Lake for tailings requires listing on Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER). Minnova has filed an Environment Act Intent to Apply form for a major Notice of Alteration, and is targeting filing of the updated Schedule 2 listing documents by the end of 2026.

1.16 Capital and Operating Costs

All costs are in real United States dollars (US$). **The cost base date is Q3 2026.** The capital costs are summarized in Table 1-2 and the operating costs in Table 1-3.

Table 1-2: Capital cost summary (US$M)

Capital class	US$M
Pre-production capital (2026 and 2027), including US$7.0M contingency	53.4
Total initial (non-sustaining) capital	55.4
Sustaining capital	62.9
Closure and disestablishment	2.6
Total LOM capital	120.9

Source: ABGM cash flow model (2026). Totals may not add because of rounding.

Table 1-3: Operating cost summary

Cost area	LOM (US$M)
OP mining	217.1
UG mining	726.5
Processing (including mill power)	182.8
Site general and administration (G&A) and other shared costs	154.4
Total operating cost	1,280.8
Unit cost per tonne milled (US$/t)	**197.08**
Unit cost per ounce recovered (US$/oz)	2,061

Source: ABGM cash flow model (2026).

The total cash cost is US$2,183/oz and the all-in sustaining cost (AISC, World Gold Council basis) is US$2,288/oz. G. Cunningham, QP for Recovery Methods, has estimated process plant and tailings capital that differs from the model. These differences are reported as a cost sensitivity; the base case is unchanged.

1.17 Economic Analysis

Cautionary statement. The preliminary economic assessment (PEA) is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

At the base-case gold price of US$3,000/oz the project has an after-tax net present value (NPV) at a 5% discount rate of US$180.6M (US$151.4M at 8%), an after-tax internal rate of return (IRR) of 111.0% and an after-tax payback of 0.68 years from the start of production in 2028. The pre-tax NPV at 5% is US$283.4M and the pre-tax IRR 143.1%. After-tax free cash flow is US$247.6M after taxes of US$139.6M.

Table 1-4: Economic results by gold price (after tax)

Indicator	US$3,000/oz (base)	US$3,500/oz	US$4,000/oz
After-tax NPV at 5% (US$M)	180.6	310.8	438.9
After-tax NPV at 8% (US$M)	151.4	260.1	367.2
After-tax IRR (%)	111.0	151.6	187.5
After-tax payback (years)	0.68	0.54	0.45
AISC (US$/oz)	2,288	2,305	2,323

Source: ABGM cash flow model (2026).

The project is most sensitive to gold price, head grade and recovery, followed by operating cost. It is relatively insensitive to capital cost and to the exchange rate. On the project ramp-up and capital sensitivity case (a slower ramp to 2,000 t/d with first feed in January 2029, a recovery ramp of 80%, 85% and 90%, and the revised plant and tailings storage facility capital), the after-tax NPV at 5% is US$159.8M, US$20.8M below the

base case, and the after-tax IRR is 70.4%. The January 2029 first feed applies to that QP sensitivity case only; the base case remains first feed in March 2028 and is unchanged.

1.18 Risks and Opportunities

- Plant ramp-up: the base case reaches 2,000 t/d in Month 8 (August 2028; first feed March 2028), faster than the plant construction sequence. The January 2029 first-feed date applies only to the project ramp-up sensitivity case (slower ramp to 2,000 t/d) and would defer revenue relative to the base case.
- Costs not carried in the base case: mine-air heating (by-product heat from on-site generation can and should dictate genset location); backfill (to be determined in detailed engineering: waste rock, cemented aggregate, or both); and the QP's higher plant expansion and tailings estimates (see Table 21-4 and sensitivity C2: expansion to 2,000 t/d about US$6.0M higher, including contingency, than the model; Ragged Lake/TSF about US$9M versus the model US$4M).
- Inferred Mineral Resources in the mine plan.
- Permitting of the open pit, the expansion to 2,000 t/d and the Ragged Lake tailings management area.
- Power supply: the model assumes grid power from Month 37 (2031), but Manitoba Hydro has advised that the Sherridon supply may take about two to three years to restore and may then be below the project demand.
- Opportunities: gold prices above the base case, ore sorting (testwork), down-dip and PL North resource extensions, an earlier grid connection, and the contiguous Nokomis deposit as a future satellite source of plant feed.

1.19 Conclusions and Recommendations

In the QPs' opinion, the PEA shows that the restart of the PL Gold Mine as a combined OP and UG operation has positive economics at the base-case gold price, and that further study is warranted. The QPs recommend a work program comprising the Mineral Resource update program, metallurgical testwork, geotechnical, hydrogeological and ventilation engineering, permitting, and the reconciliation of the open items listed in this Technical Report, before a pre-feasibility study (PFS). The QPs recommend that the project proceed to a PFS, with a recommended budget of C$1.7M for the Mineral Resource and permitting program (Table 26-1) and about US$0.9M for engineering studies, consistent with the allowances in the financial model (Table 26-2).

2 INTRODUCTION

2.1 Issuer and Terms of Reference

Minnova Corp. is a Canadian company listed on the TSX Venture Exchange (TSXV: MCI) with its head office at 217 Queen Street West, Suite 401, Toronto, Ontario M5V 0R2. Before June 2014 the Company was named Auriga Gold Corp. Minnova commissioned A and B Global Mining (Pty) Ltd (ABGM), of Centurion, South Africa, to prepare the PEA and this Technical Report for the PL Gold Mine.

2.2 Purpose of the Report

The Technical Report supports the scientific and technical disclosure of the PEA in Minnova's news release of October 14, 2026. It is filed under section 4.2 of NI 43-101. The report title is 'NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project'. The Mineral Resources are those of the 2026 MRE, effective September 10, 2026, described in the technical report 'NI 43-101 Technical Report and Mineral Resource Estimate for the PL Gold Mine, Manitoba, Canada'.

2.3 Qualified Persons and Responsibilities

The Qualified Persons (QPs) responsible for this Technical Report, and the Items for which each is responsible, are listed in Table 2-1.

Table 2-1: Qualified Persons and Item responsibilities

Qualified Person	Position / company	Items responsible
Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat.	Principal Geologist, ABGM	Items 4 to 12, 14 (except subsection 14.8) and 23. Shared: Items 1, 2, 3, 24, 25, 26 and 27.
Ettienne Jaques Oosthuizen, B.Tech. (Mining Engineering), MIMMM, QMR	Principal Mining Engineer and Director, ABGM	Subsection 14.8, Items 15, 16, 18, 19, 20, 21 (except process plant and tailings costs) and 22. Shared: Items 1, 2, 3, 24, 25, 26 and 27.
Gordon Ian Cunningham, B.Eng. (Chemical Engineering), **Pr.Eng ECSA, FSAIMM**	Associate Principal Process Engineer, ABGM	Items 13 and 17, and the process plant and tailings capital and operating costs in Item 21. Shared: Items 1, 2, 3, 18, 20, 24, 25, 26 and 27.

Source: ABGM.

Mr. King has overall responsibility for the 2026 MRE. Mr. Oosthuizen is responsible for the mine plan, including the geotechnical, hydrogeological, ventilation and backfill inputs to Item 16, project infrastructure and environmental studies (Items 18 and 20), the cost model and the economic analysis. Mr. Cunningham is responsible for metallurgy, the process plant and the tailings facilities.

2.4 Site Visits

Mr. King visited the PL Gold Mine from April 24 to 27, 2026. During the visit he reviewed diamond drill core, core logging and sampling procedures, drill-data management and the verification of drill collars.

Mr. Oosthuizen and Mr. Cunningham visited the PL Gold Mine together from January 25 to 28, 2026.

2.5 Sources of Information

Information for this Technical Report came from:

- The Minnova drillhole database (as of August 2026) and ABGM's geological and resource modelling;
- The Feasibility Study, PL Gold Project, Manitoba, Canada (A-Z Mining Professionals Ltd. and CSA Global), dated April 23, 2018 with an effective date of October 31, 2017 (2018 FS);
- The CSA Global block model reports by domain and validation results, dated October 3, 2017;
- Minnova news releases from 2014 to 2026;
- Information supplied by Minnova on tenure, royalties, permits and the power supply status.
- The PEA cash flow model (ABGM, October 14, 2026);
- The Recovery Methods and metallurgical drafts and the ramp-up workbook prepared by G. Cunningham (2026);
- The final geotechnical study by R. Lousteau (2026), the VENTSIM life-of-mine ventilation study by Umoya Occupational Hygiene & Laboratory Solutions (Pty) Ltd (2026), the dewatering assessment by Water Hunters (October 19, 2026) and the mining methods draft by P. Cronje (2026);
- The AECOM draft project description for the Environment Act Proposal (2026).

The QPs have reviewed this information and believe it is reliable for the purposes of this Technical Report, except where stated otherwise. Mr. Oosthuizen has reviewed and relied on the final geotechnical study (R. Lousteau, 2026), the VENTSIM ventilation study (Umoya, 2026) and the dewatering assessment (Water Hunters, October 19, 2026) for those items.

2.6 Effective Date

The Effective Date of the Mineral Resource Estimate is September 10, 2026. The Effective Date of this Technical Report is **September 10, 2026**. The cost base date of the capital and operating cost estimates is **Q3 2026.**

2.7 Units, Currency, Abbreviations

Metric (SI) units are used throughout. Gold grades are in grams per tonne (g/t Au) and contained gold in troy ounces (oz; 1 oz = 31.10348 g). Unless otherwise stated, all currency is in United States dollars (US$). Canadian dollar amounts are stated as C$ and are converted at US$0.74 per C$1.00 where so stated. South African rand amounts are stated as R. Coordinates are in NAD83 UTM Zone 14N. Abbreviations are listed at the front of the report.

3 RELIANCE ON OTHER EXPERTS

The QPs have relied on the information, reports and opinions listed below for matters outside their expertise. As permitted by Item 3 of Form 43-101F1, except as described below, they have not independently verified this information.

Table 3-1: Information relied upon

Subject	Relied on	Document	Extent of reliance (Sections)
Mineral tenure and title: mining claims and Mining Lease ML065, registered ownership and good standing	Minnova Corp. and its legal counsel	Minnova Corp. (2026)	Sections 4.2 and 4.7, and the related parts of Section 1
Surface rights and Crown land permits	Minnova Corp.	Minnova Corp. (2026)	Section 4.4
Royalties, back-in rights, offtake, supply and community agreements, and other encumbrances	Minnova Corp. and its legal counsel	Minnova Corp. (2026); NSR Royalty Agreement dated October 8, 2010 and Royalty Deed and Assignment to Franco-Nevada Corporation effective November 1, 2013 (copies supplied by Minnova)	Section 4.5, and Items 19 and 22
Environmental liabilities, permits and licences (EAL 1207E, Water Rights Act licence 2017-116, MDMER Schedule 2 status, closure plan, quarry permit applications)	Minnova Corp. and its environmental consultants	Minnova Corp. (2026), including the quarry permit applications	Sections 4.6 and 4.7, and Item 20
Opening tax pools and tax attributes	Minnova Corp.	Minnova Corp. audited financial statements (March 2026)	Item 22 (taxation)
Power supply status: grid connection applications, Manitoba Hydro capacity and timing, and the power-line easement	Minnova Corp. (including advice received from Manitoba Hydro)	Minnova Corp. (2026); AECOM draft project description (2026)	Items 1, 18, 19, 20 and 25
Environmental and permitting status for the PEA project	Minnova Corp. and AECOM Canada Ltd.	AECOM draft project description (2026); Minnova Corp. (2026)	Item 20

The QPs have used specialist technical studies for the geotechnical (R. Lousteau, 2026), ventilation (Umoya, 2026) and dewatering (Water Hunters, October 19, 2026) inputs to Item 16. These are technical matters, which cannot be relied on under Item 3 of Form 43-101F1; a QP must take responsibility for them. Mr. Oosthuizen has reviewed these studies and takes QP responsibility for those inputs.

The 2018 FS authors relied on a letter dated October 29, 2010 from the Manitoba Mines Branch to Aikens, MacAulay & Thorvaldson LLP (Winnipeg). The letter confirms the transfer of ML065 from Pioneer to Minnova (then Auriga Gold), registered under Document No. 17465 on October 14, 2010. That letter is historical. It is cited here for context only and has not been used to confirm current title.

The QPs checked tenure against Manitoba iMaQs records on October 24, 2026 and reviewed copies, supplied by Minnova, of the NSR Royalty Agreement dated October 8, 2010 and the Royalty Deed and Assignment to

Franco-Nevada Corporation effective November 1, 2013. They rely on Minnova (G. Glenn, written communication, October 24, 2026) for the current status of the permits, the royalties, the closure financial assurance and the Nokomis royalty, and for the absence of offtake, LNG supply and community agreements. They rely on written communications from G. Glenn of Minnova dated October 21 and 22, 2026 for the status of the power supply, including the advice Minnova has received from Manitoba Hydro, and for the October 2026 quarry permit applications.

The QPs do not offer a legal opinion on the status of title, royalties or permits.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The PL Gold Mine is in northwestern Manitoba, Canada, within NTS map sheet 63N/02 . It lies approximately 63 km northeast of Flin Flon and approximately 12 km southeast of the community of Sherridon. The 2018 FS gives the centre of the Maverick Project (PL and Nokomis properties) as about 55°02’ N, 100°59’ W, and the PL deposit on Mining Lease ML065 as near 55°02’05” N, 100°58’52” W (2018 FS, pp. 2 and 28). The Nokomis deposit is near 55°04’50” N, 100°52’00” W, less than 8 km northeast of the PL mill. In UTM terms, the PL mine-site drilling lies near 373,000 mE and 6,101,000 mN (NAD83 UTM Zone 14N), for example hole M-20-01 at 373,028 mE and 6,100,986 mN (July 30, 2020 news release) .

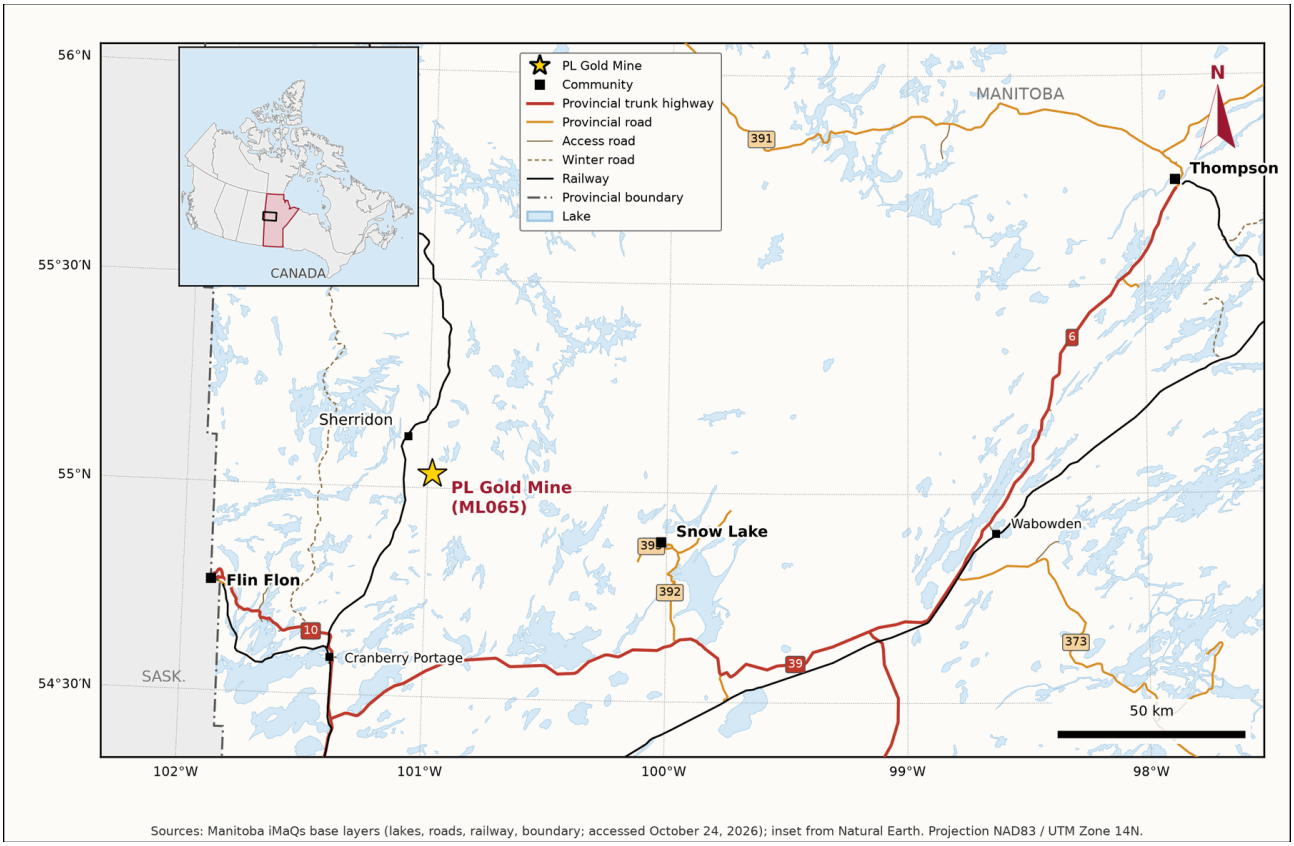


Figure 4-1: Location of the PL Gold Mine. Source: Manitoba iMaQs (rdmaps.gov.mb.ca), accessed October 24, 2026.
Note: iMaQs may be inaccurate for the Sherridon access-road alignment; tenure mapping from this source is retained.

4.2 Mineral Tenure

Manitoba mineral dispositions are administered under The Mines and Minerals Act and the Mineral Disposition and Mineral Lease Regulation, 1992 (M.R. 64/92). Table 4-1 lists the mineral dispositions held by Minnova as recorded in the Manitoba Integrated Mining and Quarrying System (iMaQs) on October 24, 2026, and Figure 4-2 shows their location. The PL Property comprises Mineral Lease ML065 (378 ha) and 20 mining claims (3,552 ha), and the Nokomis Property comprises 27 mining claims (566 ha), for a total of 4,496 ha. All of these dispositions are registered to Minnova Corp. (iMaQs client 259161) and were in good standing at that date. Mineral Lease ML065 is in good standing. It is maintained by an annual rental payment due on April 1 each year; iMaQs records the current anniversary date as April 1, 2027 and an expiry date of May 1, 2027, which applies only if the annual rental is not paid. Minnova has advised that the rental has been paid each year.

Table 4-1: Mineral tenure, Minnova Corp. (Manitoba iMaQs, October 24, 2026)

Disposition no.	Name	Type	Issue date	Anniversary (good-to) date	Expiry date	Area (ha)
PL Property						
ML065 (ML65)	ML065	Mineral lease	1992-04-01	2027-04-01	2027-05-01 (a)	378
MB13110	HAM01	Mining claim	2019-06-17	2030-06-17	2030-08-16	150

Disposition no.	Name	Type	Issue date	Anniversary (good-to) date	Expiry date	Area (ha)
MB13111	HAM02	Mining claim	2019-06-17	2030-06-17	2030-08-16	128
MB13105	MIT-009	Mining claim	2017-11-23	2029-11-23	2030-01-22	248
MB13104	MIT-013	Mining claim	2017-11-23	2029-11-23	2030-01-22	256
MB13106	MIT-014	Mining claim	2017-11-23	2029-11-23	2030-01-22	256
MB13108	MIT-015	Mining claim	2017-11-23	2029-11-23	2030-01-22	256
MB12262	MIT-016	Mining claim	2017-11-23	2029-11-23	2030-01-22	157
MB12169	MIT-017	Mining claim	2017-05-25	2029-05-25	2029-07-24	160
MB12247	MIT-020	Mining claim	2017-05-25	2030-05-25	2030-07-24	190
MB12246	MIT-021	Mining claim	2017-05-25	2029-05-25	2029-07-24	212
MB12243	MIT-022	Mining claim	2017-05-25	2029-05-25	2029-07-24	128
MB12244	MIT-025	Mining claim	2017-05-25	2031-05-25	2031-07-24	128
MB12245	MIT-026	Mining claim	2017-05-25	2029-05-25	2029-07-24	120
MB12242	MIT-027	Mining claim	2017-05-25	2029-05-25	2029-07-24	128
MB12259	MIT-044	Mining claim	2017-08-23	2029-08-23	2029-10-22	217
MB12260	MIT-045	Mining claim	2017-09-22	2029-09-22	2029-11-21	220
MB12261	MIT-046	Mining claim	2017-11-23	2029-11-23	2030-01-22	89
MB12263	MIT-047	Mining claim	2017-11-23	2029-11-23	2030-01-22	117
P1325F	STU 9	Mining claim	1992-11-04	2029-11-04	2030-01-03	136
P1326F	STU 10	Mining claim	1992-11-04	2029-11-04	2030-01-03	256
Subtotal, PL mining claims (20)						**3,552**
Subtotal, PL Property (ML065 and 20 claims)						**3,930**
Nokomis Property						
P4173E	MIS NO. 1	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4174E	MIS NO. 2	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4175E	MIS NO. 3	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4176E	MIS NO. 4	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4177E	MIS NO. 5	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4178E	MIS NO. 6	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4179E	MIS NO. 7	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4180E	MIS NO. 8	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4181E	MIS NO. 9	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4182E	MIS NO. 10	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4183E	MIS NO. 11	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4184E	MIS NO. 12	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4188E	MIS NO. 16	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4189E	MIS NO. 17	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4190E	MIS NO. 18	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4191E	MIS NO. 19	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4192E	MIS NO. 20	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4193E	MIS NO. 21	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4195E	MIS NO. 23	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4196E	MIS NO. 24	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4197E	MIS NO. 25	Mining claim	1972-07-07	2030-07-07	2030-09-05	21

Disposition no.	Name	Type	Issue date	Anniversary (good-to) date	Expiry date	Area (ha)
P4198E	MIS NO. 26	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4199E	MIS NO. 27	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4200E	MIS NO. 28	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4201E	MIS NO. 29	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
P4202E	MIS NO. 30	Mining claim	1972-07-07	2030-07-07	2030-09-05	21
MB13101	MIT-007	Mining claim	2017-11-23	2027-11-23 (b)	2028-01-22	20
Subtotal, Nokomis Property (27 claims)						**566**
Total (1 mineral lease and 47 mining claims)						**4,496**

Notes: All dispositions are registered to (259161) Minnova Corp. and were in good standing on October 24, 2026. Dispositions are grouped by location, consistent with the 2018 FS: STU 9 and STU 10 adjoin ML065 and form part of the PL Property, and MB13101 (MIT-007) lies at the Nokomis deposit. Four MIT claims (MB13102, MB13103, MB13107 and MB13109; 774 ha) were cancelled in 2020 and are excluded. A mining claim remains in good standing to its anniversary date and expires 60 days later if the required work or payment has not been recorded. (a) Mineral Lease ML065 is in good standing. It is maintained by an annual rental payment due on April 1 each year; iMaQs records the current anniversary date as April 1, 2027 and an expiry date of May 1, 2027, which applies only if the annual rental is not paid. Minnova has advised that the rental has been paid each year. (b) MB13101: the renewal has been paid; iMaQs shows the claim in good standing to November 23, 2027.

Source: Manitoba iMaQs (rdmaps.gov.mb.ca), accessed October 24, 2026; Minnova Corp. disposition listing.

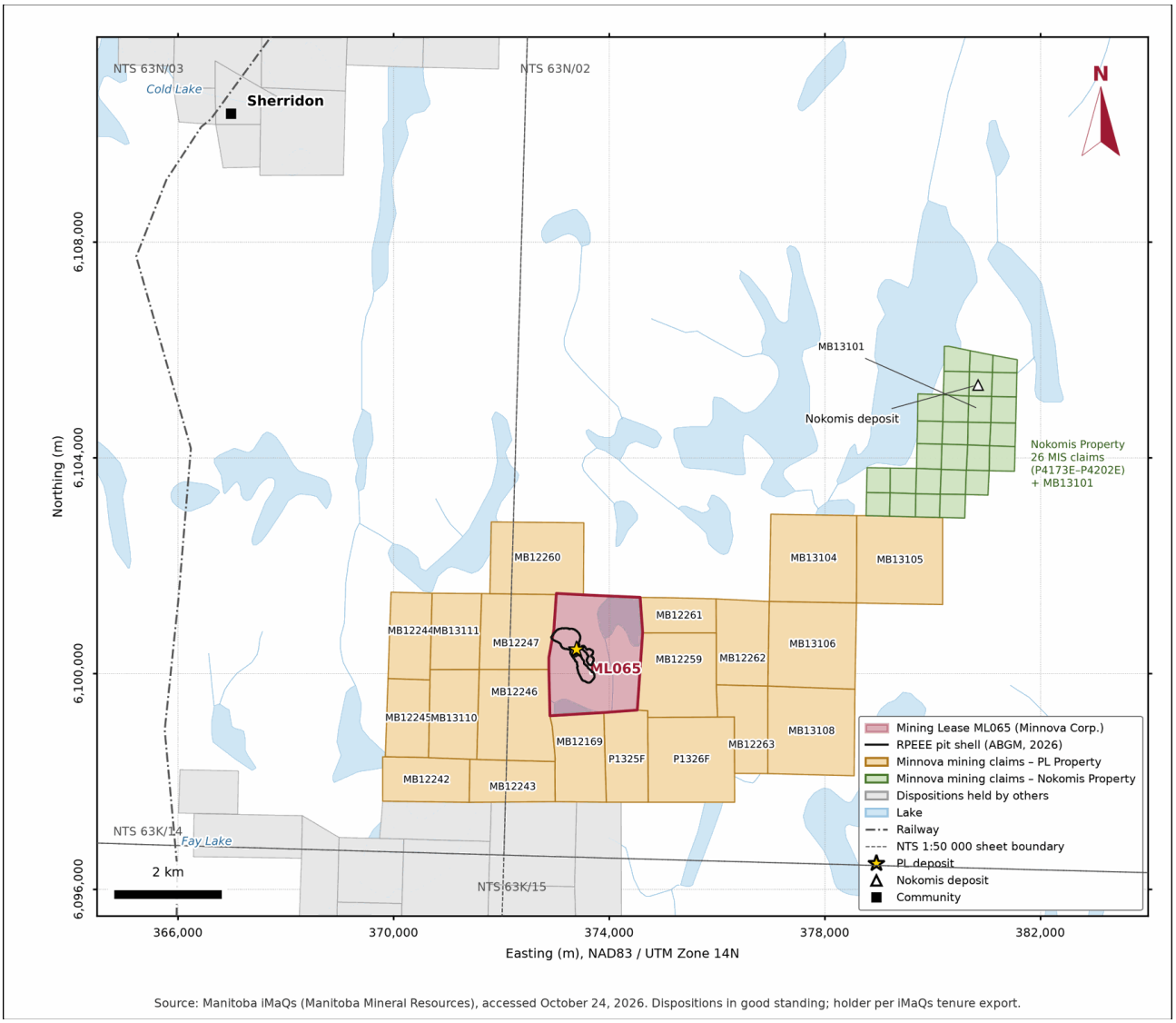


Figure 4-2: PL and Nokomis mineral tenure, Minnova Corp. Source: Manitoba iMaQs (rdmaps.gov.mb.ca), accessed October 24, 2026.

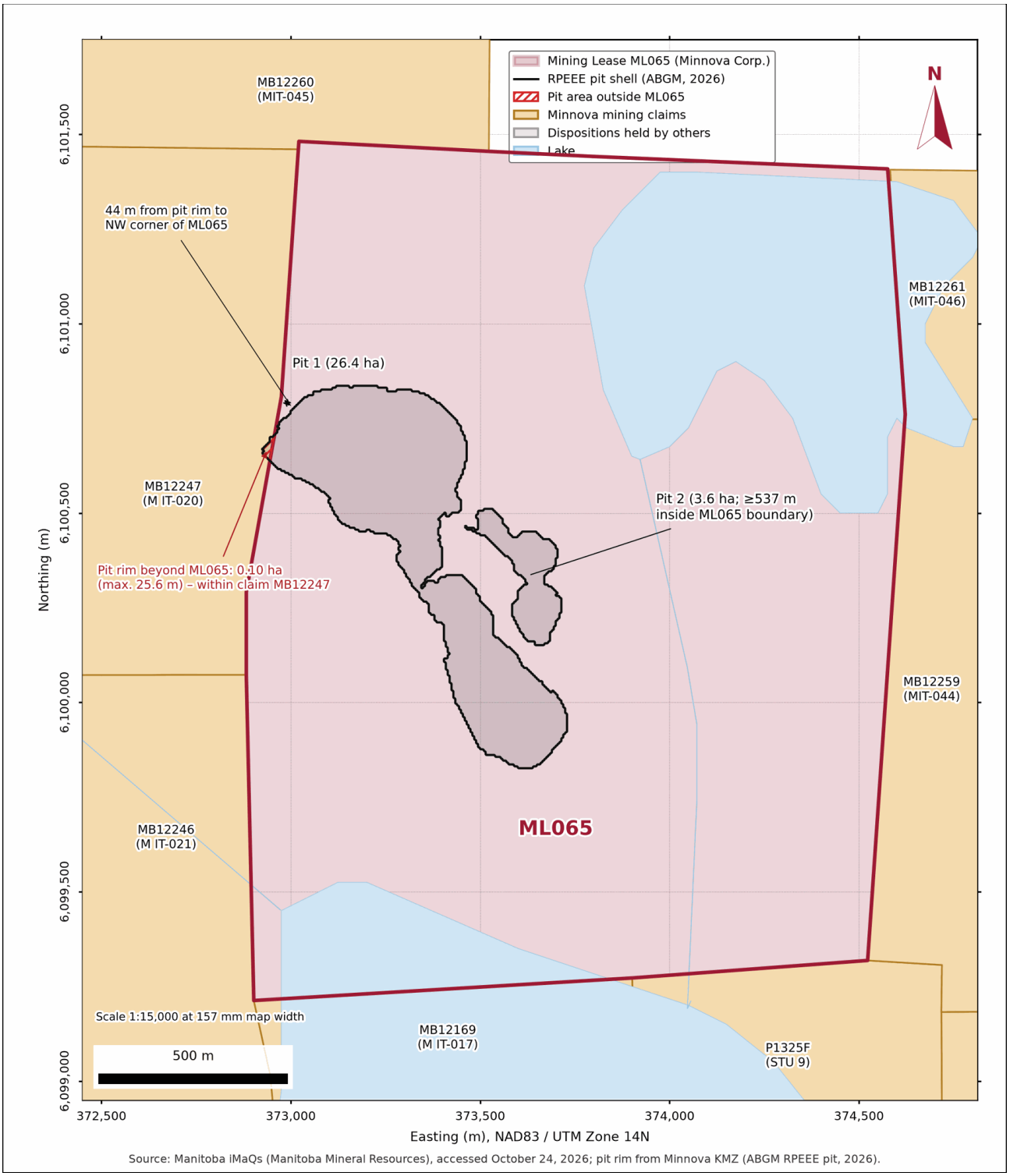


Figure 4-3: RPEEE open pit shell relative to Mineral Lease ML065. Source: ABGM, 2026 (pit shell); Manitoba iMaQs (rdmaps.gov.mb.ca), accessed October 24, 2026 (lease boundary).

ML065 originated as claim CB 10186, which was converted to a mining lease on October 30, 1987. It was surveyed by Bastin & Shepherd, Manitoba Land Surveyors, in September 1987. The staked claims have not been surveyed (2018 FS, pp. 31 and 47).

The 2018 FS places the PL deposit on ML065. Figure 4-3 shows the RPEEE open pit shell relative to the lease. The RPEEE open pit shell lies almost entirely within Mineral Lease ML065. A sliver of approximately 0.1 ha on the western rim extends up to about 25 m beyond the lease boundary as mapped in iMaQs, onto the adjoining claim MB12247, which is also held by Minnova Corp. The iMaQs boundary is not a legal survey, and this overlap is within plausible digitizing error; it should be checked against the ML065 survey plan.

4.3 Assessment Work and Lease Rental

A recorded Manitoba mining claim is in good standing for two years plus 60 days. It may be renewed each year if the required work is done: C$12.50/ha in years 2 to 10 and C$25/ha from year 11, or an equivalent cash payment. Claims may be grouped so that work credits are shared. A mining lease has a 21-year renewable term and an annual rental.

4.4 Surface Rights

Surface rights over the property are held by the Crown (Province of Manitoba). Surface infrastructure outside the lease is held under Crown land permits GP0004134 (tailings dam), GP0002799 (road to Puffy), GP0003758 (water pipe right of way) and GP0004038 (access road). These permits are renewed every year.

Table 4-2: Permits and licences held by Minnova Corp.

Permit / licence	Number	Expiry date
Environment Act Licence – Permit to Mine	1207E	N/A
Mining Lease	ML065	Annual renewal; rental due April 1 (next April 1, 2027)
Licence to Use Water for Industrial Purposes	2017-116	5 September 2027
Crown Land Permit – Access Road Right of Way	GP0002799	31 December 2026
Crown Land Permit – Water Pipeline Right of Way	GP0003758	31 December 2026
Crown Land Permit – Access Road Right of Way	GP0004038	31 December 2026
Crown Land Permit – Mine Tailings Containment	GP0004134	31 December 2026
Casual Quarry Permit	CP-2026-1025197	30 November 2026

Source: Minnova Corp., October 2026. Mining Lease ML065 is maintained by an annual rental payment (see Table 4-1).

4.5 Royalties / Encumbrances

The following royalties apply:

- PL Property: a sliding-scale NSR royalty granted to Pioneer Metals ULC, then a Barrick subsidiary, under Minnova's 2010 acquisition agreement, and assigned to Franco-Nevada Corporation effective November 1, 2013. The rate is 3.0% at gold prices at or above US$1,000/oz, 2.5% from US$750/oz to below US$1,000/oz, and 2.0% below US$750/oz. It applies to Mineral Lease ML065 and the claims listed in the agreement.
- Nokomis Property: an underlying 2% NSR royalty, listed in the 2018 FS as held by Hudbay Minerals Inc. and Placer Dome Inc. (now Barrick). Minnova advises that Hudbay has notified it of its interest,

but that the 1% attributed to Placer Dome was never registered on title and has not been confirmed by Barrick.

The QP is not aware of any other back-in rights, payments or encumbrances.

4.6 Environmental Liabilities

The PL Gold Mine is a past producer. It operated under Manitoba Environment Act Licence 1207E from late 1987 to April 1989. The site contains a portal and ramp, more than 7,000 m of underground workings, the processing plant, and more than 350,000 t of tailings in the Ragged Lake tailings management area (TMA) (2018 FS, p. 33). Minnova has provided a C$75,000 letter of credit under its closure plan and has pledged site assets as security of up to C$5,000,000. It carries a closure and reclamation provision of C$3,718,583 at June 30, 2026.

4.7 Permits

Table 4-3 lists the principal permits and approvals. Federally, deposition of tailings in Ragged Lake requires the lake to be listed on Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER, formerly the MMER). Listing requires an approved fish-habitat compensation plan and an irrevocable letter of credit. Minnova has advised that AECOM has completed a gap analysis of the earlier Alternatives Assessment report and will update it on the basis of the development plan, with Minnova targeting filing by the end of 2026.

Table 4-3: Permits and approvals – status as reported by Minnova Corp. to the QP, updated to October 24, 2026

Permit / approval	Authority	Status / comment
Environment Act Licence 1207E (underground mine, mill, Ragged Lake TMA)	Manitoba Environmental Approvals Branch	In place. Manitoba confirmed in May 2012 that the licence was in good standing for a restart. Open-pit mining and other changes to the licensed development need an approved alteration. Minnova has filed an Environment Act Intent to Apply form for a major Notice of Alteration
Mining Lease ML065	Manitoba Mines Branch	In good standing. Maintained by an annual rental payment due on April 1 each year; iMaQs records the current anniversary date as April 1, 2027 and an expiry date of May 1, 2027, which applies only if the annual rental is not paid. Minnova has advised that the rental has been paid each year.
Crown land permits GP0002799, GP0003758, GP0004038, GP0004134	Manitoba	Renewed annually
Water Rights Act licence 2017-116 (industrial water)	Manitoba	Issued September 2017; expires 5 September 2027
MDMER Schedule 2 listing of Ragged Lake TMA	Environment and Climate Change Canada / DFO	Application first submitted to DFO in May 2017. Minnova has advised that AECOM has completed a gap analysis of the earlier Alternatives Assessment report and will update it on the basis of the development plan, with Minnova targeting filing by the end of 2026.
Closure plan and financial assurance (Mine Closure Regulation, M.R. 67/99)	Manitoba Mines Branch	The Company has provided a letter of credit in the amount of C$75,000 to the Government of Manitoba under the terms of the closure plan of the PL Property. The Company further provided all assets, goods and personal property involved in the operation of the PL Property, as a security of up to C$5,000,000 for the performance of the closure plan and the rehabilitation program.

4.8 Other Significant Factors and Risks

Apart from the confirmations flagged above, the QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political factor that would materially affect access to the property, title to it, or the right or ability to carry out work on it.

5 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

An all-weather gravel road runs 78 km north from Provincial Highway 10, south of Flin Flon, to Sherridon. At about km 65.5 the 9 km PL mine access road branches east to the mine site. The access road was built in 1986 and crosses the Keewatin Railway Company line at Mile 33 (km 52.8) from Sherritt Junction. The rail line passes about 6 km west of the property. Flin Flon airport, which has scheduled flights from Winnipeg, offers charter floatplane and helicopter services.

5.2 Climate

The area has a continental subarctic climate with short, warm summers and long, cold winters. The nearest Environment and Climate Change Canada station is at Flin Flon. The 1991–2020 Canadian Climate Normals for Flin Flon give a mean daily minimum temperature of −24.1 °C in January, a mean daily maximum of 23.7 °C in July, a mean annual temperature of 0.2 °C and mean annual precipitation of 490.6 mm, with July the wettest month at 85.7 mm. The 2018 FS notes that about 30% of precipitation falls as snow and about 55% falls from June to September. Exploration can be carried out all year. Drilling in wet lowland areas is best done in winter, and a mining operation with suitable infrastructure can run all year.

5.3 Local Resources

Sherridon (2021 population of about 56) is about 20 km from the site by road. Flin Flon (2021 population of 4,940, Manitoba part) is about 115 km by road and has Hudbay's metallurgical complex and a range of mining suppliers and contractors. Winnipeg is about 750 km south of Flin Flon. Experienced mining personnel are available in the Flin Flon and Snow Lake mining communities.

5.4 Existing Infrastructure

Much of the infrastructure from the 1986–1989 operation is still on site on a care-and-maintenance basis. The operation was suspended in March 1989 and closed by June 1989. Later in 1989 (August 8 to 11), after the operation had been placed on care and maintenance, forest fires damaged the power line and the freshwater and tailings pipelines; the crusher, mill and office complex had a firebreak and escaped damage (2018 FS, pp. 43 and 47; AECOM, 2014; Manitoba Mineral Inventory File 0984). Parts of the site have since been vandalized, and the 2025 wildfires caused further damage to the power line. The condition descriptions are based on site information from Minnova and ABGM. It comprises:

- a portal and ramp to about 135 m below surface, with level and raise development. The workings are flooded;
- the 1,000 tpd processing plant building, which is structurally sound, with crushing, grinding, gravity, flotation and leach equipment. Some equipment and wiring have been removed or vandalized;
- offices, a mine dry, a shop, a warehouse and an assay laboratory, all requiring refurbishment, a core-logging building and a 16-person exploration camp;
- the Ragged Lake TMA, which received about 350,000 t of tailings;
- fresh water from nearby lakes under Water Rights Act licence 2017-116; the original freshwater and tailings pipelines were largely destroyed by the 1989 forest fires;
- a 22 km, three-phase 25 kV power line from the Manitoba Hydro substation at Sherridon. Minnova agreed to buy the line from Manitoba Hydro for C$150,000 (November 12, 2021 news release) and completed the purchase in December 2021 (December 13, 2021 news release). The section of the

line along the mine access road was destroyed by the 1989 forest fire (2018 FS, p. 43). The line needs to be refurbished before it can be used: all poles were replaced with new poles in 2012, but new conductors (wires) are required, and the 2025 wildfires caused further damage, with several poles lost;

- limited cellular coverage (Telus) at the fine ore bin; site communications are mainly by Starlink satellite internet at the camp and Starlink Mini terminals at the core shack and for mobile use.

The 2018 FS considered the property large enough for mining, processing, tailings and waste rock storage.

5.5 Physiography

The terrain is typical of the Canadian Shield, with low relief, rock ridges trending north, lakes and swamps. Elevations range from about 350 m above sea level at the mill site to 340 m at Puffy Lake. The boreal forest is dominated by black spruce and tamarack in the lowlands, and by poplar, white spruce and jack pine on the uplands. Glacial overburden varies in thickness. Much of the western part of the property was burned in the 1989 forest fire (2018 FS, p. 44).

6 HISTORY

6.1 Ownership History

Table 6-1: Ownership history of the PL Property

Year	Event
1953	A. Oleson stakes claims NIP 1–4 over the area of the present mine site.
1960–1966	Interests are assigned to Hudson Bay Exploration and Development (HBED). The NIP claims lapse in 1966.
1979–1980	J.J. Studer stakes claim CB 10186 over the Puffy Lake prospect, and it is transferred to Granges Exploration AB (Granges). On June 4, 1980 Maverick Mountain Resources Ltd. (Maverick) options a 60% interest from Granges.
1984–1985	Exploration, development and mining agreement (June 29, 1984). Granges transfers 100% to Maverick on December 16, 1985. An option by Homestake (1985) is terminated in January 1986.
1986	Various minority working-interest financing agreements are made. Maverick and Pioneer Metals Corporation merge under the Pioneer name (September 1986).
1987	A third group of 19 claim blocks is added. CB 10186 is converted to a mining lease (October 30, 1987).
2010	Auriga Gold Corp. (renamed Minnova Corp. in 2014) acquires 100% of the PL Mine and PL Property, and Pioneer's 54% interest in Nokomis, from Pioneer Metals ULC (a Barrick subsidiary), subject to the sliding-scale NSR. The transfer of ML065 is registered on October 14, 2010
2011	Auriga acquires the remaining 46% of Nokomis from Claude Resources Inc. (November 22, 2011).
2016–2017	Minnova stakes the MGC 16 and MIT claims and relinquishes six original claims (2018 FS, p. 30).

6.2 Previous Exploration

HBED drilled 14 holes in 1960 to test a horizontal-loop EM conductor. Sherritt Gordon flew an airborne EM and magnetic survey in 1972. Maverick drilled 30 holes in 1981 and 29 holes in 1984–1985. These extended the Main Zone to a strike length of 1,037 m and discovered a parallel zone 275 m to the north. A further 29 holes in 1985 traced the Main Zone to 183 m vertical depth and identified the Sherridon, Upper Main, Main and Lower Main lenses. In 1986, 75 holes (9,145 m) were drilled. Canadian Mine Development started a C$2 million decline, drifting, trial stoping and bulk sampling program in May 1986. Pioneer completed 225 holes in 1987 and 8 infill holes (633 m) in 1994. In total, about 410 historical holes tested the PL deposit (2018 FS, Tables 6.2 and 6.3). Earlier studies included a feasibility study by Kilborn Engineering (B.C.) Ltd. (January 1987), reactivation studies by Kilborn (1992), and the Kilborn (1993) feasibility study. The property was dormant from 1994 to 2010.

6.3 Historical Production

Pioneer started building the mill in May 1987. Milling began on December 4, 1987, and the first doré was poured on December 15, 1987. Pioneer suspended the operation in March 1989 because it failed to reach commercial production. The plant was placed on care and maintenance in April 1989, and the mine was closed by June 1989. Later in 1989 (August 8 to 11), after the operation had been placed on care and maintenance, forest fires in the Sherridon and Puffy Lake area damaged the power line and the freshwater intake and tailings pipelines; the crusher, mill and office complex had a firebreak and escaped damage (2018 FS, pp. 43 and 47; AECOM, 2014; Manitoba Mineral Inventory File 0984). The 2018 FS reports production of 21,000 oz of gold in 1988 and 8,500 oz in 1989 from about 350,000 t of mill feed, against a target of 72,000 oz per year. At its peak the operation employed 170 people (2018 FS, pp. 47 and 60).

Table 6-2: Summary of historical production, 1987–1989

Period	Tonnes milled	Head grade (g/t Au)	Gold produced (oz)	Recovery (%)
December 1987	n/r	n/r	450	72.9
January–December 1988	291,800	about 2.0 to 4.1 (monthly)	19,746	73 to 89 (monthly)
January–May 1989	71,918 (January–March)	2.4 to 3.4 (monthly)	8,284	84 to 89 (monthly)
Total (monthly records)	**about 363,700**		**about 28,480**	**about 83 overall**
2018 FS summary	about 350,000		about 29,500 (21,000 in 1988; 8,500 in 1989)	

6.4 Historical Resource Estimates (pre-2010)

Table 6-3 summarizes the historical estimates prepared by previous owners before Minnova acquired the property. They are historical estimates as defined in NI 43-101. Their categories do not conform to the CIM 2014 Definition Standards, and no technical documentation is available for most of them (2018 FS, pp. 53–57).

Table 6-3: Historical estimates for the PL deposit (not current, not relied upon)

Source / date	Category as reported	Tonnes	Au (g/t)	Comment
Maverick, 1981	Uncategorized	428,000	7.68	30 drillholes
Pioneer, 1985	Uncategorized	603,000	6.86	58 drillholes
Pioneer, May 31, 1986	Uncategorized	696,700	7.99	154 drillholes
Pioneer (B. Simmons), April 1, 1987	Uncategorized	2,604,500	7.54	Polygonal; SG 2.77
Piteau, April 1, 1987 (review of the Pioneer estimate)	Uncategorized	1,192,000	6.23	
Pioneer, 1989 (post-production; 3.5 g/t cut-off)	'Indicated' / 'Inferred'	1,346,177 / 883,718	7.86 / 6.35	
Kilborn, 1993 (3.5 g/t cut-off)	'Proven and Probable' / 'Possible' resource	1,346,200 / 883,700	7.86 / 7.11	Inverse distance; Datamine
Tonto Mining, 1993	'Proven and Probable' ore reserve	855,000	6.7	15% dilution; 20% pillar loss

A QP has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves. Neither ABGM nor Minnova is treating them as current Mineral Resources or Mineral Reserves, and they should not be relied upon. They are presented for historical context only and are superseded by the 2026 MRE. No further work is planned to verify them.

6.5 Previous NI 43-101 Estimates (Superseded)

Minnova (and, before 2014, Auriga) has disclosed earlier NI 43-101 Mineral Resource estimates for the PL deposit: P&E Mining Consultants (August 2011, and April 2014) and CSA Global (L. McGarry, effective October 31, 2017). The 2017 estimate was reported at a 2.5 g/t Au cut-off, a gold price of US$1,250/oz and a density of 2.81 t/m³. It comprised 1,481 kt at 5.93 g/t Au (282,500 oz) Measured and Indicated and 1,846 kt at 5.08 g/t Au (301,700 oz) Inferred. The 2018 FS reported Proven and Probable Mineral Reserves of 1,266,438 t at 6.34 g/t Au, based on that estimate.

These estimates were prepared for the Company and are therefore not 'historical estimates' as defined in NI 43-101. However, they are superseded by the 2026 MRE and are no longer current. The 2018 FS Mineral Reserves are not current Mineral Reserves, and the 2018 FS is no longer current. The 2026 MRE is not directly comparable with the 2017 estimate because of its lower reporting cut-off grades, higher gold price, open-pit and MSO reporting constraints, and additional drilling.

The Nokomis deposit has a 2014 estimate by ACA Howe International (L. McGarry, effective April 15, 2014). It reported 371,000 t at 3.41 g/t Au Indicated and 247,000 t at 2.41 g/t Au Inferred at a 0.6 g/t cut-off (2018 FS, Table 14.14). This estimate is historical in the context of this Technical Report, has not been verified by the QP, is not relied upon, and is not part of the 2026 MRE.

6.6 Previous Technical Studies

- Kilborn Engineering (1987, 1993): feasibility studies for Pioneer.

- ACA Howe International: PEAs for Auriga and Minnova in 2011, 2012 and 2014 (the 2014 study also covered Nokomis).
- A-Z Mining Professionals Ltd. and CSA Global: Feasibility Study – PL Gold Project, dated April 23, 2018 and effective October 31, 2017. This study is historical and superseded. No economic results from it are relied upon in this Technical Report.

6.7 Minnova Work, 2010–2026

Table 6-4: Summary of Auriga and Minnova work on the PL Property

Year(s)	Work	Source
2010–2011	Phase 1 drilling: 47 holes (5,816 m) to confirm the historical resources	2018 FS, §10.3.1
2011	LiDAR survey (6,171 ha); P&E Mineral Resource estimate (August 2011)	2018 FS; MRE chapter, Item 9
2011–2012	Phase 2 drilling: 57 holes (5,211 m); airborne VTEM and magnetic survey (2012; about 866 line-km); power-line upgrade (about C$550,000)	2018 FS, §10.3.2, §5.3.5.4
2013–2014	Phase 3 drilling: 4 holes (2013); P&E estimate update (April 2014); ACA Howe updated PEA (July 2014)	2018 FS; July 9, 2014 news release
2016–2017	Phase 4 (2 holes) and Phase 5 (48 holes, 8,523 m) infill drilling; 2017 field program (88 rock samples, 6 gold showings, discovery of PL North); CSA 2017 MRE; 2018 FS	2018 FS, §10.3.4–10.3.5; MRE chapter, Item 9
2018	Metallic screen fire assay (MSFA) check-assay program on historical core. The first 10 samples averaged 25% above the original fire assays	Sep 27 and Nov 12, 2018 news releases
2019	No drilling or field work on the PL Property	Minnova Corp.
2020	13-hole step-out program on ML065 (about 2,000 m planned), mapping and prospecting of the footwall tonalite	Jul 30 and Dec 29, 2020 news releases
2021	Winter drilling at PL North (2,500 m planned), follow-up holes (642 m), including M-21-048x (101.6 g/t Au over 0.50 m); agreement to buy the power line from Manitoba Hydro (purchase completed December 2021)	Jan 22, May 27, Nov 12 and Dec 13, 2021 news releases
2022–2024	Review of the database for LCT pegmatites (2023); surface sampling by Minnova personnel; no drilling	Feb 14, 2023 news release; Minnova Corp.
2025–2026	Infill and expansion drilling: more than 13,500 m in 132 holes (September 2025 to April 2026), including 18 holes at PL North; 126 of these holes (13,092 m) are in the MRE database and the remaining six will be incorporated in the planned PFS; database clean-up and 2026 MRE	Jul 29 and Sep 10, 2026 news releases

7 GEOLOGICAL SETTING & MINERALIZATION

7.1 Regional Geology

The PL Gold Mine is in central Manitoba near the boundary between the Kisseynew and Flin Flon domains of the Paleoproterozoic Trans-Hudson Orogen. Regional lithotectonic units have been metamorphosed to middle- to upper-amphibolite facies, with metamorphic grade generally increasing northward toward the

Kisseynew Domain. The Flin Flon Domain forms an approximately east-trending, 230 km by 80 km belt of complexly folded and metamorphosed volcanic, sedimentary and intrusive rocks. It is bounded to the north by the Kisseynew Domain, to the south by Phanerozoic cover, to the east by Archean rocks of the Superior Province and, to the west by the Tabbernor Fault and granitoid rocks of the Glennie Domain.

The project area includes supracrustal rocks correlated with the Amisk volcanic assemblage and the Burntwood and Missi sedimentary successions, together with granitoid gneisses of the Sherridon-Hutchinson Lake Complex. The regional architecture records accretion, high-grade metamorphism, polyphase folding and later ductile to brittle-ductile deformation. This structural history is fundamental to the PL deposit because the gold-bearing vein and sulphide system is localized within later shear zones that transect or reactivate the metamorphic fabric.

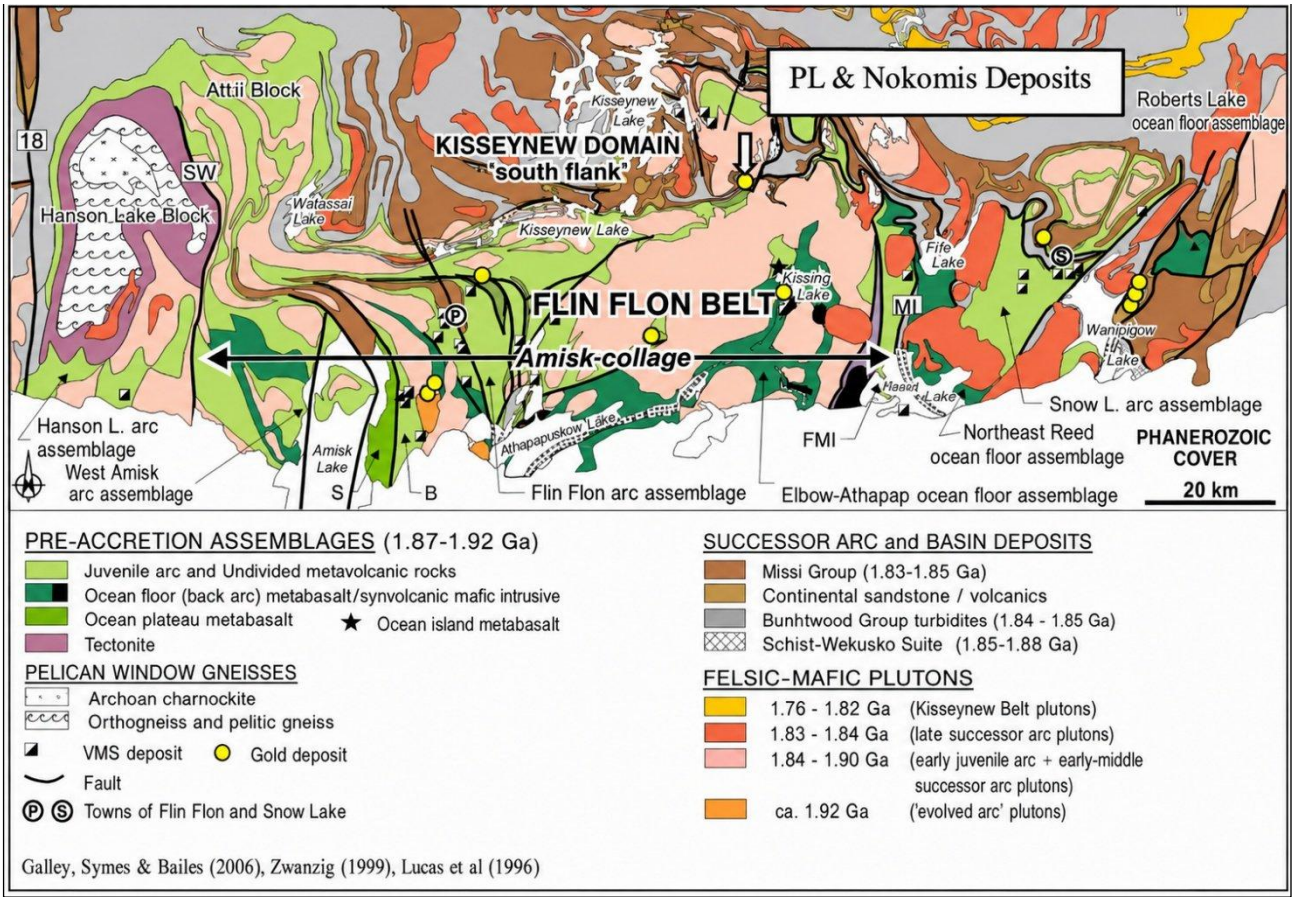


Figure 7-1: Regional geological setting of the PL and Nokomis deposits. Source: A-Z Mining Professionals Ltd., 2018, modified after regional Manitoba mapping.

7.2 Local Geology

At deposit scale, the PL area is underlain by strongly deformed para- and orthogneisses interpreted to represent Amisk metavolcanic rocks, Missi and Burntwood metasedimentary rocks and intermediate to felsic intrusive rocks. Four broad lithological packages are material to the deposit interpretation. Structurally, the lowest is a blue-grey tonalitic gneiss with a strong metamorphic fabric. Above it is an approximately 85 m

package of transposed metavolcanic and metagreywacke rocks that hosts the principal mineralized shear zones. This is overlain by an approximately 12 m polymictic metaconglomerate correlated with the basal Missi succession, followed by a thicker sequence of biotite-quartz, biotite-garnet and amphibolite gneisses with local calc-silicate and leucocratic layers. A pink granitoid gneiss, historically termed the Ragged Lake Granite, occurs structurally above the succession and is separated from it by the Ragged Lake Fault Zone.

The tonalitic gneiss was historically used as a practical drilling termination in parts of the deposit. Subsequent drilling demonstrated that quartz-vein and gold mineralization can extend into the tonalite, particularly toward the northern part of the mineralized system. Consequently, the current interpretation is controlled by observed structure, veining, sulphide distribution and assay continuity rather than by a lithological contact alone.

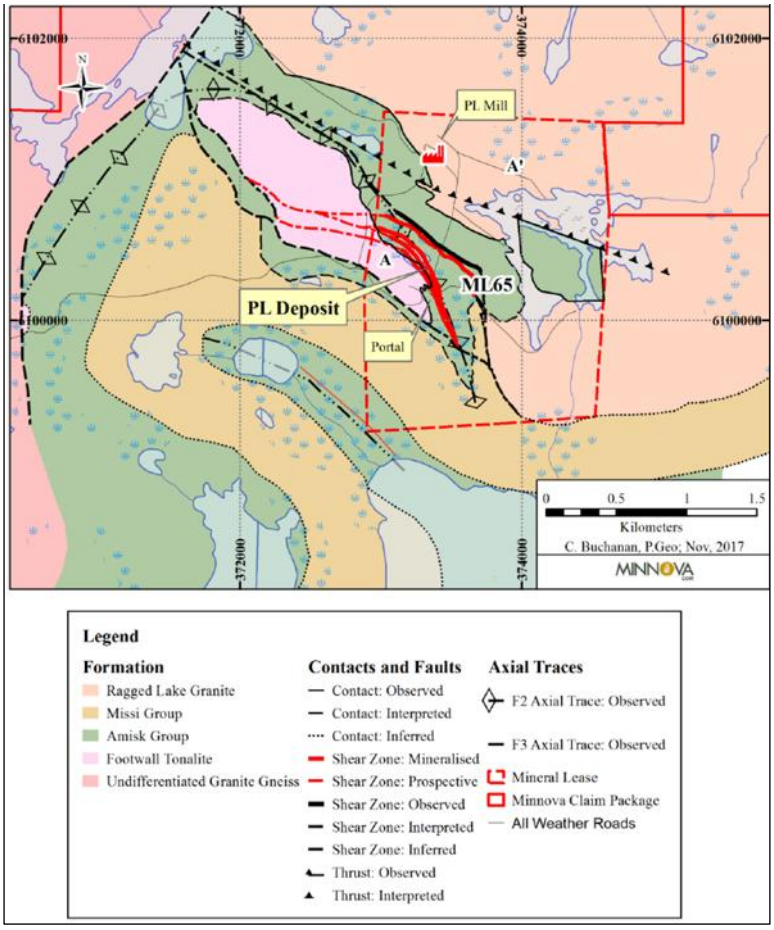


Figure 7-2: Deposit area geology, principal structures and mineral lease context. Source: Minnova Corp., 2017.

7.3 Structural Geology

The PL deposit lies on the northeast limb of the Puffy Lake Fold, a large, moderately plunging, reclined and asymmetric fold interpreted to be kinematically associated with sinistral displacement along the regional Nokomis structure. Earlier isoclinal folding was subsequently refolded, producing an interference geometry that locally preserves an upright but structurally repeated succession near the Amisk-Missi unconformity.

Later D3 reverse-sense brittle-ductile shear zones dismembered the fold limb and formed an imbricate to duplex-style structural array. The principal mineralized structures generally strike approximately N30°W and dip moderately, commonly about 30° to the northeast, subparallel to the regional foliation. Local changes in dip, competency contrasts, flexures, splays and intersection zones created dilational sites favourable for quartz-vein development and sulphide-gold deposition. The current 2026 vein framework retains the principal historical architecture while resolving additional splays and northern/central lenses at the scale supported by the updated drilling.

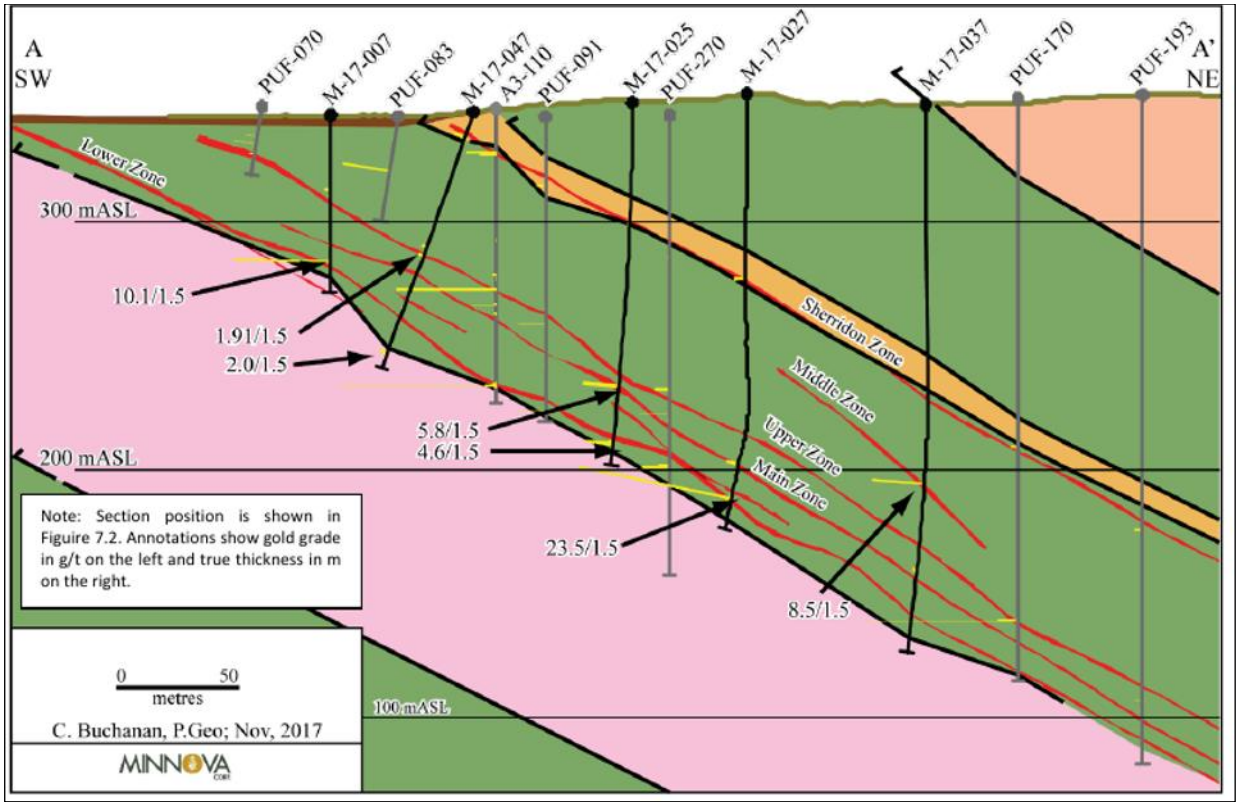


Figure 7-3: Historical geological cross-section through the PL deposit showing the relationship between lithology, mineralized shears and drilling. Source: Minnova Corp., 2017.

7.4 Mineralization Controls

Gold mineralization is structurally controlled and occurs within an array of moderately northeast-dipping shear zones historically grouped as the Sherridon, Upper, Main and Lower zones. The mineralized system subcrops beneath a northwest-southeast trending swamp and has been tested by drilling to approximately 500 m vertical depth, corresponding to roughly 1,200 m down dip in the historical interpretation. Upper, Main and Lower mineralization is hosted mainly by amphibolite-facies mafic metavolcanic rocks and metasedimentary gneisses close to the sheared tonalite contact, with the Lower system locally extending into tonalitic gneiss toward the north.

The mineralized shear zones commonly range from approximately 1.3 m to 2.5 m in width and contain variable quartz veining. Individual fault-fill and tension veins range from centimetre scale to greater than 1

m, with historical descriptions indicating common vein thicknesses of approximately 25 cm to 35 cm. The dominant sulphide assemblage comprises arsenopyrite, pyrrhotite and pyrite with local galena, chalcopyrite and sphalerite. Visible gold has been observed locally.

Alteration includes biotite, muscovite, carbonate, chlorite and tourmaline, with local calc-silicate minerals. Mineralized zones pinch and swell, split into parallel lenses and are locally offset or reoriented by the structural framework.

The Lower Zone is generally the most laterally continuous of the major structural families. Upper, Main and Middle zones are more discontinuous and locally thinner. At the scale of resource modelling, continuity is therefore established by combining geological logging, wireframe geometry, drill intercepts and assay continuity rather than by projecting grade alone.

7.5 Deposit Model

The PL deposit is best described as a structurally controlled, metamorphosed lode-gold system hosted by amphibolite-facies volcanic, sedimentary and intrusive gneisses. The historical descriptor "mesothermal" is broadly consistent with the metamorphic and structural setting but is not used as a substitute for deposit-scale observations. The practical exploration and Mineral Resource model is a multi-lens shear- and vein-hosted system in which grade continuity follows the interpreted structural framework.

The 2017 Mineral Resource interpretation represented 14 hard-boundary mineralized domains. The 2026 interpretation retains the major historical structural families and introduces revised domain names and additional splays supported by the updated drilling. The plan-view projection of the current model shows PL North, North, Centre, South, Lower-Lower and other structural families, together with the current block-model footprint and selected drillholes. These surface traces represent the interpreted intersections of three-dimensional mineralized structures with the topographic surface and are intended as a geological framework rather than mapped surface veins.

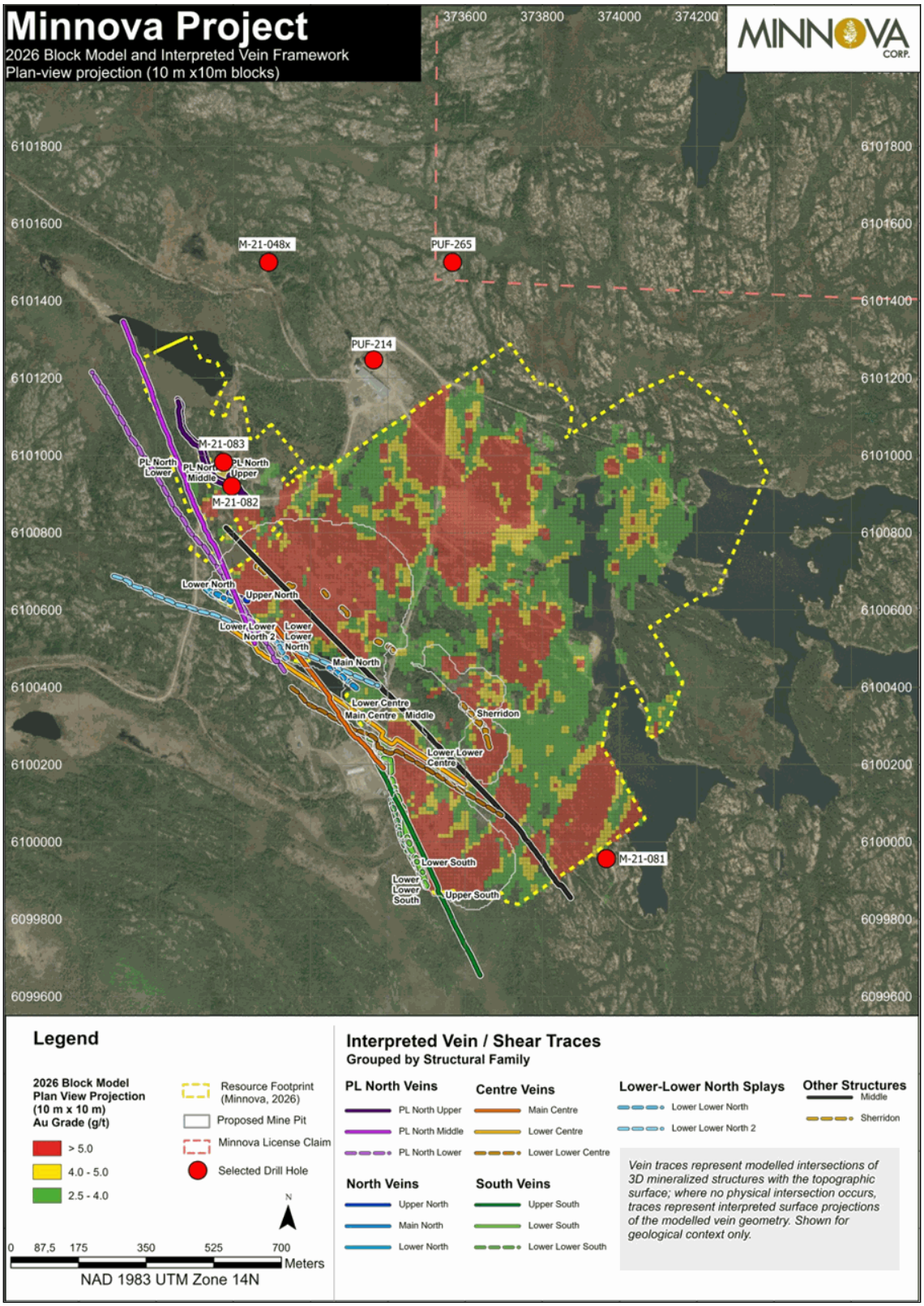


Figure 7-4: 2026 block-model plan projection and interpreted vein/shear framework. Source: Minnova Corp. / ABGM, 2026

8 DEPOSIT TYPES

The PL deposit is interpreted as a structurally controlled, metamorphosed lode-gold system consistent with the broader orogenic gold deposit model. Gold mineralization is associated primarily with shear zones and quartz-sulphide vein systems developed within amphibolite-facies volcanic, sedimentary and intrusive rocks. Mineralization is controlled principally by the structural framework rather than by a single stratigraphic or lithological horizon.

The deposit model comprises multiple moderately northeast-dipping mineralized structures that locally split into parallel lenses and splays and may pinch, swell or change orientation. Quartz veining and sulphide mineralization, particularly arsenopyrite, pyrrhotite and pyrite, are closely associated with the mineralized structures. Structural flexures, dilation zones, competency contrasts and structural intersections are interpreted as favourable sites for gold deposition.

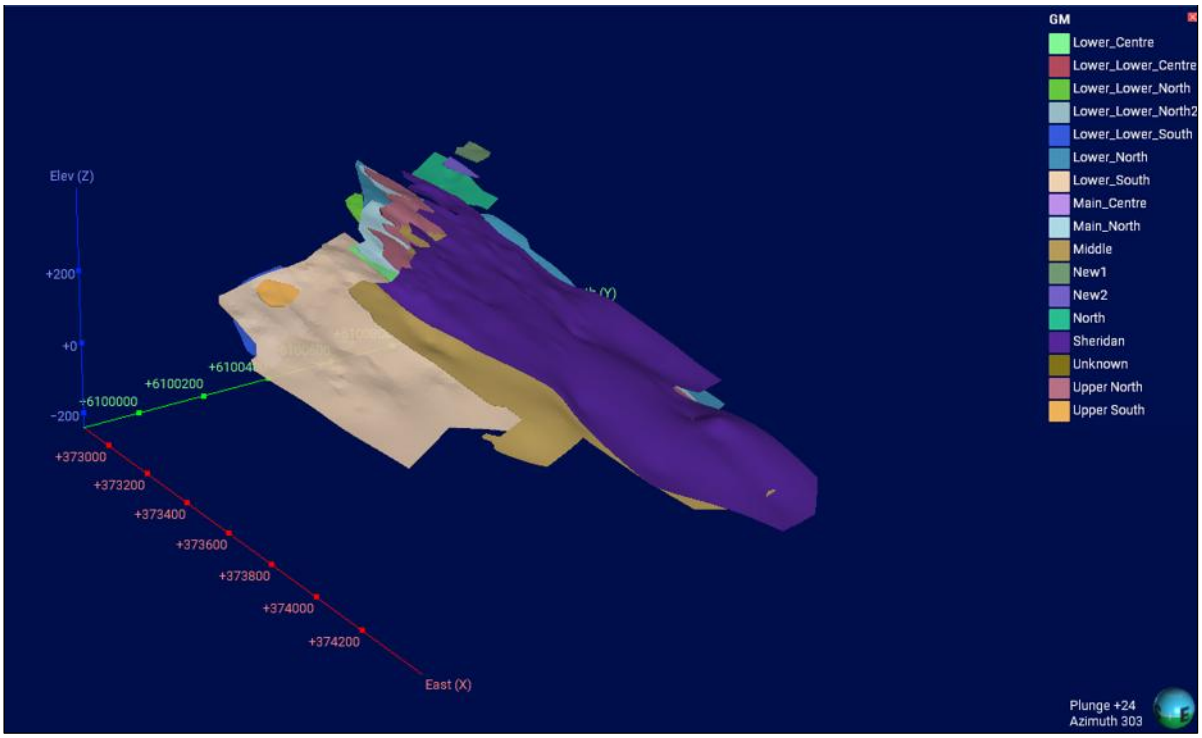


Figure 8-1: Oblique View of the Vein/Shear Framework. Source ABGM 2026.

8.1 Deposit Style (Conceptual models underpinning the exploration.)

The PL deposit comprises a multi-lens system of structurally controlled gold mineralization hosted mainly within amphibolite-facies volcanic, sedimentary and intrusive gneisses. Mineralization occurs principally within moderately northeast-dipping shear zones and associated quartz-sulphide veins, with the principal structural families represented by the Sherridon, Upper, Main and Lower zones.

Mineralized structures are generally tabular to lens-shaped and may pinch and swell, split into parallel lenses or splays, and locally offset or change orientation. Quartz veins range from centimetre scale to greater than

1 m in thickness and are associated mainly with arsenopyrite, pyrrhotite and pyrite, with lesser galena, chalcopyrite and sphalerite. Alteration includes biotite, muscovite, carbonate, chlorite and tourmaline.

Although mineralization is concentrated near the sheared tonalite contact, drilling has demonstrated that mineralization can extend into tonalitic gneiss. The deposit style is therefore defined primarily by the structural framework and associated veining and sulphide distribution rather than by lithology alone.

9 EXPLORATION

9.1 Surface Exploration

Gold mineralization at PL was discovered during follow-up of airborne geophysical anomalies in 1960. The property was subsequently explored by a series of operators through drilling, geological mapping, trenching, geophysics and underground development before the Puffy Lake mine was placed into production in the late 1980s. Since closure, exploration has focused on consolidating the historical database, improving the geological and structural interpretation, testing extensions of known mineralized zones and identifying satellite targets.

Modern surface programs have included a 6,171 ha LiDAR survey completed in 2011, property-scale geophysics, geological mapping, prospecting and rock sampling. The 2017 field program updated surface geology, collected 88 rock samples and documented six gold showings within approximately 1.8 km of the mill. These programs provide important structural and lithological context, but drillhole data remain the principal support for the Mineral Resource model.

9.2 Geophysics

Airborne and ground geophysical methods have been used throughout the history of the project to define conductive and magnetic trends, structural corridors and prospective iron-formation or sulphide-bearing targets. A 2012 airborne VTEM and magnetic survey covered approximately 66 km² on 866 line-km. The original survey grids were not available for this PEA, so Figures 9-1 and 9-2 show the interpreted magnetic lineaments and VTEM conductor axes (Hogg, 2011) clipped to current Minnova tenure, rather than the gridded data. The interpretation is useful at property scale for tracing lithological contacts and structures beneath overburden and lakes and for identifying targets on Minnova tenure outside the PL deposit footprint.

Within the immediate resource area, geophysics is used as supporting geological evidence rather than as a direct grade estimator. The current deposit model is constrained primarily by drillhole geology, assays, interpreted shear/vein geometry and, where available, historical underground information.

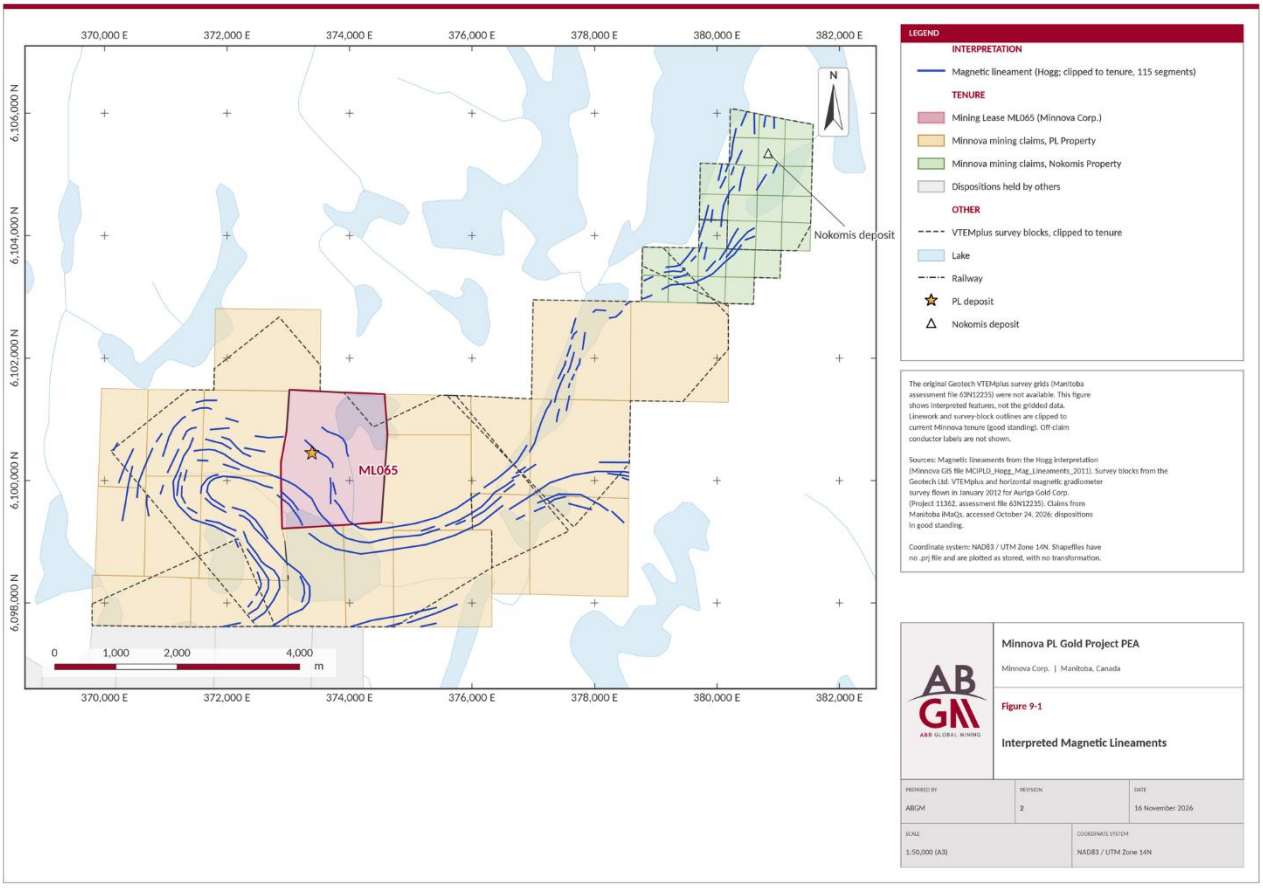


Figure 9-1: Interpreted magnetic lineaments over the current PL and Nokomis mineral tenure. The linework shown is clipped to current Minnova tenure. Source: Hogg interpretation (Minnova GIS file MCIPLD_Hogg_Mag_Lineaments_2011); survey blocks from the Geotech Ltd. helicopter-borne VTEMplus and horizontal magnetic gradiometer survey flown in January 2012 for Auriga Gold Corp., Maverick Gold Project (Geotech Project 11362, Manitoba assessment file 63N12235); claims from Manitoba iMaQs, accessed October 24, 2026.

Note: Interpretation linework only; the underlying magnetic and VTEM grids are not reproduced.

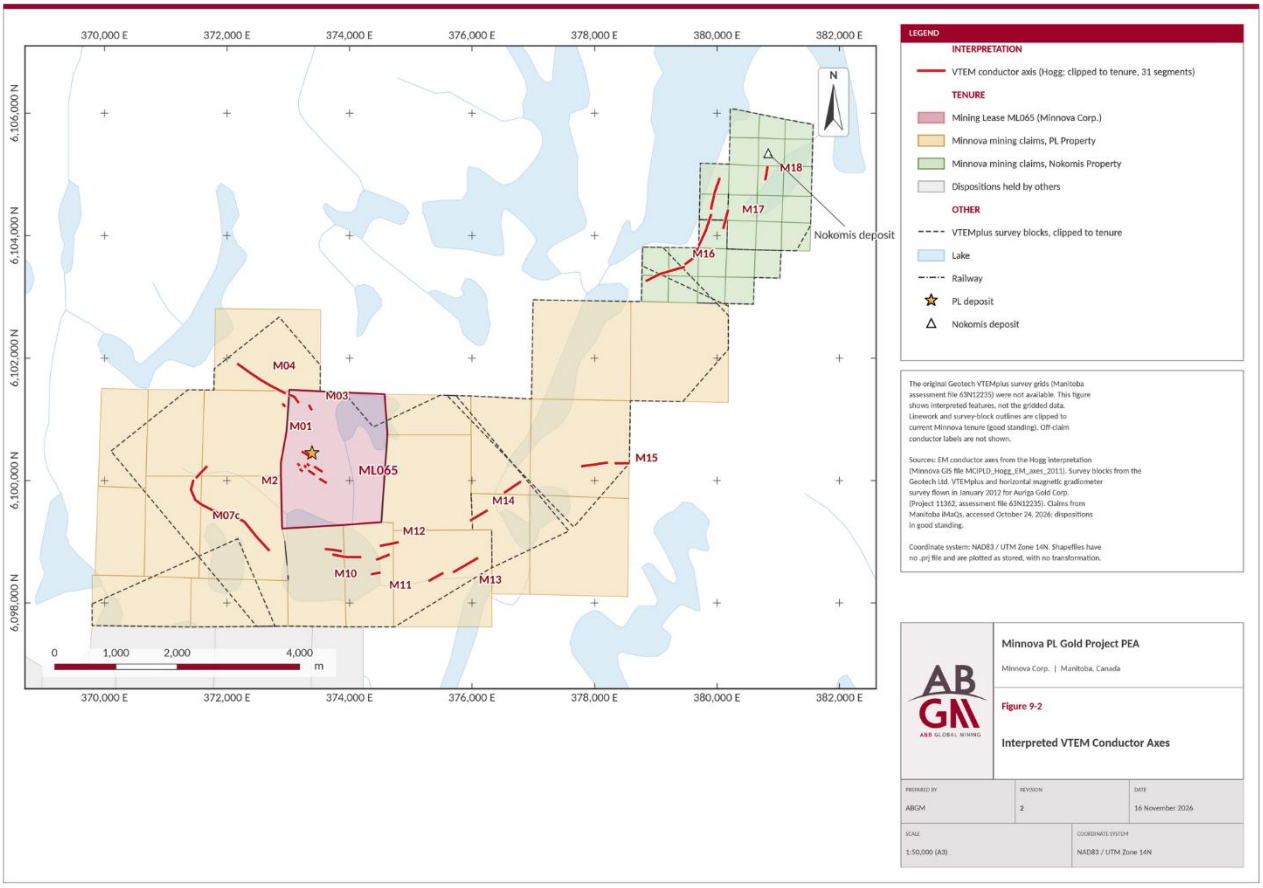


Figure 9-2: Interpreted VTEM conductor axes over the current PL and Nokomis mineral tenure. The linework shown is clipped to current Minnova tenure. Source: Hogg interpretation (Minnova GIS file MCIPLD_Hogg_EM_axes_2011); survey blocks from the Geotech Ltd. helicopter-borne VTEMplus and horizontal magnetic gradiometer survey flown in January 2012 for Auriga Gold Corp., Maverick Gold Project (Geotech Project 11362, Manitoba assessment file 63N12235); claims from Manitoba iMaQs, accessed October 24, 2026.

Note: Interpretation linework only; the underlying magnetic and VTEM grids are not reproduced. Conductor labels are shown only for axes on current Minnova tenure.

9.3 Mapping

Property and deposit-scale mapping have progressively refined the relationships among Amisk metavolcanic rocks, Missi/Burntwood metasedimentary units, tonalitic and granitoid gneisses and the principal ductile to brittle-ductile structures. The 2017 geological compilation provides the current regional and deposit-scale mapping framework used in this chapter. Subsequent drilling and three-dimensional modelling have refined the internal geometry of the mineralized zones without materially changing the regional geological setting.

9.4 Sampling Programs

Surface sampling programs have included grab and channel-style rock sampling during historical and modern exploration campaigns. These data are useful for target generation and geological interpretation but are not used interchangeably with drill-core assays in the Mineral Resource estimate. The current Mineral Resource

database is based on drillhole intervals that have been desurveyed, domained and composited within the three-dimensional mineralized framework.

9.5 Interpretation

Exploration targets are developed along extensions and splays of the known mineralized shear system, particularly where favourable structures intersect competent lithological contacts or where geophysical responses indicate prospective buried geology. The 2025-2026 infill program was directed principally at shallow mineralization within the proposed open-pit area and was intentionally concentrated to approximately 200 m depth. Future drilling is expected to test down-dip continuity, convert lower-confidence material and evaluate structurally related targets beyond the present resource footprint.

10 DRILLING

10.1 Historical Drilling

The PL deposit has a long drilling history extending from early exploration through the 1980s mine-development period and subsequent Auriga and Minnova programs. The 2017 feasibility-study database included 416 historical Maverick/Pioneer holes drilled between 1981 and 1988, predominantly on a northwest-southeast grid at approximately 30 m to 60 m spacing. Early PUF-series holes included AQ core, with later historical drilling generally using NQ core. Most historical holes were vertical, with selected inclined holes used to test geometry and true thickness.

Auriga drilling from 2010 to 2013 and Minnova drilling in 2016-2017 materially improved the density and quality of information around the deposit. The historical study documents NQTW core for the Auriga programs and NQ core for the Minnova programs, together with progressively improved collar and downhole survey methods. These programs formed the basis of the 2017 geological and Mineral Resource model and remain an important component of the 2026 database.

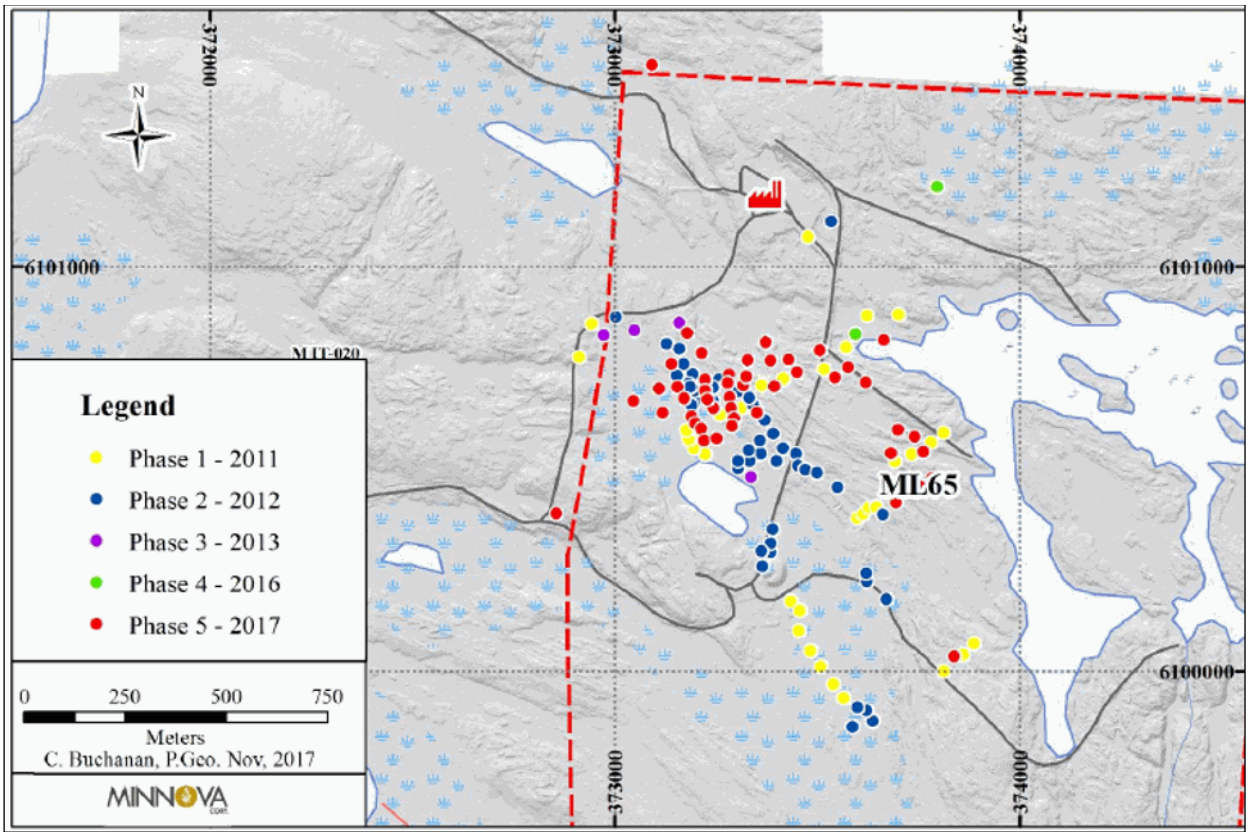


Figure 10-1: Historical Minnova/Auriga drillhole locations by campaign. Source: Minnova Corp., 2017.

10.2 Recent Drilling

The cleaned August 2026 database contains 1,005 unique collar records across the PL and Nokomis project areas. Of these, 147 collars are coded to Nokomis, and 858 are PL-related. Derived hole lengths, based on the maximum recorded depth in the available depth-bearing tables, total approximately 146,743 m. The database identifies 187 holes and approximately 23,431 m associated with the 2026 resource-update population, including 126 holes totalling approximately 13,092 m drilled during 2025-2026. The 2025–2026 program comprised 132 holes in total; the remaining six holes are not in the database used for the 2026 MRE and will be incorporated in the planned pre-feasibility study (PFS).

The 2025-2026 program was focused on near-surface mineralization within and around the proposed open-pit resource area. The campaign was intentionally limited to a maximum depth of approximately 200 m and was designed primarily to increase drill density and improve confidence in shallow mineralized structures. The resulting data were incorporated into the updated geological interpretation and 2026 Mineral Resource model.

Table 10-1: Cleaned drillhole database summary

Database population	Holes	Drilling. Metres (m)	Campaign Duration
MCINOK	147	14,016.73	15/May/1958 to 28/February/2012
PMPLD	455	80,413.64	26/September/1980 to 02/January/1989
ARPLD	108	11,724.88	01/January/2011 to 02/August/2013
MCIPLD	295	40,489.95	01/April/2016 to 15/March/2026

10.3 Drill Methods

Historical drilling was dominated by diamond core methods. The 2017 technical report records AQ core for the earliest PUF-series holes, NQ for later historical drilling, NQTW for Auriga drilling and NQ for Minnova's 2016-2017 drilling. Core logging, sampling and geological coding evolved through time and the historical data therefore represent several generations of procedures and operators.

The current cleaned database is suitable for spatial reconstruction of the drilling and assay intervals, but the database tables supplied for this update do not contain a complete set of recent contractor, rig, core-diameter or core-orientation metadata. These operational records should remain linked to the master drillhole archive so that drill method and orientation can be traced by campaign.

10.4 Core Recovery

Core recovery and geotechnical data were reviewed as part of the 2026 drillhole database validation. The available dataset contains 504 intervals from 41 drillholes, representing approximately 2,284 m of drilled interval. Intervals are predominantly coded as competent core with greater than 80% recovery (CO), although RQD varies considerably both between and within drillholes. Several holes show generally improved RQD with depth, while lower-RQD and more highly fractured intervals occur locally, particularly in shallower portions of the holes. Fracture counts generally decrease in intervals with higher RQD, providing supporting information on changes in rock-mass quality. The data also contains isolated RQD values exceeding 100%, which are considered data-entry or calculation anomalies. These values have been validated against the original logging records for the final database acceptance. Overall, the available data provide supporting information for geological and geotechnical interpretation.

10.5 Collar Surveys

Historical collar positions were established using methods appropriate to the period, ranging from theodolite and chain-and-compass surveys to professional survey control for later campaigns. The 2017 report records professional collar surveying for Auriga and Minnova drilling. The current collar table contains complete easting, northing, and elevation coordinates for all 1,005 collars, and no duplicate collar identifiers were identified during the structural database review.

10.6 Downhole Surveys

The downhole survey data generally indicate consistent drillhole trajectories, with most successive survey measurements showing relatively small changes in azimuth and dip. A small proportion of observations show unusually large changes and have therefore been flagged for review. Of the 1,005 drillholes reviewed, 794 drillholes were considered to have adequate downhole survey coverage and were subjected to further statistical screening. A total of 2,418 evaluable survey intervals were assessed for survey interval, azimuth change and dip change.

11 SAMPLE PREPARATION, ANALYSIS & SECURITY

11.1 Sampling Procedures

Historical and modern PL drilling programs used diamond-drill core that was geologically logged, marked for sampling and split or sawn before dispatching to commercial laboratories. The 2017 report provides the most complete description of historical sampling procedures. Sample intervals were selected to respect geological and mineralized boundaries, with quality-control samples inserted into the sample stream for the modern Auriga and Minnova campaigns.

The 2026 assay database contains interval from/to depths and a model gold field used in the resource workflow.

11.2 Laboratory Methods

For the 2010 to early-2017 drilling, the 2017 technical report documents sample preparation and analysis at Accurassay Laboratories. The documented preparation route included drying, crushing to approximately 85% passing 2 mm, riffle splitting to a nominal 500 g subsample and pulverizing to approximately 90% passing 74 µm. Gold was analyzed by fire assay with an atomic-absorption finish, with gravimetric procedures used for over-range samples. Later 2017 samples were also analyzed by SGS and Activation Laboratories using 30 g fire-assay charges with atomic-absorption finishes and gravimetric over-range analysis. The 2025 / 2026 drill program relied on SRC Laboratories using 30 g fire-assay charges with atomic-absorption finishes and gravimetric over-range analysis. The laboratories were independent commercial facilities and were reported as appropriately accredited at the time of the work.

Table 11-1: Principal analytical methods by drilling program

Program / laboratory	Preparation / analytical method	Gold over-range
Auriga / Accurassay (2010-early 2017)	Dry, crush, split, pulverize; 50 g fire assay with AAS finish	Gravimetric
Minnova / SGS (2017)	Commercial sample preparation; 30 g fire assay with AA finish	Gravimetric
Minnova / Activation Laboratories (2017)	Commercial sample preparation; 30 g fire assay with AA finish	Gravimetric
Minnova / SRC Laboratories (2026)	Commercial sample preparation; 30 g fire assay with AA finish	Gravimetric

11.3 QA/QC Program

Samples from the 2025 / 2026 Minnova drilling program were analyzed by SRC Laboratories (Section 11.2). For the 2025 / 2026 drilling campaign, Minnova inserted certified reference materials, blanks and duplicates into the sample stream sent to SRC Geoanalytical Laboratories at an average rate of one control sample per ten samples. In the QP's opinion, the SRC assays are reliable for Mineral Resource estimation.

The 2017 Minnova drilling program incorporated certified reference materials, blanks and quarter-core duplicates at an overall frequency of approximately one quality-control insertion per eight routine samples. Reference standards were generally reported within accepted control limits, while blank and duplicate performance reflected the expected challenges associated with coarse and locally nuggety gold. Six SGS blank samples exceeded 0.025 g/t Au following high-grade material, although none exceeded 0.1 g/t Au. Quarter-core duplicates displayed scatter consistent with heterogeneous gold distribution.

Auriga programs from 2010 to 2013 used a QA/QC insertion frequency of approximately one in twenty. The original gneiss blank material was found to contain gold and was subsequently replaced with a pre-pulverized barren blank. Historical pre-2010 drilling has more limited modern QA/QC documentation and is therefore assessed together with spatial support, geological continuity and later verification work when considering its influence on the Mineral Resource.

11.4 Data Integrity

The 2026 assay table contains 31,592 assay intervals from 873 holes. Structural checks identified no duplicate hole-sample keys, duplicate hole-from-to interval keys, negative model grades, or non-positive sample intervals. The model gold field ranges from 0.0 to 190.8 g/t Au. Database integrity checks confirm interval and key consistency but are not equivalent to laboratory-certificate verification; analytical provenance remains controlled through the original certificates, QA/QC records and database change history.

12 DATA VERIFICATION

12.1 2026 Qualified Person Site Visit

Nick King visited the PL site from 24 to 27 April 2026. The site visit included reviews of drill-core logging, sampling procedures and drilling-data management relevant to the current geological and Mineral Resource work. The visit provides direct project familiarity for the 2026 update and supplements the historical verification described in the 2017 technical report.

12.2 Historical Verification

During the 2017 study, Leon McGarry inspected representative drill core, located 13 drill collars and collected nine quarter-core verification samples. Differences between handheld GPS observations and surveyed collar positions were approximately 2.0 m to 5.8 m, consistent with the expected precision of the field verification

method. Verification samples confirmed the presence of gold mineralization, although individual paired results showed variability consistent with the nuggety nature of mineralization. Approximately 10% of the available historical digital gold assays were compared with scanned logs, with no significant transcription errors reported in that review.

12.3 Database Verification

The August 2026 database construction was conducted by Mr. Ulrich van der Heyde (Senior Mineral Resource Geologist, ABGM) and was reviewed and signed off by Mr. Nick King. The construction and cleanup were conducted structurally across collar, survey, assay, lithology, structural and wireframe-assay tables. The review focused on unique identifiers, interval consistency, coordinate completeness, duplicate keys, range checks and relationships among tables. The resulting database is internally coherent for geological reconstruction and modelling, subject to the normal limitations of combining multiple generations of historical and modern information.

Table 12-1: Cleaned database structure reviewed for the 2026 update

Table / file	Rows	Unique holes	Principal observations
Collar	1,005	1,005	Complete coordinates; no duplicate hole IDs
Survey	2,632	1,005	No duplicate hole-depth station keys
Assay	31,592	873	Complete model-grade intervals; no duplicate interval keys
Lithology	6,392	1,005	Standardized and original lithology fields; SG field present
Structure	7,194	168	Orientation fields present but unevenly populated

12.4 Data Quality Considerations

The database integrates drilling completed over several decades and therefore includes campaign-to-campaign differences in drilling, survey, logging, sampling, analytical and QA/QC procedures. Historical holes with limited downhole-survey or QA/QC information are not automatically excluded; their suitability is assessed in the context of geological continuity, nearby modern drilling and the confidence assigned to the resulting block estimates. Current model interpretation should maintain traceability from source assay to composite, domain, estimate and reported block.

Core-recovery, analytical-certificate and recent drill-method metadata are maintained outside some of the cleaned modelling tables. Those records form part of the project QA archive and should be retained with the signed technical report package so that material Mineral Resource inputs remain auditable.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The existing PL Mill was built in 1987 and operated from January 1988 to April 1989. The nameplate rating of the Mill is 1,000 t/d and rates of up to 1,500 t/d for short periods were reported, although average daily production was generally well below 1,000 t/d.

The process consisted of conventional two stage crushing and rod/ball mill grinding to a target grind of 100µm followed by flotation to produce a sulphide concentrate containing gold. A jig in the grinding circuit recovered coarse free gold. The two concentrate were reground and leached in a conventional cyanidation circuit using drum filtration for liquid/solid separation followed by Merrill-Crowe recovery of gold from solution. Precipitated gold was calcined and smelted to yield a doré gold product.

Flotation tailings and cyanide circuit filtration tailings were combined and discharged directly to the submerged tailings storage facility. A barren solution bleed to tailings was required and was treated with hydrogen peroxide to destroy cyanide prior to discharge.

13.1 Historical Metallurgy

Information in this section was taken from the "Technical Report on Puffy Lake Mine," dated June 30, 2006 and is a summary of all testwork programs employed prior to mine development.

On April 13, 1981, Britton Research Limited ("Britton") reported on a series of metallurgical tests on a Sample of Gold Ore from Puffy Lake, which included grinding, cyanidation, and flotation tests. The test results were summarized as follows:

- A composite sample was made up from one hundred three (103) diamond drill core assay rejects. It had an average gold assay of 7.1 g/t. Due to the erratic distribution of coarse gold in the sample, individual assays, including calculated test heads, ranged from 4.4 to 12.3 g/t.
- The sample had a Bond Work Index of 14.7 kWh per ton for grinding from 1.68 mm to 63% -75 µm.
- The sample had a specific gravity of 2.87 g/cm3.
- Although an appreciable amount of arsenopyrite was present in the sample, it was not refractory to cyanidation following fine grinding. Cyanidation was carried out for 72 hours, after grinding to 63% -75 µm and followed by amalgamation of the coarse fraction of the residue to remove coarse gold. This extracted 90.3% of the gold. Increasing the fineness to 87% -75 µm raised the extraction to 92.7%. A further increase to 95% -75 µm yielded an extraction rate of 95.4%.
- Three-stage rougher flotation, after grinding to 63% -75 µm, followed by amalgamation of the tailing, recovered 93.1% of the gold, in combined concentrates, assaying 73.7 g/t. Finer grinding to 95% -75 µm increased the recovery to 94.8%.
- Possible methods of treating the mineralized material include the following with gravity recovery incorporated:
    - Direct cyanidation after grinding.
    - Flotation, followed by shipment of the concentrate to a smelter.
    - Flotation followed by re-grinding and cyanidation of the concentrate.

- Indications were that at least 90% of the gold could be recovered by any of these methods.

On October 4, 1984, Lakefield Research ("Lakefield") reported on an investigation of the recovery of gold on a bulk sample from PL. The results are summarized as follows:

- Thirty (30) drums of material were received - weight 6,818 kg
- The overall 'calculated' grade was 4.17 g/t. Additional analyses on a representative head sample were: 2.2 g/t Ag; 0.37% Pb; 2.07% S; and 0.027% As.
- A pilot plant circuit was operated for a period of 19 hours over 3 days and 6.8 tonnes of ore was processed.
- **Grinding Circuit**. The average feed rate to the mill was 349 kg/hr and the net power consumption was 12.01 kW/t. The cyclone overflow was 71% passing 75 µm. The calculated work index was 13.4.
- **Flotation and Gravity Circuits**. Upon completion, the pilot plant was cleaned out and a mass balance was established.
    - Gold recovery during concentration was 95.9% with 4.1% reporting to the tailing product. Distribution of gold to various products was gravity concentrate: 77.2%; Pyrite concentrate: 11.8%; Arsenopyrite concentrate: 6.9%; and Tailing product: 4.1%.
- **Gold Recovery**. Concentrates were cyanided to extract gold from the gangue minerals, and this gold was recovered by electrolysis. During this process, some further gold losses were experienced. Final recoveries were stated as Gravity concentrate: 76.0%; Pyrite concentrate: 11.5%; Arsenopyrite concentrate: 5.7%; and Tailing product: 4.8%.

On November 25, 1986, Coastech reported on confirmatory metallurgical testwork. This work was summarized as follows:

The most significant estimate of head gold grade is by multiple sub-sample assay by fire/metallics. The mean gold grade for the twelve composite lots was 7.20 g/t. Coarse, free gold was evident due to high variance of individual head assays. There was no economically significant copper, lead, or zinc values indicated. Silver is of minor economic and metallurgical significance.

All of the composite lots responded to direct cyanidation. It is estimated that following optimization of the cyanidation parameters gold extractions typically > 95% could be expected resulting in tailings grading 0.30 to 0.50 grams per tonne. Reagent consumptions <0.2 kg NaCN per tonne and <1 kg CaO per tonne are expected with minor solution fouling by soluble coppers. High dissolved arsenic is expected in the pregnant solution.

Gravity concentration resulted in gold recoveries of 50 to 85% to a concentrate (uncleaned) grading >500 g Au/t and 30 to 40% As. Gravity and froth flotation combined bulk concentrate resulted in >95% gold recovery for all composite lots. All flotation products contained high levels of arsenic. A separate base metal flotation

concentrate for market is not viable. The most sensible processing route to pursue is a combined gravity flotation concentration followed by cyanidation of the combined or separate concentrates and conventional solution refining to bullion techniques.

Waste management should not present extraordinary processing measures.

All of the composite lots responded similarly to exploratory metallurgical testing. Future detailed design testwork should be limited to two composites, a high grade and a low grade.

13.1.1 Summary of Historical Testwork

The summary of all historical testwork is indicated in the following table.

Table 13-1: Gold Recovery Testwork Summary

Source	Basis	Au recovery
Britton Research, 1981 (103 core rejects, 7.1 g/t Au)	72 h cyanidation at 63% to 95% passing 200 mesh, with amalgamation of the coarse residue; rougher flotation	90.3 to 95.4%; 93.1 to 94.8%
Lakefield Research, 1984 (6.8 t bulk sample, 4.17 g/t Au, pilot plant)	Gravity + flotation concentrates; after cyanidation of the concentrates (sum of the stated final recoveries)	95.9%; 93.2%
Coastech Research, 1986 (12 composite lots, 7.20 g/t Au)	Direct cyanidation (expected after optimization); gravity + flotation bulk concentrate	>95%; >95% (all lots except Lot 1)
Kilborn feasibility study, 1987	Expected recovery	93%
PL mill, December 1987 to April 1989	Monthly range; overall to doré	72.9 to 89.2%; ~83%
ACA Howe PEA, 2014	Projected recovery, refurbished plant	90%
CSA Global MRE, 2017 (in the 2018 FS)	Cut-off grade input	95%
2018 FS	Expected plant recovery; pit optimization and cut-off grade input	90%

Source: 2018 FS; 2026 MRE.

The Britton sample contained appreciable arsenopyrite, which was not refractory to cyanidation. Coastech's gravity concentrates graded 30% to 40% As, and high dissolved arsenic and minor fouling by soluble copper were expected in the leach solution. Copper, lead and zinc were not economically significant.

13.2 Current Testwork

No further metallurgical testwork has been implemented or reported since the mine closure in 1989.

13.3 Historical Production Achievements

The historical production has been obtained from the 'A-Z Mining Professionals' 2017 Feasibility Study and by interpretation from the available mine and metallurgical month end reports. There is adequate production data for the period January to December 1988 with scant data available to mine closure in March 1989.

The available production data is detailed in Table 13-2. It is noted that the daily tonnage is calculated erroneously by using the tonnes per month per calendar day - the tonnes per hour is a better reflection of what the plant is capable of treating with an average of 43.3 tph for the 11 months, this is close to 980 t/d for an acceptable plant run time of 8 000 hours per annum. Two months did achieve an average of 50 tph or 1,150 t/d. The plant did not achieve the targeted 1,000 t/d due to unacceptable levels of equipment breakdowns and lack of feed stock.

Table 13-2: Puffy lake Historical Production

Month	Tonnage treated				Gold Produced		Silver Produced		Gold Recovery
	(tpm)	(t/d)	(tph)	g/t	(kg)	(oz)	(kg)	(oz)	(%)
Dec-87					14.0	450	5.9	191	72.9
Jan-88	27,306	881	N/A	1.98	42.6	1,369	12.0	386	78.7
Feb-88	22,379	799	42.6	2.20	37.8	1,216	11.9	383	83.8
Mar-88	27,920	901	45.4	2.14	45.0	1,447	11.8	381	81.7
Apr-88	29,370	979	49.7	2.66	49.8	1,601	10.4	334	81.2
May-88	31,773	1025	48.6	2.21	56.1	1,804	9.6	310	73.1
Jun-88	24,978	833	40.3	2.70	69.3	2,227	22.5	722	80.0
Jul-88	18,988	613	33.1	2.90	34.6	1,111	13.6	437	87.8
Aug-88	24,317	784	40.0	1.88	40.6	1,307	24.3	782	81.4
Sep-88	19,846	662	50.5	3.52	46.6	1,498	23.1	741	89.2
Oct-88	22,859	737	48.9	2.68	81.4	2,617	29.3	941	87.0
Nov-88	21,845	728	41.5	4.05	56.8	1,828	19.4	623	82.6
Dec-88	20,219	652	36.2	3.05	53.5	1,721	17.1	550	82.1
Jan-89	25,722	830	N/A	3.38	57.9	1,863	17.7	568	86.2
Feb-89	23,098	583	N/A	N/A	62.3	2,003	15.8	508	88.5
Mar-89	23,098	745	N/A	2.40	51.2	1,647	13.9	446	83.8
Apr-89		694	N/A	N/A	29.1	936	8.6	276	86.0
May-89					57.1	1,835	13.7	442	

Source: 2018 FS and PL mill month-end reports.

The mass pull to gravity and flotation concentrate is 4.4% with the historical gravity recovery to Jig concentrate of 47%, flotation recovery of 43% resulting in a total concentrate recovery of 90%. The total leach recovery is 91% resulting in an overall gold recovery to doré of 82%.

The tailings produced from the flotation plant averaged 0.27 g/t and combined with the leach plant tailings at 5.6 g/t resulted in a combined discard of 0.46 g/t. The tailings from both circuits are blended and discarded to the Ragged Lake storage facility.

13.3.1 Doré Quality

The quality of the doré produced at PL Mine is impacted by the quantity of silver recovered from the mineralized material. The average quality of the 51 ingots dispatched to the refinery is 69.6% Au and 22.1% Ag with the balance of 8.3% being base metals. The overall precious metal quality as reported is 91.7%. The

gold content in doré varied from 45% to 81%, subject to the quantity of base metals being recovered. One ingot was rejected by the refinery due to extremely high levels of contained base metals.

The doré quality is regarded as excellent and suitable as feed stock to any refinery.

13.3.2 Gold Inventory

Reviewing the production data, it would appear that the gold inventory and plant clean up accounted for the gold production in April and May 1989. This amounted to 86.2 kg of gold which is about 6 weeks historical production, which is excessive for a plant of the size of PL and the production rate.

Based upon the milling, flotation, leaching and smelting capacity within the plant and the 1,000 t/d treatment rate at 3 g/t, the plant gold inventory is expected to be 6-8 days and if shipping is done weekly, this may increase to 10-12 days production. At 1,000 t/d, the gold inventory within the plant is expected to be between 30 and 35 kg, assuming no significant spillage within the plant. This gold will vary based upon the head grade being treated and is not lost but is delayed production to be recovered when the mine eventually closes as was evident in 1989.

At 2,000 t/d, the inventory will increase due to additional equipment storage capacity to between 40 and 45 kg of gold.

A gold lock-up in the plant of about two weeks of production is expected at start-up, as the circuit inventory builds and before doré shipments become regular.

13.4 Projected Performance

It is proposed to implement a number of performance enhancing improvements to the gold plant flowsheet which are forecast to improve the historical gold recovery to about 90%, following the necessary refurbishment of the plant. These improvements include centrifugal gravity concentrator recovery systems with an associated leach reactor, enhanced oxygenation of the flotation concentrate leach with shear reactors and potentially improved fine grinding technology.

13.5 Future Testwork Recommendations

A metallurgical testwork program on representative 2025 to 2026 drill core is recommended for the pre-feasibility study (PFS) stage. It should comprise variability composites by domain and by open pit or underground source, comminution, gravity-recoverable gold, flotation and leaching of the concentrates, arsenic deportment and cyanide destruction, and ore sorting (as testwork only).

The historical testwork and the plant performance support the future expectations from the optimized performance treating the PL Mine mineralized material from both open pit and underground sources, once the operational difficulties have been eliminated, especially the stop-start operation due to lack of ore and breakdowns.

In the QP's opinion, the historical testwork and the historical plant performance support the 90% design recovery used in the PEA for both open pit and underground plant feed, provided that the operational problems of 1987 to 1989 are eliminated, in particular the stop-start operation caused by lack of ore and equipment breakdowns.

An investigative metallurgical report during late 1988 highlighted high levels of pyrrhotite in the ore and also indicated that this mineral was gold free. Pyrrhotite is magnetic and an oxygen consumer in the leach, and there is an opportunity for magnetic separation of this mineral which will reduce the load on the leaching circuit as well as improve oxygen levels through the leach. It is recommended that this testwork be initiated once production commences.

An ore sorting investigation is recommended as testwork only. Ore sorting could reduce the load on the mill, and so the size of the plant expansion, or increase the grade to the mill. Initial testwork can be done on drill core or on the stockpile at the decline, with final testwork on production from the mine once the plant is recommissioned. The PEA flowsheet does not include ore sorting, the 90% design recovery does not depend on it, and the US$0.5 million ore-sorting allowance in the pre-production costs is a testwork (study) cost only.

The arsenic content of the combined concentrate is expected to be high and as arsenic is partially soluble in a cyanide leach, the precipitation of dissolved arsenic is to be confirmed during the cyanide detoxification process. This may require a modification to plant operations for environmental reasons.

Once back in production at 1,000 t/d, it is recommended that typical concentrate samples be collected for test work associated with concentrate cleaning to produce a higher grade concentrate to assess the capacity of the leaching circuit. In addition, the concentrate is to be tested metallurgically to confirm that a shear reactor (such as Aachen or Mach1) will significantly improve the leach kinetics with the inclusion of oxygen.

It is recommended that the mine schedule incorporate the geometallurgical parameters of gold, sulphur and arsenic as a minimum, as the flotation plant performance is dependent upon the sulphur content far more than the gold content.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Over the period from June to August 2026, ABGM conducted a Mineral Resource Estimate (MRE) that serves to update the 2017 MRE conducted by CSA Global

The MRE for the PL Deposit has been prepared by Mr. Nick King, ABGM Principal Geologist and a Qualified Person (QP) for the reporting of Mineral Resources, as defined by NI 43-101. Mr. King is responsible for the geological domaining, block modelling, and MRE studies presented in this Report Section. Mr. King visited the PL Project site between April 24, 2026, and April 27, 2026, to review diamond drill core, core sampling methodologies and borehole collar verification.

Puffy Lake's Mineral Resources have been assessed through the application of a range of technical and economic factors to ensure reasonable prospects for eventual economic extraction. In the opinion of the QP, the resource evaluation reported herein is a reasonable representation of the gold mineral resources at the deposit based on the available information. The updated MRE has an effective date of September 10, 2026, and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and the CIM Definition Standards for Mineral Resources & Mineral Reserves (2014). The MRE was prepared in conformity with the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).

Reported Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of a Mineral Resource will be converted into a Mineral Reserve.

14.2 Database

The 2026 Mineral Resource work is based on the cleaned project drilling database and updated three-dimensional mineralized framework. The cleaned property database contains 1,005 collars and approximately 146.7 km of drilling across PL and Nokomis, of which 858 collars are PL-related. The modelling dataset integrates historical PUF-series drilling, Auriga drilling, Minnova 2016-2017 drilling and the subsequent 2018-2026 programs. The updated drilling materially increases data density in the shallow portion of the PL deposit targeted for open-pit evaluation.

All drill hole data was imported into Datamine™ software and interrogated via Leapfrog™ validation functions prior to constructing a drill hole database for the deposit. The resulting database contains all available drilling and sampling data for the Project. Key fields within these critical drill hole database data files are validated for potential numeric and alpha-numeric errors. Data validation cross referencing collar, survey, assay, and geology files were performed to confirm drill hole depths, inconsistent or missing sample/logging intervals, and survey data. The data was validated and checked for logical or transcription errors, such as overlapping intervals. There were a few, very minor errors that were corrected.

Model inputs were desurveyed in Datamine and linked to the interpreted mineralized wireframes. Additional geometric normalization of sample support relative to local wireframe orientation before compositing was applied. This step was introduced because core bedding/orientation angles were not consistently available and vertically or obliquely drilled holes intersect the shallow-dipping structures at different apparent widths.

The QP has reviewed sample collection methodologies adopted by Minnova and previous operators and is satisfied that data collection methodologies are of a standard that allow the estimation of resources under CIM Guidelines and that mineral resource databases for the PL Deposit fairly represent the primary information.

14.3 Geological Model

The geological model represents the PL deposit as a set of discrete mineralized shear and vein lenses within the broader structural framework. Geological modelling was undertaken by ABGM using Datamine™ and Leapfrog™ software

Principal domains include the historical Sherridon, Upper, Main and Lower systems together with Central, southern, Lower-Lower and PL North splays resolved by the updated drilling. Domain construction follows interpreted structural continuity rather than a single grade shell. Hard boundaries were used in the 2026 estimation workflow so that informing grades did not cross the defined statistical/domain boundaries.

Where the available wireframes did not fully bound the top and bottom of all mineralized intervals, the modelling workflow used indicator estimation to assist in defining the likely extent of mineralized blocks. The indicator calibration relates estimated indicator values to the observed presence or absence of mineralization and is used as a geometric support tool rather than as a direct gold-grade estimate.

14.4 Estimation Methodology

14.4.1 Data Preparation

14.4.1.1 Desurveying and Geometric Normalization

Drillholes were desurveyed using the Datamine desurvey process. For each wireframe triangle, local strike and dip were calculated from the triangle vertices. Drillhole orientation at the wireframe intersection was then compared with the local wireframe orientation to derive a corrected length normal to the mineralized structure. Sample coordinates were recalculated using the corrected support. This procedure reduces support bias where holes intersect the shallow-dipping mineralized structures at markedly different angles.

14.4.1.2 Compositing

A 1 m composite length was adopted following review of alternative support lengths and is consistent with previously applied estimation methodologies. The intent was to provide consistent estimation support while limiting residual-length effects. Composites were generated within the interpreted mineralized framework after geometric normalization. Domain boundaries were honoured during compositing and estimation.

14.4.1.3 Statistical Zoning and Local Means

Three geostatistical zones have been identified and assessed, taking into consideration, stationarity, local means, outliers, variography, and estimation parameters. Moving-window variance comparisons were used to test whether samples within each zone could be considered sufficiently stationary for geostatistical estimation. Once defined, the zones should show distinct differences in their distribution characteristics.

Table 14-1: Zonal Statistics of the Puffy Lake shears

Zonal Statistics				
Zone	Minimum	Maximum	Mean	Variance
1	0.03	8290	229.14	516567.75
2	0.03	9850	49.82	142725.60
3	0.03	8000	22.95	66591.03

A local mean is an area large enough and with enough samples that the area mean remains stationary. Local-mean search radii were selected by examining the stabilization of within-search variance with increasing search distance. Where a local mean could be calculated, it was assigned to the block model; otherwise, the relevant zonal mean was used.

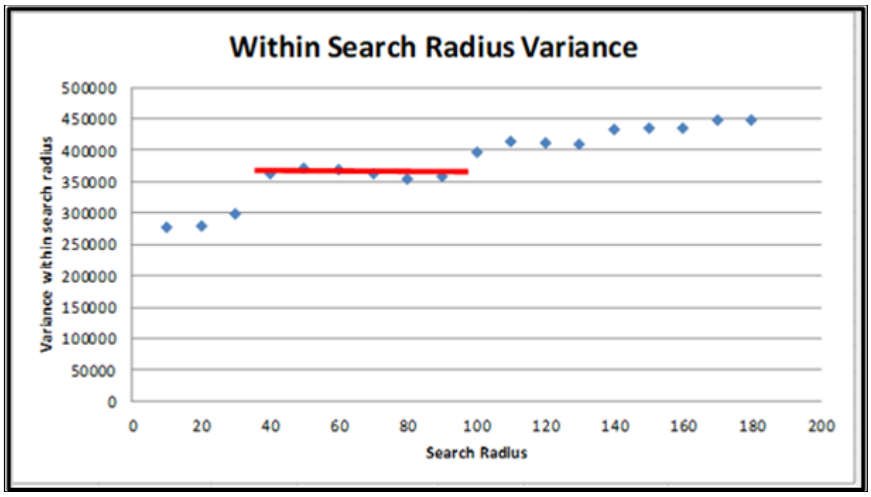


Figure 14-1: Local Mean Analysis for Zone 1.

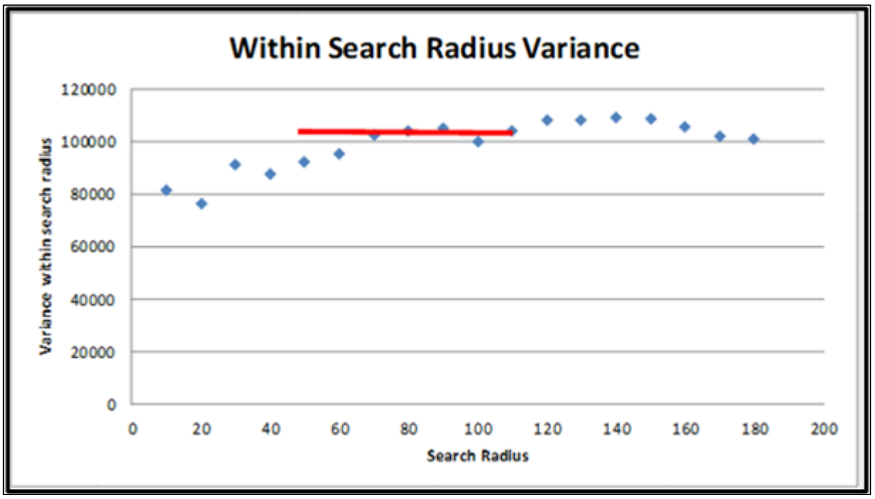


Figure 14-2: Local Mean Analysis for Zone2.

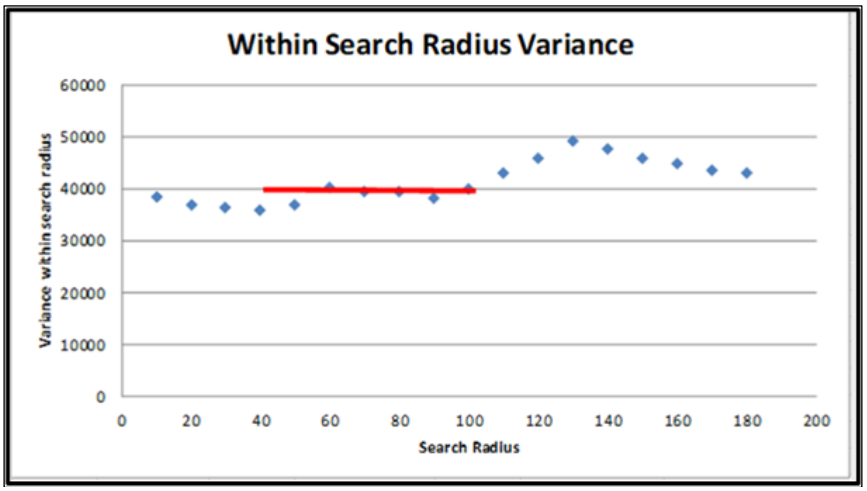


Figure 14-3: Local Mean Analysis for Zone3.

Table 14-2: Local mean criteria

Zone	Local-mean search 1	Min. samples	Local-mean search 2	Min. samples
1	40 m	38	60 m	51
2	70 m	80	90 m	97
3	60 m	80	80 m	103

14.4.1.4 Outlier Treatment

Outlier analysis was accomplished in the following manner:

- Initial potential outliers were identified by calculating the 99% upper confidence limits per zone and utilizing these to obtain the coordinates of the samples that fell above this threshold.
- Utilizing the local mean search analysis to determine the search radius and number of samples required for a localized mean. This, in turn, is used along with the coordinates of the potential outlier as the centre point to obtain samples within the search parameters.
- Once the samples in point 2 have been obtained the 99% confidence on the localized mean is calculated. If the sample in question exceeds this value the sample value in question is reduced to the 99% confidence level, otherwise the value is not an outlier locally and hence remains the same.
- This is done for each sample above the threshold.

Table 14-3: Outlier analysis results

Zone	Outlier Value	Initial Outliers	Local Outliers
1	1901	122	0
2	929	147	0
3	623	53	1

14.4.1.5 Variography

Nearest-neighbour analysis indicated a dominant horizontal spacing of approximately 17 m. Experimental semi-variograms were calculated using a 5 m lag, 2.5 m lag tolerance and up to 100 lags, giving a maximum experimental distance of approximately 500 m. Zonal-relative semi-variograms were used to stabilize variance relative to the zone mean. Directional models were fitted first in the horizontal plane and then at varying dips to characterize anisotropy. Effective model ranges were used to define estimation search distances so that the semi-variogram was not extrapolated beyond the range where the experimental data were considered informative.

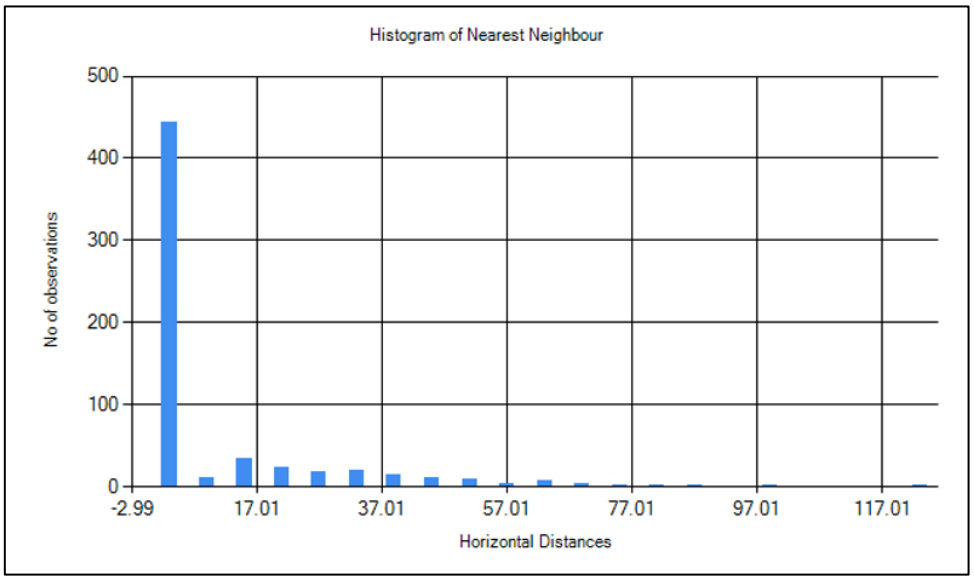


Figure 14-4: Nearest Neighbour Analysis of Horizontal Spacing.

A distance lag of 5 m with a lag tolerance of 2.5 m together with a directional lag of 30 degrees and a directional tolerance of 15 m was adopted. The calculation was done utilizing a zonal relative calculation, that is, lag variance divided by the square of the mean of the zone. This provides a more orderly semi-variogram as well as an easy back transform of the kriging variances.

Each semi-variogram was initially modelled on the horizontal utilizing the pronounced anisotropies and then modelled at different dips to determine the overall anisotropy. Since Datamine software does not support zonal anisotropy in its modelling or Kriging, the least-squares fit of the different directions was taken to ensure the zonal anisotropy effect was minimized whilst at the same time providing a single set of nugget and sills. Where necessary, the semi-variograms were modelled with extended ranges so that the models would fit the lags for as long as possible. The effective ranges (where the model best fits), were noted and utilized in the search parameters, thus limiting the utilization of the semi-variogram beyond the lengths that fit the data. The following figures are the horizontal fits to the experimental semi-variograms utilizing the zonal anisotropies.

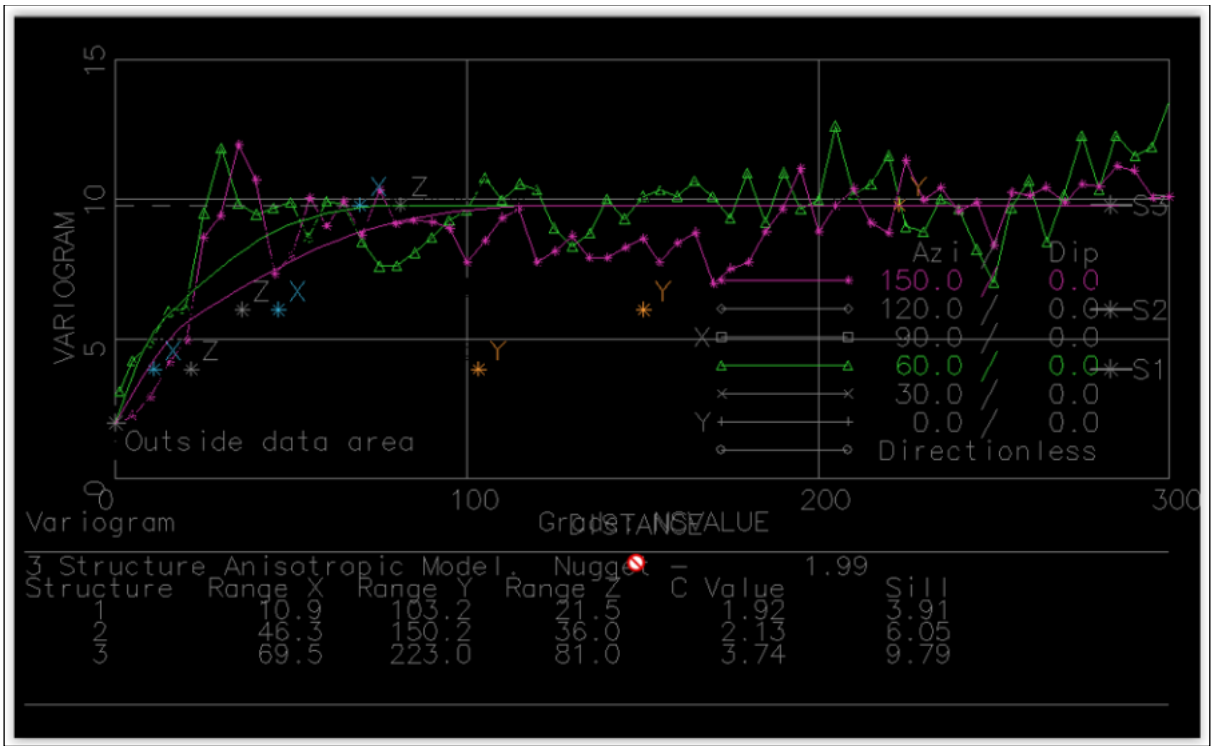


Figure 14-5: Zone 1 Semi- variogram model

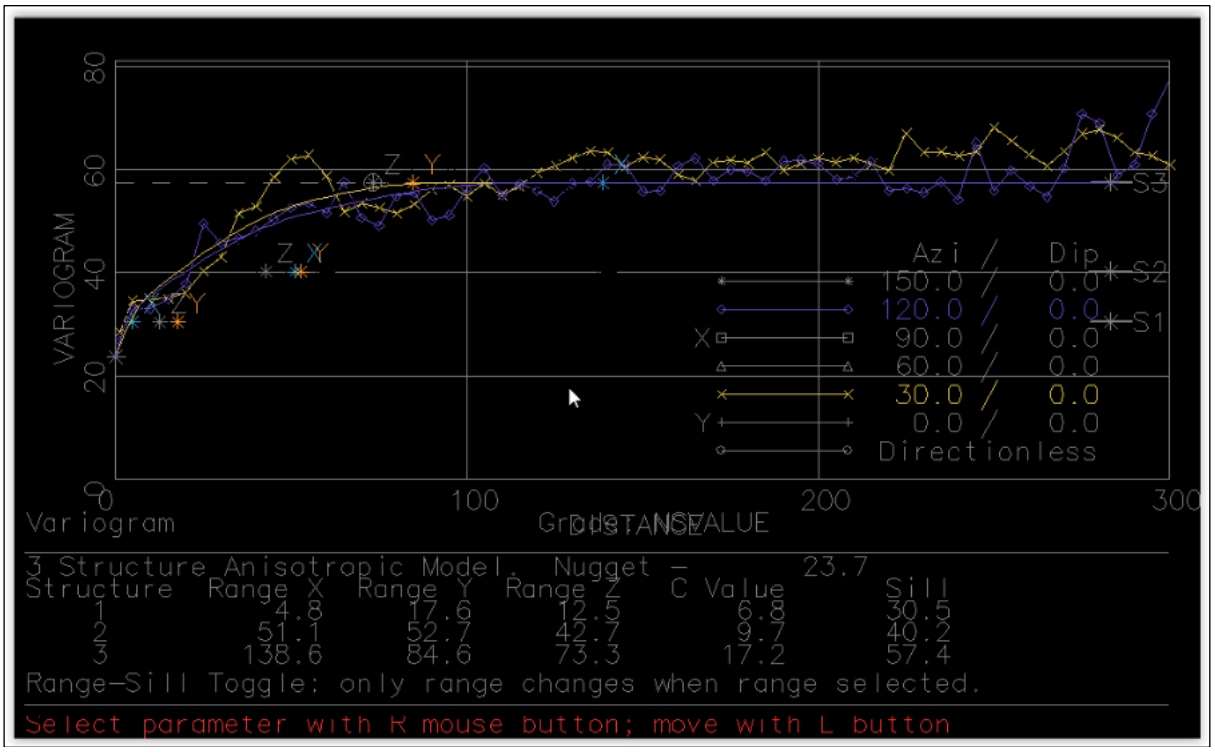


Figure 14-6: Zone2 Semi- variogram model

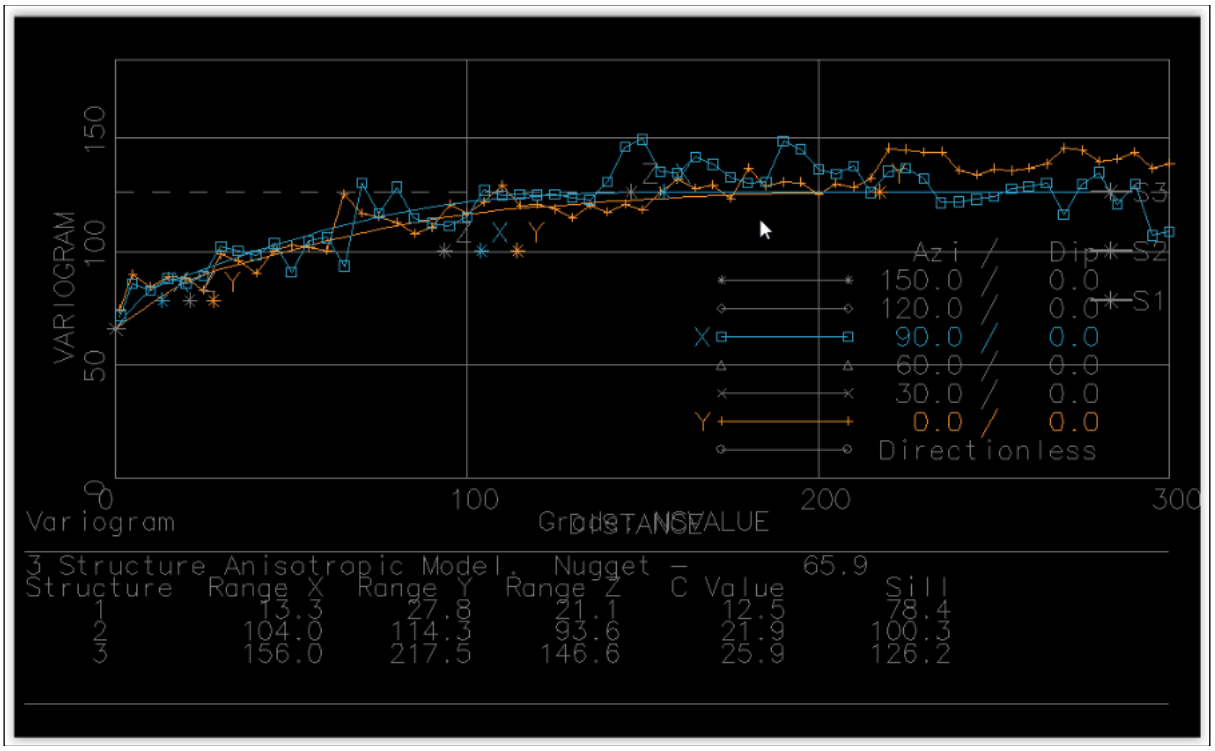


Figure 14-7: Zone 3 Semi- variogram model

The log-point semi-variograms were also modelled for potential change of support calculations; however, fortunately these were not required.

The variographies are a practical reflection of the spatial continuity of the mineralization grades, and their application to the geostatistical analysis is adequate to minimize uncertainty and to derive appropriate resource block models for use by the planning engineers.

14.4.1.6 Calculation of kriging parameters

The Kriging parameters were calculated by doing multiple cross-validation exercises using constrained numbers of samples of min 5 max 5, min 10 max 10, min 15 max 15, min 20 max 20, min 25 max 25, min 30 max 30, min 40 max 40, min 50 max 50, and min 60 max 60. These runs were then combined into a single file where the absolute errors were calculated for each estimate. This can be accomplished as both the estimate and actual are available. The errors are then calculated against the number of samples accessed and the minimized function obtained. As the function is of a quadratic nature (high errors with low number of samples and errors decreasing with an increasing number of samples, and finally errors increasing again when too many samples are accessed), the function can be solved for the minima by obtaining the first derivative of the function. This is where the minimum error occurs. The minimum error is then taken and 5% added and the intersection of the function is obtained. This then provides the minimum and maximum samples for Kriging.

The absolute minimum number of samples to be used is 5. This is because of degrees of freedom lost due to:

- Calculation of mean.
- Calculation of variance.
- Constrained lag distance.
- Constrained directions.

Four degrees of freedom are thus lost, as one requires a minimum of one degree of freedom in the estimate, this translates into a minimum of 5 samples. Following this methodology, the minimum and maximum number of samples calculated for each zone are tabulated below.

Table 14-4: Table indicating the minimum and maximum number of samples to be used for kriging

Zone	Minimum. Samples	Maximum Samples
1	10	20
2	9	35
3	15	25

14.4.1.7 Kriging efficiency vs. Errors incurred on estimates

The kriging efficiency is mapped against the error incurred and a function fitted. This is utilized after kriging to determine the error incurred with a certain kriging efficiency, which can then be used to check the confidence of the estimate against the lower confidence determined from the calculation from the kriging variances and values.

14.4.1.8 Discretization for kriging

A level of discretization that will ensure that the estimates follow the dispersion variance of the block size within the zone is required. This is obtained from the point semi-variogram and calculating the variance between points in a block with increasing numbers of regularly placed samples. When the variance levels off, we have the required discretization for block kriging. The following are the charts showing the models of this relationship for each zone.

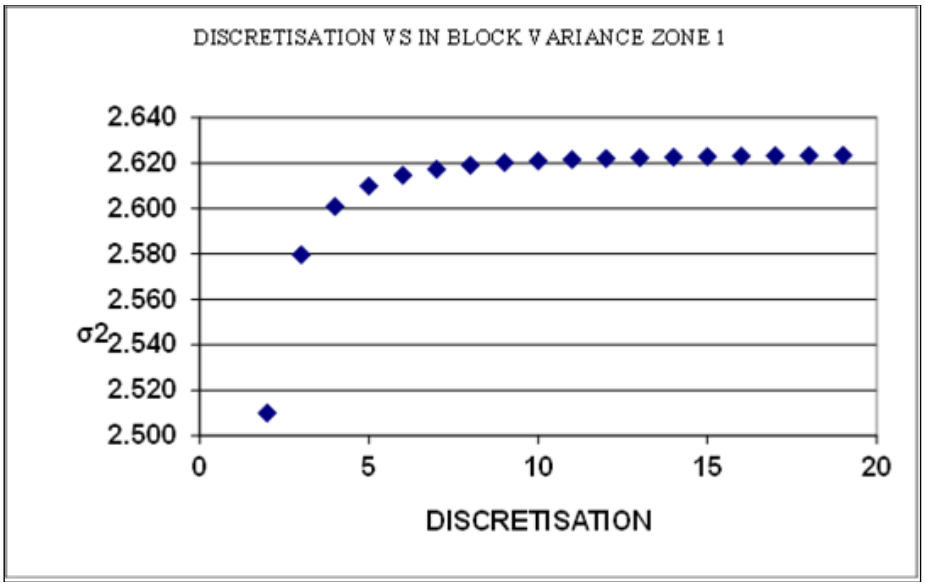


Figure 14-8: The relationship between discretization and block variance for Zone 1

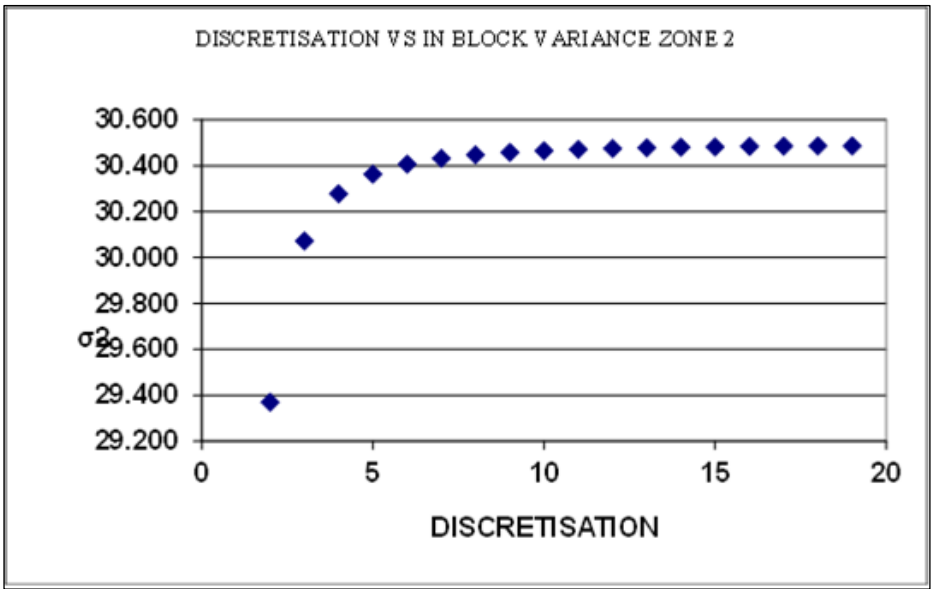


Figure 14-9: The relationship between discretization and variance for Zone 2

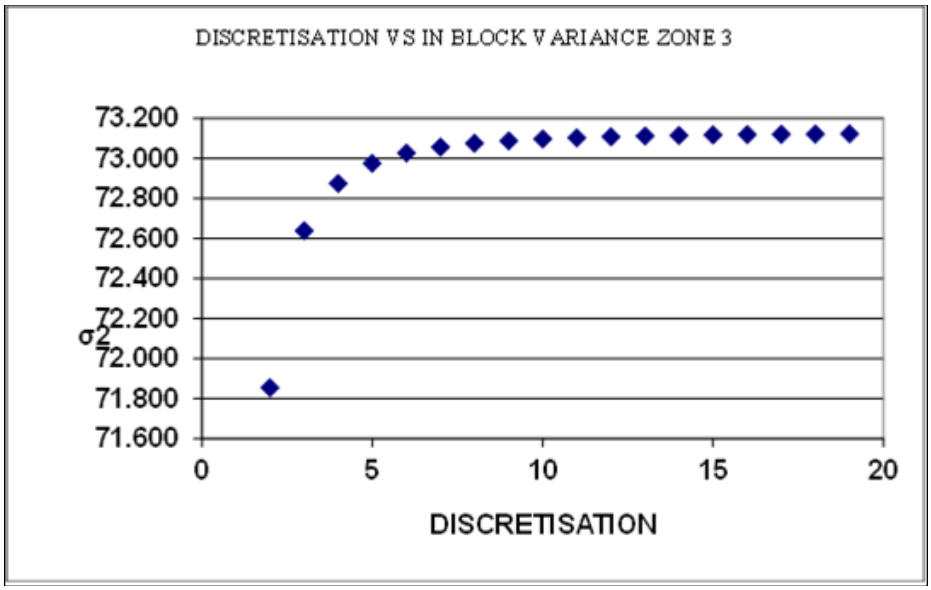


Figure 14-10: The relationship between discretization and variance for Zone 3

14.4.2 Kriging and Post Processes

Kriging was carried out on 10 m × 10 m × 1 m blocks.

After kriging, the kriging efficiencies were calculated ((Block Variance - Kriging Variance)/ Block Variance). From these kriging efficiencies, the error is calculated from the functions modelled as indicated in the section "Kriging efficiency versus errors incurred on estimates", as well as utilizing the formula for a lower confidence (in this case 90%). This, in turn, was utilized to calculate the percentage error via the formula (estimate – lower confidence)/estimate. Which finally is translated into a confidence level via the following table, Table 14-5.

Table 14-5: Confidence levels and percentage accuracy

Confidence	Percent
Measured	85 to 100
Indicated	70 to 85
Inferred	Less than 70

The mineralization extends beyond the shear zones themselves into both the hangingwall and footwall of each zone. To correctly categorize individual blocks as mineralized material or waste, it was decided to do an indicator kriging exercise. Here all portions of the boreholes with value were assigned a 1, and those with no value were assigned a zero. These were then utilized to calculate and model an indicator semi-variogram. The indicators were then kriged and mapped against the actuals on the drill holes to determine the ranges and confidences of the indicator estimates. The following table indicates the results.

Table 14-6: Indicator confidences

Indicator Value		Chance	0 = Waste
From	To		1 = Mineralized
0	0.216	100%	0
0.216	0.487	70%	0
0.487	0.705	80%	1
0.705	1	100%	1

Table 14-6 can be interpreted as follows:

- If the indicator falls between 0 and 0.215 then the block is definitely not mineralized.
- If the indicator falls between 0.215 and 0.487 then approximately 70% of the time the block is not mineralized or alternatively 30% of the block could be mineralized.
- If the indicator falls between 0.487 and 0.705 then 80% of the time the block is mineralized or alternatively 20% of the block is not mineralized.
- Finally, if the indicator is above 0.705 the block is definitely mineralized.

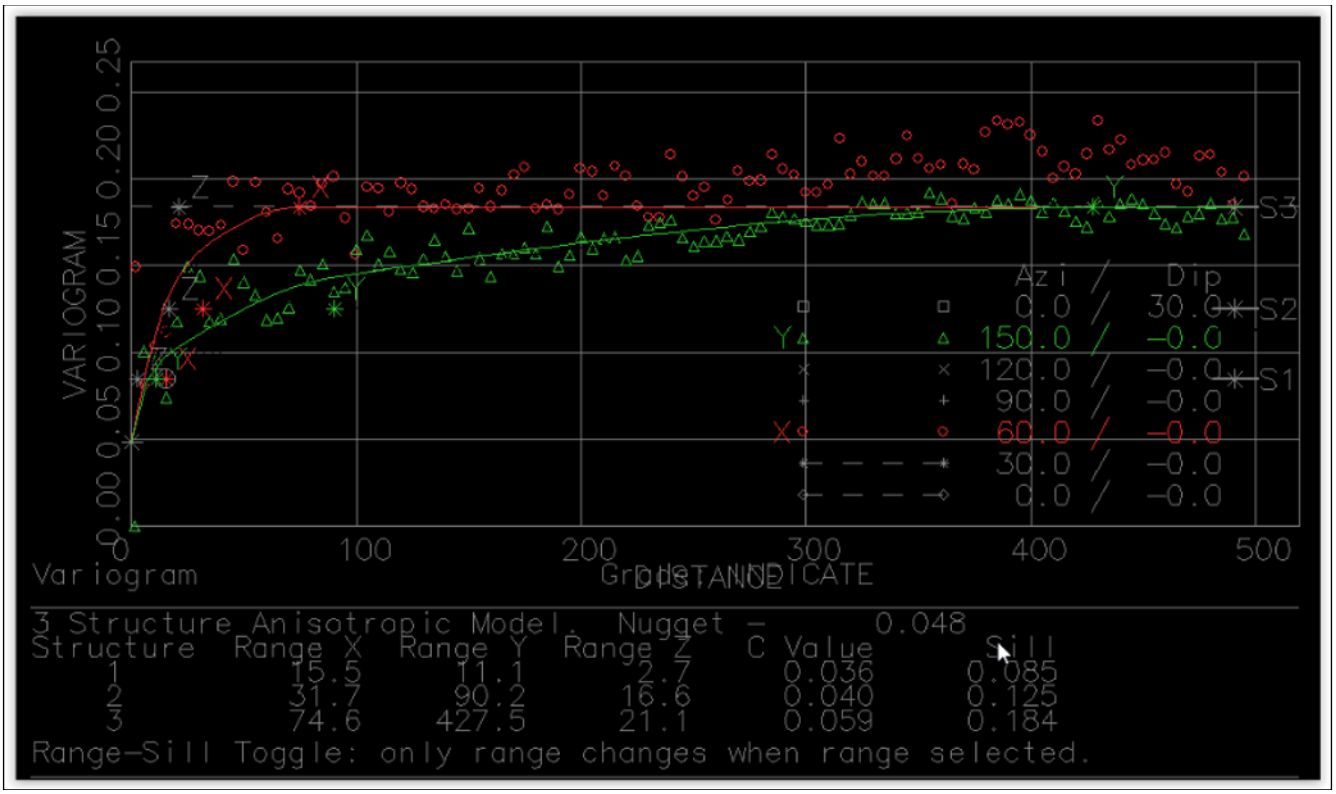


Figure 14-11: Indicator semi-variogram

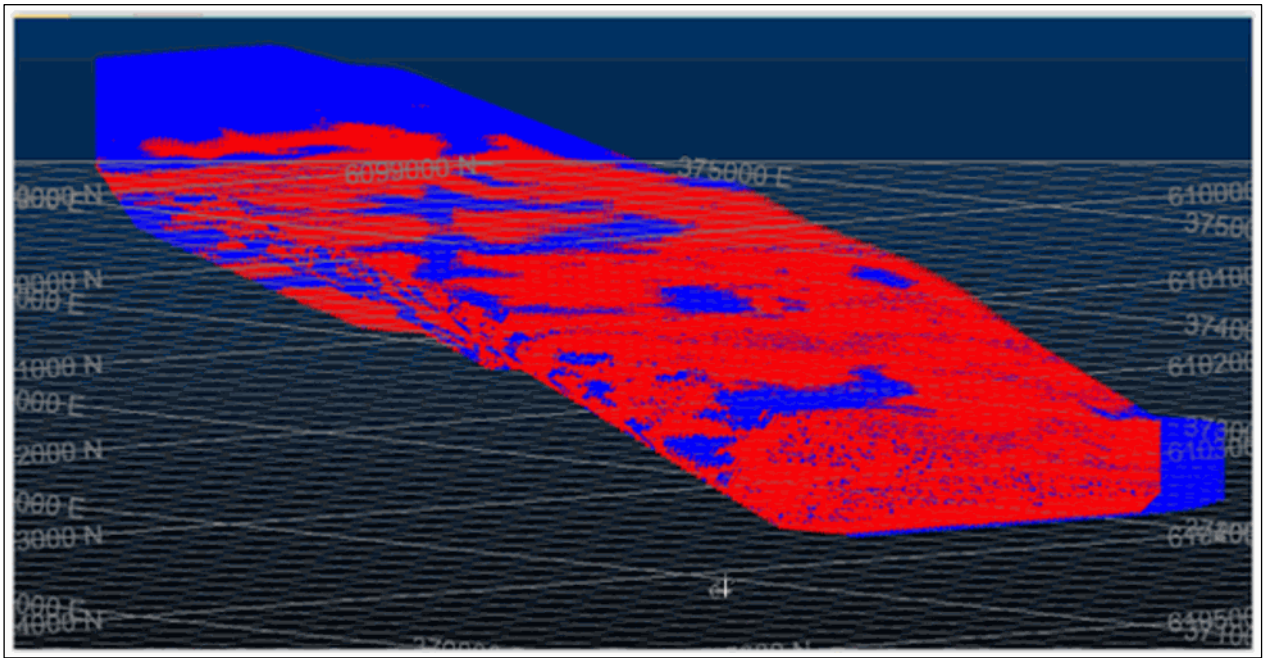


Figure 14-12: Oblique view of Mineralized (red) and non-mineralized (blue) blocks

14.4.3 Local and Zonal Means

Initially the zonal mean for each zone was populated into the model. Then, the local means were calculated using the local mean search and number of samples. Nevertheless, not all blocks were populated with local

means, thus where a local mean exists, this was utilized, alternatively where the local mean is absent, the zonal mean was accepted.

14.5 Block Modelling

The Block model extends from above the uppermost to below the bottom shear zones. This allows the mine planning engineers to utilize the wireframes to constrain the planning without ending up with blank values due to insufficient sub-cell division of the blocks.

14.5.1 Checking the extent of smoothing

A final check to determine the extent of overall smoothing effect is by calculating the block dispersion variance from the semi variograms and comparing this to the dispersion variance of the blocks within the block model. The between block variance was calculated using a 8 by 8 by 2 discretization and the formula (all points variance – points in block variance = between block variance). The table below shows the results of the comparison.

Table 14-7: Semi-variogram vs. block dispersion variance

Variable	Dispersion Variance
Required Dispersion Variance	6677
Estimates Dispersion Variance	6674

14.6 Classification

Puffy Lake's in situ Mineral Resources have been subject to an assessment through the application of a range of assumed technical and economic factors and are classified as either Measured, Indicated or Inferred in accordance with the definitions in the National Instrument 43-101 Standards of Disclosure for Mineral Projects.

An Inferred Mineral Resource, as defined by the CIM Standing Committee, is "that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration."

An Indicated Mineral Resource has a higher level of confidence than that applying to an Inferred Mineral Resource. It may be converted to a Probable Mineral Reserve. An Indicated Mineral Resource, as defined by the CIM Standing Committee, is "that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve."

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve. A Measured Mineral Resource, as defined by the CIM Standing Committee, is "that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

Mineral Resources that are not Mineral Reserves do not account for mineability, selectivity, mining loss, and dilution and do not have demonstrated economic viability. These Mineral Resource estimates include Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is also no certainty that these Inferred and Indicated Mineral Resources will be converted into the Indicated and Measured categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

14.7 Resource Statement

The effective date for this report is September 10, 2026.

Gold mineralization at the PL deposit is controlled by four main shear zones named the Sherridon, Upper, Main, and Lower zones. Within each of these main shear zones, secondary, mineralized shears have been identified. A total of 14 shear zones have now been identified. These are:

1. Sherridon
2. Middle
3. North
4. Lower - North
5. Upper North
6. Main North
7. Lower Centre
8. Main Centre
9. Lower - South
10. Upper South
11. Lower Lower North 2

12. Lower Lower North
13. Lower Lower South
14. Lower Lower Centre

These shear zones, and their contained mineralization, subcrop under a northwest-southeast-trending swamp and have been outlined by surface diamond drilling to a vertical depth of up to 500 m, approximately 1,200 m down-dip. The Upper, Main, and Lower zones are hosted primarily by mafic amphibolites that are considered to be part of the Amisk Group, and by metasedimentary gneisses of the Missi Group within 50 m of the sheared tonalite contact. Toward the north end of the deposit, the Lower zone is commonly hosted within the tonalite body. The PL deposit's gold-bearing shear zones generally strike N30°W and dip moderately at 30 degrees to the northeast, subparallel to the regional foliation. The mineralized zones occur in 1.3 m to 2.5 m wide shear zones that contain variable amounts of quartz veins. The quartz veins form as 1.5 cm to >1.0 m wide fault-fill veins and tension vein arrays. The average thickness of the quartz veins is 25 cm to 35 cm. Gold mineralization is structurally controlled and post-dates both regional folding and intrusive events.

The last NI 43-101 compliant Resource Estimate was completed by Mr. Leon McGarry of CSA Global in October 2017 and is tabulated below:

Table 14-8: CSA Global 2017 PL Project Resources

Category	Au Cut-Off (g/t)	Tonnes (kt)	Au Grade (g/t)	Contained Au (oz.)
Measured	2.5	425	7.53	102,900
Indicated	2.5	1,056	5.29	179,600
M+I	2.5	1,481	5.93	282,500
Inferred	2.5	1,846	5.08	301,700

Notes CSA Global 2017:

- The volume of the historical mined areas was depleted from the resource estimate.
- Capping values range from 30 to 45 g/t Au and affected 16 samples.
- Bulk densities of 2.81 t/m³ were used for tonnage calculations at the PL Deposit.
- A gold price of US$1,250/oz and an exchange rate of US$0.80=C$1.00 was utilized in the Au cut-off grade calculations of 2.5 g/t for underground mining. Operating costs of C$125/t. Process recovery used was 95%.
- Tonnes and ounces have been rounded to reflect the relative accuracy of the mineral resource estimate; therefore, numbers may not sum precisely.
- 1 troy ounce equals 31.10348 grams.
- Mineral Resource tonnes quoted are not diluted.
- The NI 43-101 mineral resources in this Report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
- Mineral resources are not mineral reserves and by definition do not demonstrate economic viability. This mineral resource estimate includes inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to the measured and indicated resource categories through further drilling, or into mineral reserves, once economic considerations are applied.

ABGM has audited the Minnova drilling database and, where geological and grade continuity could be reasonably interpreted, modified the shear zone boundaries based on the new drill hole information.

Along with the development of open-pit mining, ABGM has contemplated the re-opening of the former producing Puffy Lake gold mine utilizing available infrastructure and new underground development. ABGM has estimated a NI 43-101 compliant resource based on the following criteria.

- The depletion of the historically mined areas from the Resource Estimate,
- A gold price of 3,000 US$/oz,
- An exchange rate of US$0.74 = C$1.00
- 1 Troy Ounce = 31.10348 grams,
- Cut-off grades of 0.6 g/t for open-pit mining, 1.92 g/t for underground mining that includes capital development, and 1.43 g/t for marginal underground mining
- Geological losses at 3 %
- A bulk density of 2.81 t/m^3, and
- A process recovery rate of 90%.

Table 14-9: ABGM 2026 PL Project Resources

ABGM 2026 Results			
OPEN PIT (@ 0.6 g/t Cut-Off Grade)			
Class	**Tonnes**	**g/t**	**Au oz**
MEASURED	1,288,360	5.35	221,575
INDICATED	289,750	4.99	46,516
INFERRED	118,689	5.75	21,955
UNDERGROUND (@ 1.92 & 1.43 g/t Cut-Off Grades)			
Class	**Tonnes**	**g/t**	**Au oz**
MEASURED	1,343,678	5.06	218,603
INDICATED	835,239	4.26	114,464
INFERRED	1,548,443	3.63	180,855
Open Pit + Underground (@ 0.6; 1.92 & 1.43 Cut-off Grades)			
Class	**Tonnes**	**g/t**	**Au oz**
MEASURED	2,632,038	5.20	440,178
INDICATED	1,124,989	4.45	160,980
INFERRED	1,667,132	3.78	202,810

***Notes**:*

1. All tabulated data have been rounded and as a result minor computational errors may occur.
2. Mineral Resources, which are not Mineral Reserves, have no demonstrated economic viability.

14.8 Reasonable Prospects for Eventual Economic Extraction

The CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014) require that a Mineral Resource has reasonable prospects for eventual economic extraction (RPEEE). The CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 29, 2019) (CIM 2019 MRMR Best Practice Guidelines) require the assumptions used to make this judgement to be stated, and the constraining surfaces and shapes to be documented. This subsection sets out how RPEEE was assessed for the 2026 MRE of the PL Gold Mine, and the parameters used.

Two conceptual extraction scenarios were assessed. Near-surface mineralization was constrained by an optimized open-pit shell. Mineralization below the shell was constrained by underground stope shapes generated with the Mineable Shape Optimiser (MSO). The shell and the shapes are reporting constraints only. A 25 m crown pillar directly underneath the open pit economical shell was excluded from the resource.

Table 14-10: Summary of RPEEE constraints and reporting cut-off grades

Scenario	Constraint	Reporting cut-off (g/t Au)	Basis
Open pit	Optimized pit shell (three pit areas)	0.60	Full-cost cut-off of 0.59 g/t Au, rounded up
Underground – economic	MSO shapes below the 25 m crown pillar	1.92	Break-even grade including development cost
Underground – marginal	MSO shapes at the margin of economic shapes	1.43	Break-even grade excluding development cost

Gold price US$3,000/oz. Metallurgical recovery 90% in all cases. All costs are in US$.

14.8.1 Approach and Responsibilities

The open-pit optimization and the underground MSO runs were overseen by Ettienne Jaques Oosthuizen of ABGM, a mining engineer and the Qualified Person (QP) responsible for this subsection. Nicholas Ross King of ABGM is the QP responsible for the geological model, the block model, classification and the 2026 MRE as a whole. Mr. King reviewed the pit shell and the MSO shapes and accepted them as the reporting limits for the Mineral Resources.

Both constraints were applied to the 2026 MRE block model, depleted for the historical underground workings. The software used was:

- Open pit: The optimization and pit-area outputs were handled in the Datamine's latest at the time of reporting NPV Scheduler (NPVS) environment.
- Underground: stope shapes were generated with latest versions at the time of reporting of Datamine's MSO (Mineable Shape Optimiser).

Measured, Indicated and Inferred blocks were all allowed to contribute to the pit shell and to the MSO shapes. The resource classification as received from the Resource QP and built within the resource model was assigned to material within each shape block by block, so that each class is reported separately.

The pit shell and the MSO shapes are used only to demonstrate RPEEE and to constrain the reported Mineral Resources. They are not mine designs and they are not Mineral Reserves. They do not demonstrate economic viability, and no mine plan, production schedule or economic analysis forms part of this Technical Report. The costs and recoveries listed below are conceptual inputs for defining reporting limits. They are not a forecast and do not reflect a production decision. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.8.2 Open-Pit Constraint

14.8.2.1 Optimization parameters

Table 14-11 lists the economic and technical parameters used to optimize the pit. The costs are expressed per tonne of material. The processing, haulage-to-mill, G&A and reclamation costs are applied per tonne of mineralized material.

Table 14-11: Open-pit optimization parameters

Parameter	Value	Unit / note
Gold price	3,000	US$/oz (US$96.45/g)
Mining cost – mineralized material	6.70	US$/t mined
Mining cost – waste	5.48	US$/t mined
Processing	26.81	US$/t mineralized material
Haulage to mill	2.44	US$/t mineralized material
General and administration (G&A)	12.60	US$/t mineralized material
Reclamation	0.61	US$/t mineralized material
Total cost of mineralized material (full cost)	**49.16**	**US$/t (sum of the five rows above, including the US$6.70/t mining cost)**
Selling cost	31.68	US$/oz produced (US$1.02/g)
Royalty	3	% of gross revenue
Metallurgical (milling) recovery	90	%
Overall pit slope angle	50	degrees, all walls
Mining loss (mining recovery)	10 (90)	% – used in defining the limits only
Mining dilution	20	% – used in defining the limits only
Pit shell selected	100% Revenue Factor	US$3,000/oz long-term price (below spot at the time of assessment), with no further revenue-factor adjustment. Used for RPEEE only, not a pit design.

14.8.2.2 Cut-off grade derivation

Net revenue per gram of recovered gold allows for the 3% royalty on gross revenue and the selling cost:

Gold price per gram = 3,000 ÷ 31.1035 = US$96.45/g

Net revenue = 96.45 × (1 − 0.03) − 1.02 = US$92.54/g

The full-cost cut-off grade uses the total cost of mineralized material of US$49.16/t:

Full-cost cut-off = 49.16 ÷ (0.90 × 92.54) = 0.590 g/t Au

The incremental cut-off grade deducts the waste-mining cost (US$5.48/t), which a block inside the pit incurs whether or not it is processed. This gives an incremental cost of US$43.68/t:

Incremental cut-off = 43.68 ÷ (0.90 × 92.54) = 0.524 g/t Au

Open-pit Mineral Resources are reported at 0.60 g/t Au. This is the full-cost cut-off rounded up, and it is above the 0.52 g/t Au incremental cut-off. Material between 0.52 g/t Au and 0.60 g/t Au inside the shell is not reported. In the QP's opinion, this choice is conservative.

14.8.2.3 Shell selection and pit description

The open-pit Mineral Resource Estimate was constrained using the 100% revenue factor pit shell generated at a long-term gold price assumption of US$3,000/oz.

The selected gold price represents a long-term planning assumption that is below the prevailing spot-price level at the time of the assessment. This approach is intended to reduce reliance on elevated spot prices when defining the conceptual open-pit mining envelope for Mineral Resource reporting.

The 100% revenue factor shell was selected because it represents the pit optimization outcome at the adopted long-term gold price, without applying an additional revenue-factor adjustment. The selection is considered appropriate for demonstrating RPEEE, subject to the Qualified Person's assessment of the reasonableness of the underlying technical and economic assumptions and the resulting mining geometry.

Pit shell sensitivity analyses were undertaken at higher gold price assumptions to assess the potential effect of improved revenue conditions on the conceptual open-pit envelope. The base-case reporting constraint remains the 100% revenue factor shell at US$3,000/oz, providing consistency between the adopted long-term price assumption and the Mineral Resource reporting constraint.

The selected shell is used solely to constrain the Mineral Resource on the basis of reasonable prospects for eventual economic extraction. It does not represent a final pit design, establish economic viability, or constitute a Mineral Reserve. The reported Mineral Resources remain subject to the uncertainties inherent in the geological model and the technical and economic assumptions adopted.

The Qualified Person considers the selected constraint in conjunction with the deposit's geological continuity, mineralization distribution, assumed mining method, metallurgical recovery, operating costs, pit slope assumptions and other relevant factors, in accordance with NI 43-101 and the applicable CIM Definition Standards and Best Practice Guidelines.

The selected shell comprises three pit areas along the mineralized system (Figure 14-14 to Figure 14-16). The North-West area is a large excavation. The North-East and South-East areas are smaller excavations that extend southeast and east from it. The overview in Figure 14-13 carries local depth annotations of 35 m to 155 m (including 45 m, 55 m, 65 m and 100 m). Historical underground workings lie within and beneath the South-East area.

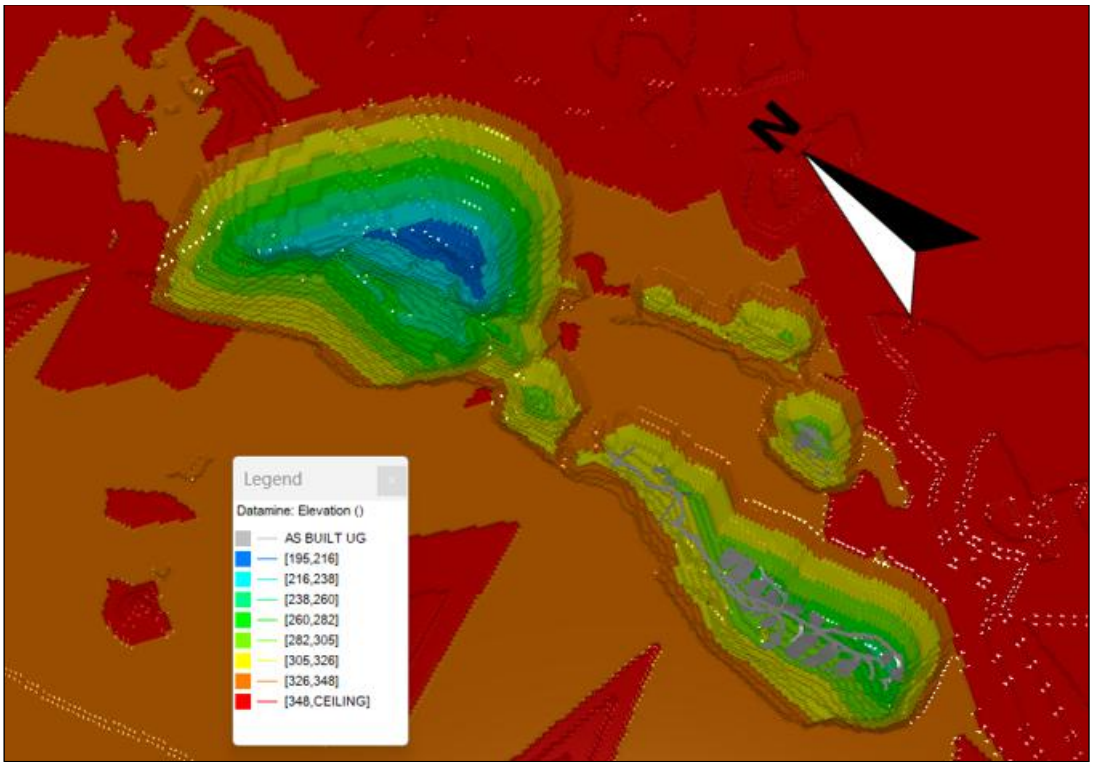


Figure 14-13: Oblique view of the RPEEE pit shell showing the three pit areas, coloured by elevation, with historical underground workings in grey.

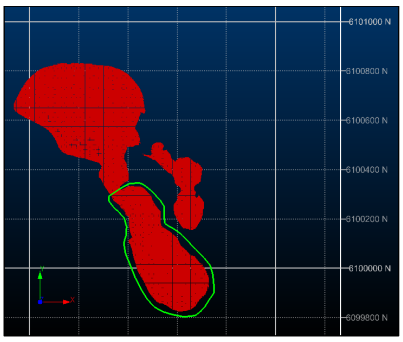


Figure 14-14: South-East pit area (outlined in green).

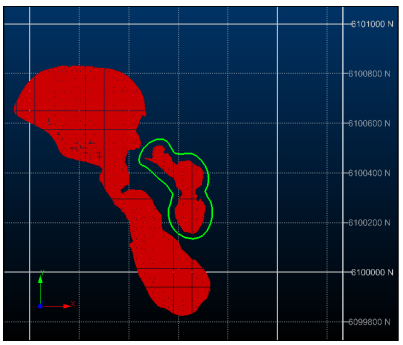


Figure 14-15: North-East pit area (outlined in green).

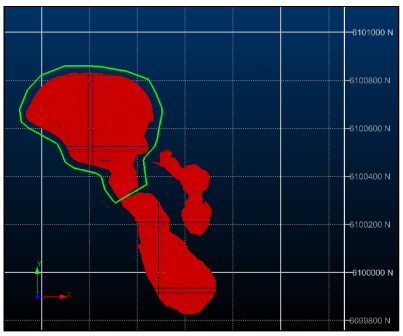


Figure 14-16: North-West pit area (outlined in green).

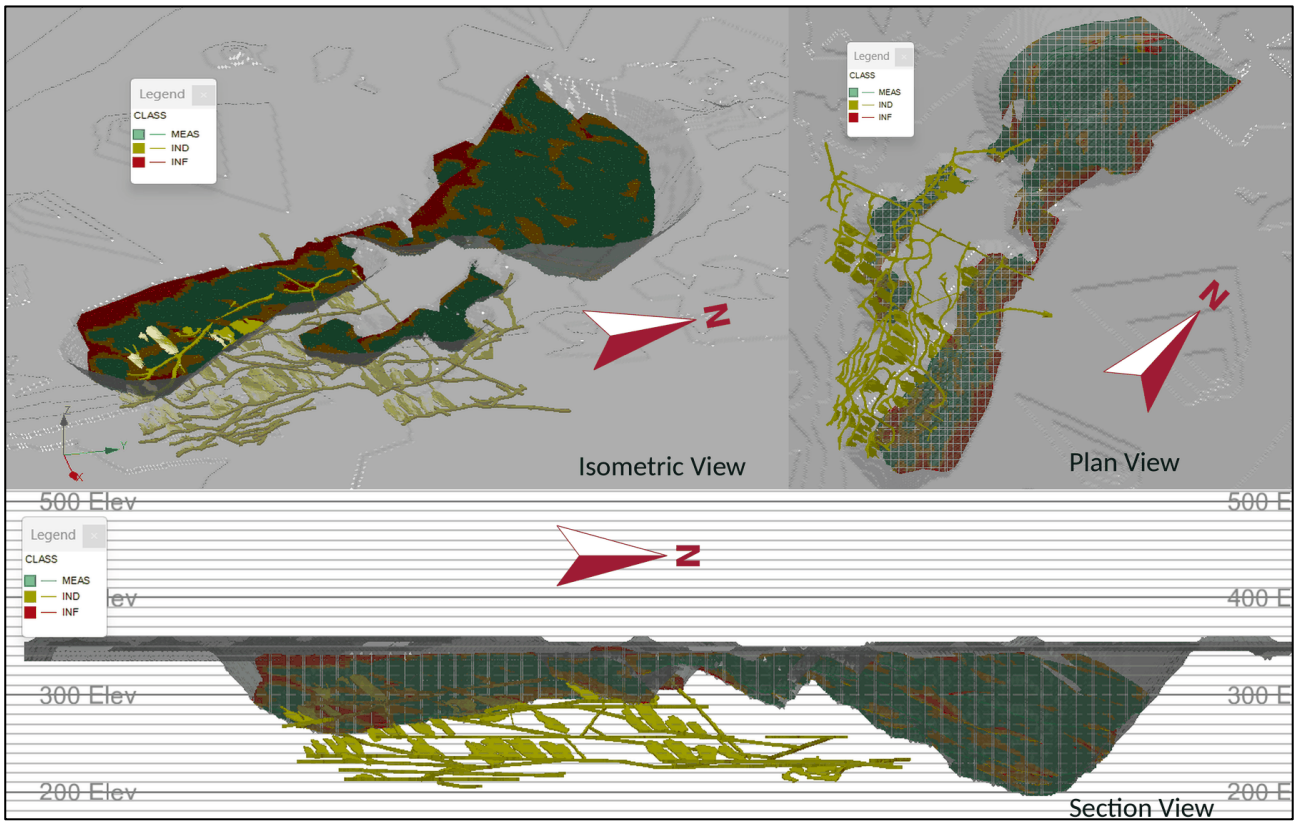


Figure 14-17: Plan view of the RPEEE pit shell with reported blocks by resource class

14.8.3 Underground Constraint

14.8.3.1 Conceptual mining method

The underground constraint assumes mechanized mining of narrow, moderately dipping veins. The vein has a true thickness of 1.2 m to 2.0 m and dips moderately east approximately 30° northeast. The rock mass has a rock mass rating (RMR) of 77.79, and the hanging wall and footwall are competent metavolcanic rocks.

The assumed method is underhand drift mining without backfill (Figure 14-18). Mineralized drifts 2.5 m high are developed on strike from stope access cross-cuts. Each subsequent drift is excavated below the previous one, leaving a supported void above. Sill pillars divide the mining sequence and provide regional stability, with a new mining front started beneath each pillar (Figure 14-19). Production levels are spaced 17.5 m vertically. A 25 m crown pillar separates the underground shapes from the base of the pit shell. Access would be by a new decline from the South-East pit area (Figure 14-20). The method is assumed only to define mineable shapes. It is not a mine design.

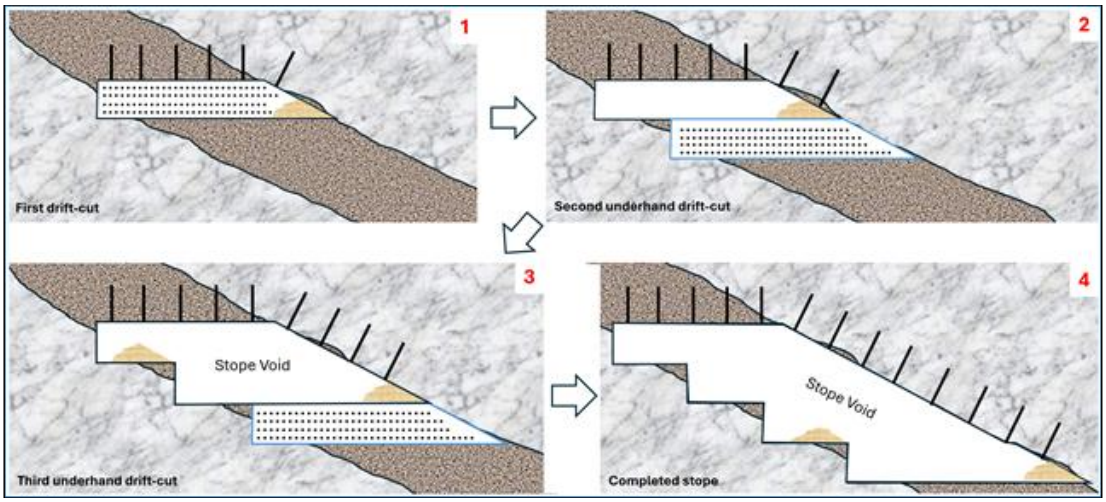


Figure 14-18: Vertical section looking along strike, showing the conceptual underhand drift mining sequence. (Not to Scale)

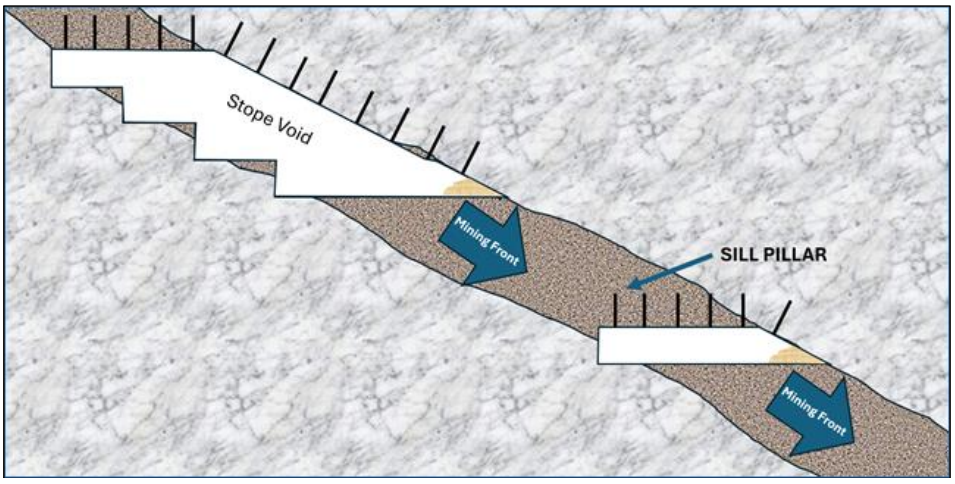


Figure 14-19: Schematic of the relationship between sill pillars and mining fronts. (not to scale).

14.8.3.2 Break-even cut-off grade

Table 14-12 shows the break-even grade (BEG) calculation. Costs are expressed per tonne of run-of-mine (RoM) material. The 3% geological loss reduces both the cost and the recovered metal per in-situ tonne, so it cancels out of the grade.

Table 14-12: Underground break-even cut-off grade calculation

Item	Value	Unit
Gold price	3,000 (96.45)	US$/oz (US$/g)
Process recovery	90	%
Geological recovery (3% geological loss)	97	%
Royalty	3	% of gross revenue
Unplanned dilution	0	%
Production mining – drift	80.98	US$/t Mineralized Material
Mining development	41.16	US$/t Mineralized Material
Treatment plant	26.81	US$/t Mineralized Material
G&A	12.60	US$/t Mineralized Material
Total cost per RoM tonne	**161.54**	**US$/t Mineralized Material**
Total cost per in-situ tonne (161.54 × 0.97)	156.70	US$/t in situ
Net gold price per in-situ gram (96.45 × 0.90 × 0.97 × 0.97)	81.68	US$/g
Break-even in-situ grade (156.70 ÷ 81.68)	**1.92**	**g/t Au**

14.8.3.3 Marginal cut-off grade

The marginal cut-off grade excludes the mining development cost. It assumes that marginal drifts need no additional development, because their access is already paid for by the adjacent economic shapes:

Marginal cut-off = (161.54 − 41.16) × 0.97 ÷ 81.68 = 116.77 ÷ 81.68 = 1.43 g/t Au

14.8.3.4 MSO parameters

Table 14-13 lists the MSO stope shape parameters. Several drift heights and sidewall angles were tested, and the combination in Table 14-13 was selected to minimize dilution and maximize contained gold. The 2.5 m strike increment equals the average planned face advance per blast in the mineralized drifts. Table 14-14 summarizes how other underground factors are treated in the reported Mineral Resources.

Table 14-13: MSO stope shape parameters

Parameter	Value
Reporting cut-off (break-even)	1.92 g/t Au
Marginal cut-off	1.43 g/t Au
Drift height	2.5 m
Drift width	5.5 m to 11 m
Strike increment	2.5 m
Footwall sidewall dip	45°
Hanging wall sidewall dip	35°

Source: ABGM, 2026.

Table 14-14: Treatment of underground factors in the reported Mineral Resources.

Factor	Study value	Treatment in reported Mineral Resources
Planned dilution (waste inside MSO shapes)	~35%	Included, as part of the shape tonnes and grade
Geological loss	3%	Applied
Unplanned dilution	0%	Not applied
Mining recovery (including geotechnical pillars)	83%	Not applied
Mine call factor	100%	Not applied
Tonnage discrepancy	0%	Not applied

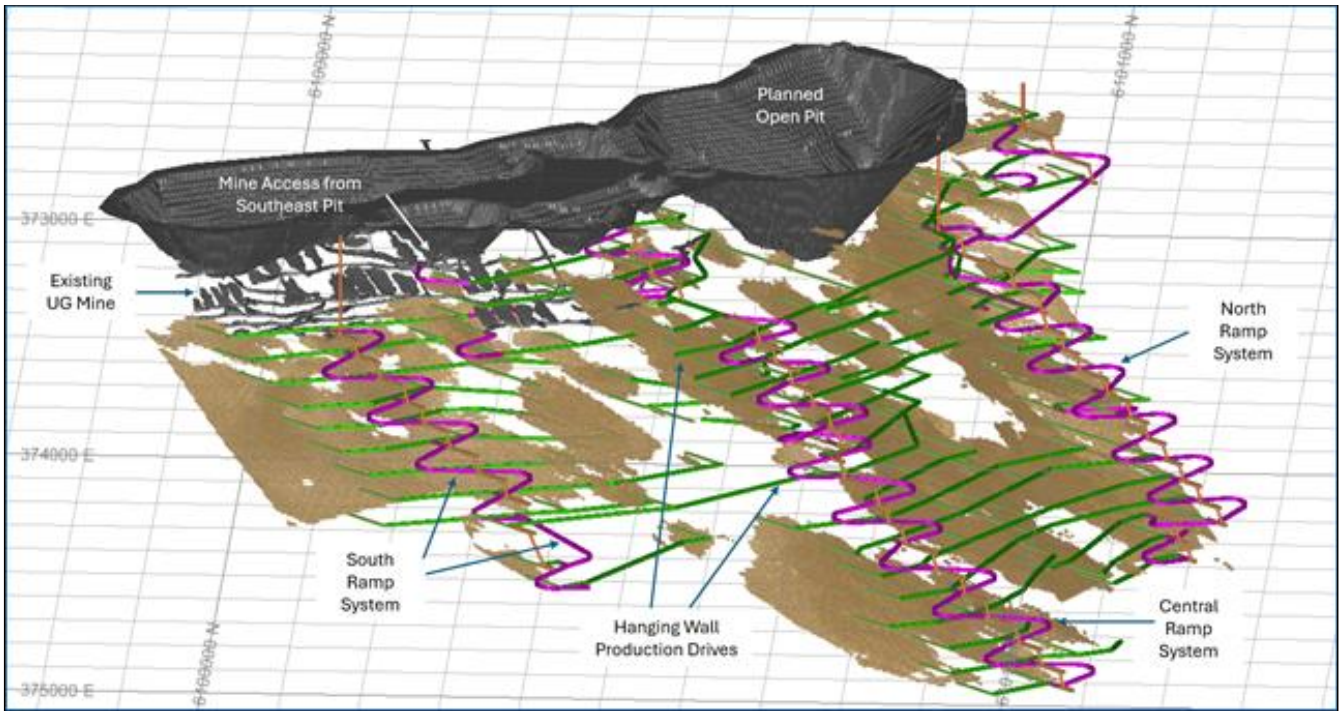


Figure 14-20: Isometric view looking west, showing the planned pit, the existing underground mine and the underground mineralized zones (brown), with conceptual access development. The access layout is conceptual and is not part of the Mineral Resource.

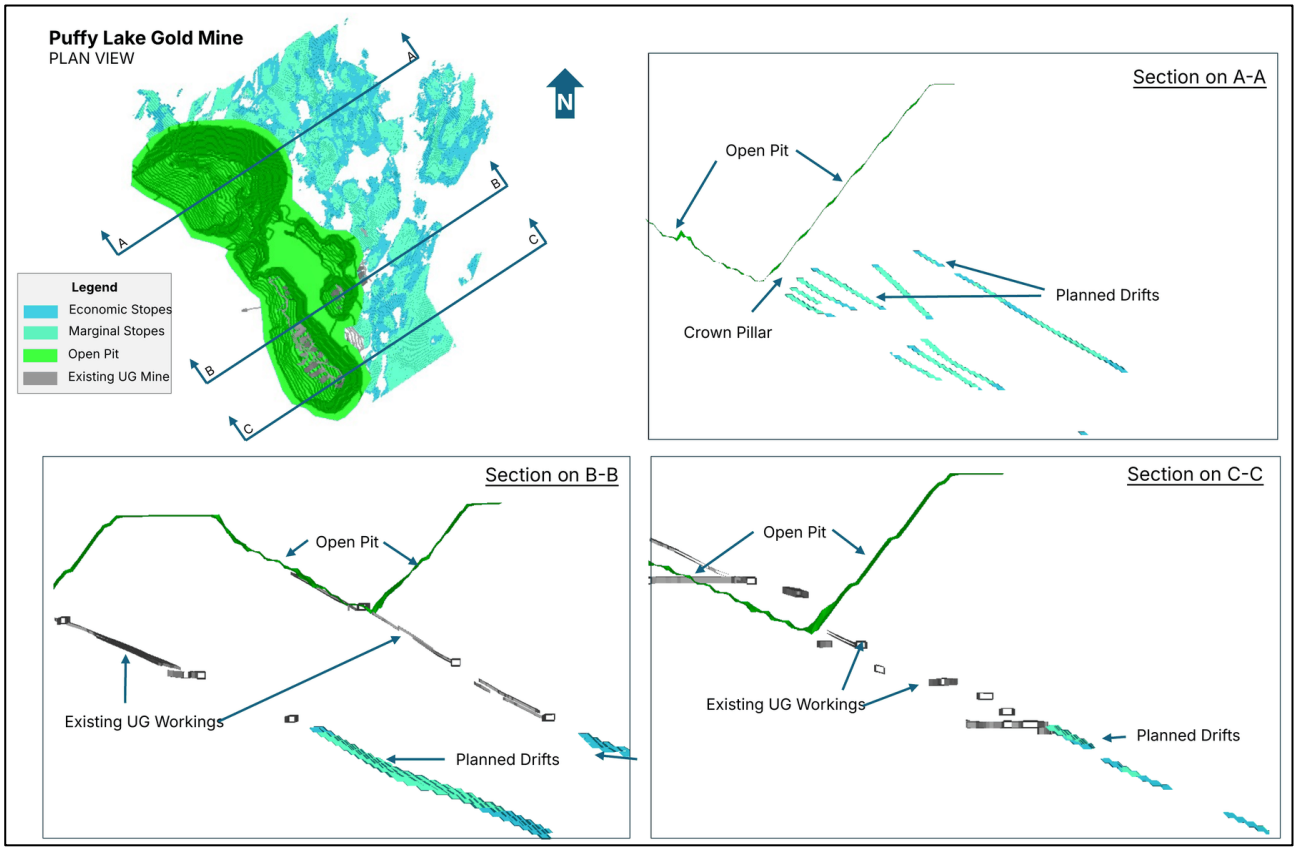


Figure 14-21: Plan view of the RPEEE open-pit shell, the economic and marginal MSO stope shapes and the existing underground mine, with representative cross-sections A-A, B-B and C-C showing the pit shell, the 25 m crown pillar, the existing underground workings and the MSO shapes (labelled planned drifts). The MSO shapes are reporting constraints only and are not a mine design.

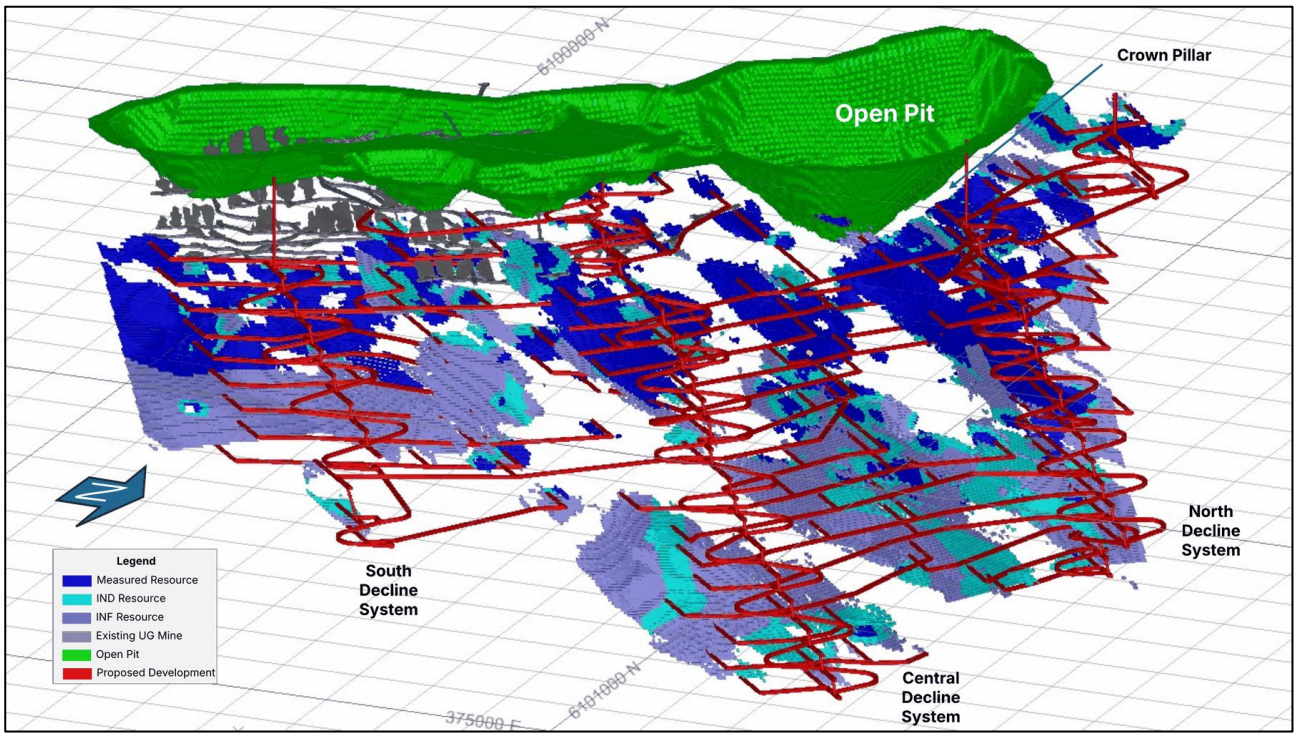


Figure 14-22: Oblique view of the underground Mineral Resources by resource class (Measured, Indicated and Inferred) below the RPEEE open-pit shell and the crown pillar, with the existing underground mine and the conceptual North, Central and South decline access systems (proposed development). The access development is conceptual and is not part of the Mineral Resource.

14.8.4 Open-Pit and Underground Interface and Depletion

The open-pit and underground Mineral Resources are mutually exclusive. The 25 m crown pillar below the pit shell separates the two. Material within the crown pillar is not reported in either category. Double counting was avoided by flagging and depleting blocks inside the pit shell and excluding them from the MSO run, and clipping MSO shapes 25 m below the shell.

The historical 1986–1989 underground workings lie within the SLO orebody and were depleted from the block model before the pit optimization and MSO runs. Depletion was carried out in two steps. First, the block model was depleted using the as-built survey wireframes of the historical stopes, drifts, ramps and raises. Because the updated vein interpretation placed parts of the modelled veins outside the as-built wireframes, some mined-out areas were not fully depleted in that first step. To address this, the QP constructed perimeters around the extent of the mined-out workings, based on the as-built survey wireframes. All blocks within these perimeters were then removed completely from the model, so no partially depleted blocks remain.

No additional stand-off was applied around the workings. The QP considers the perimeter-based depletion conservative because it removes all material within the mined-out extent, including the remnant pillars between historical stopes. These pillars are excluded from the Mineral Resource and represent potential upside, subject to future geotechnical assessment and confirmation that they can be extracted safely.

The depleted volume has not been reconciled with the historical production records of about 350,000 t to 364,000 t milled. Reconciling the depletion against the production records is recommended.

14.8.5 Risks and Uncertainties Relevant to RPEEE

The following project-specific factors could change the reporting limits:

- Historical workings: The 1986–1989 underground workings lie partly within and beneath the conceptual open-pit shell. Any unmapped or inaccurately surveyed voids could reduce the Mineral Resource tonnes near the historical workings and affect pit wall and crown pillar stability. The QP has reduced this risk by depleting all material within the mined-out perimeters. The void model should nonetheless be verified against the historical production records before any mine design is done.
- Metallurgical recovery: The cut-off grades use 90% recovery, based on historical testwork and the 2018 FS. The historical plant achieved about 83% to doré. At 82% recovery, the full-cost open-pit cut-off would be 0.65 g/t Au and the underground break-even grade 2.11 g/t Au. No new metallurgical testwork has been completed for the 2026 MRE.
- Underground mining method: Underhand drift mining without backfill has not been used at PL, where historical stoping was up-dip slusher mining. The stability of unfilled drift voids, the sill pillar spacing and the 25 m crown pillar beneath the pit are not yet supported by detailed geotechnical analysis. The underground costs are based on contractor prices applied to a generic design and on benchmarking of similar operations.
- Cost and price assumptions: Costs are in US$. The open-pit costs are taken from the September 2026 optimization input workbook (Table 14-11) and, like the underground costs, are based on 2026 contractor rates and benchmarking against similar operations. The selling cost and haulage to mill are included in the open-pit cut-off but not in the underground cut-off. Including them would raise the underground cut-offs to about 1.98 g/t Au (break-even) and 1.48 g/t Au (marginal). The US$3,000/oz gold price is a long-term planning assumption below the spot price at the time of assessment
- Permitting and tenure: The existing Environment Act Licence 1207E relates to the historical underground operation and the existing plant. An open pit would require an amendment to reflect the addition of open pit mining methods and expansion to 2,000 tpd. The RPEEE open pit shell lies almost entirely within Mineral Lease ML065. A sliver of approximately 0.1 ha on the western rim

extends up to about 25 m beyond the lease boundary as mapped in iMaQs, onto the adjoining claim MB12247, which is also held by Minnova Corp. The iMaQs boundary is not a legal survey, and this overlap is within plausible digitizing error; it should be checked against the ML065 survey plan. The pit shell should also respect the required setbacks from water bodies, the tailings management area and the plant.

In the opinion of the QP responsible for this subsection, the parameters above are reasonable for demonstrating RPEEE at the Effective Date. The Mineral Resources within the pit shell and the MSO shapes have reasonable prospects for eventual economic extraction.

15 MINERAL RESERVE ESTIMATES

No Mineral Reserves have been estimated for the PL Gold Mine. This Technical Report is a preliminary economic assessment, and the mine plan and economic analysis use Measured, Indicated and Inferred Mineral Resources. The Mineral Reserves reported in the 2018 FS (effective October 31, 2017) are historical and not current, and are not relied upon.

16 MINING METHODS

16.1 Introduction

The PL Gold Mine is planned as a contractor-mined open pit (OP) and underground (UG) operation feeding the refurbished PL process plant. In the PEA cash flow model, open pit mining runs for three years, from 2028 to 2030, and delivers 1.76 million tonnes (Mt) of mineralized material at 4.68 g/t Au with 36.91 Mt of waste. Underground mining starts in 2029 and delivers 4.74 Mt of mineralized material at 2.80 g/t Au to 2037. The plant ramps up to 2,000 tonnes per day (t/d) during 2028 and treats 6.50 Mt at 3.30 g/t Au over the life of mine (LOM), with 621,513 oz of gold recovered at 90% recovery.

The geotechnical design parameters are from the final geotechnical study by R. Lousteau (2026). The ventilation design is from the Umoya Occupational Hygiene & Laboratory Solutions VENTSIM LOM study. Hydrogeology and dewatering are from the Water Hunters (2026) hydrogeological desktop study and gap analysis.

16.2 Geotechnical Design Parameters

This section summarizes the PEA-level geotechnical study completed by the ABGM Geotechnical Department for the PL open pits, the pit-to-underground crown pillar, the modified drift-and-fill (MDF) underground method and the regional stability of the underground layout. It summarizes the final geotechnical study by R. Lousteau (2026). The parameters are envelopes for pit-shell and stope design at PEA level; they are not a construction specification.

16.2.1 Data Sources and Limitations

The rock-mass work is based on the February 1987 Piteau Associates investigation (A.F. Stewart, P.Eng.), which included structural line mapping of the decline and a test stope, core logging and point-load testing, together with re-processed geotechnical logs from more than 60 boreholes (mainly the M-17 and M-18 series; more than 4,500 logged intervals), sections 300S, 900S and 1500S, and the mining-method description in the 2017 Feasibility Study. The logs were classified with RMR89 (Bieniawski), the Barton Q-system and the Laubscher MRMR, and design rock-mass strength (DRMS) was derived from these values.

The main limitations of the data set are:

- Hanging wall, mineralized zone and footwall currently share one pooled Q and rock mass rating (RMR), because the inherited log codes do not separate the domains. The hanging wall, which controls crown and stope stability, must be classified separately.
- Intact strength comes from historical point-load conversions. No laboratory uniaxial compressive strength (UCS) tests loaded parallel and normal to the foliation are available.
- Discontinuity orientations have not been measured (no oriented core or borehole imaging), so wall-by-wall kinematic analysis is not yet possible.
- In-situ stress has not been measured; a horizontal-to-vertical stress ratio of k = 1.0 is assumed.
- No packer tests, piezometers or groundwater model are available. All factors of safety (FoS) are dry and elastic, and none is an effective-stress design.

16.2.2 Lithology and Rock Mass Characterization

The dominant host rock is Nokomis Group greywacke gneiss with biotite and garnet porphyroblasts. Gold is hosted in several sub-parallel quartz-sulphide veins (Sherridon, Upper, Main and Lower zones) with true thicknesses of about 1.5 m to 2.5 m and dips of about 30° to 32° to the northeast. Waste rock subtypes include hornblende-biotite gneiss, quartz-biotite gneiss, muscovite-biotite gneiss/schist, quartz porphyry and foliated granite. Lithological boundaries generally parallel the gold-bearing horizons.

Rock quality is consistently good. Most rock quality designation (RQD) values fall between 70% and 100% and average 75% to 85% with depth, with isolated drops below 50% at fault zones and granite intrusions. The 15 boreholes of the M-17-007 to M-17-040 series average 75% to 82% RQD. Most 3 m core runs record 5 to 15 fractures; runs with more than 25 fractures are associated with faulted or broken ground and will need targeted support. High core recovery (more than 95%) is almost always accompanied by high RQD (more than 75%), which supports the reliability of the logging.

The pooled classification gives a mean Barton Q of 8.89 (range 1.60 to 12.48) and a mean RMR89 of 77.79 (range 56.48 to 87.25; standard deviation 6.35). After Laubscher adjustments for blasting (0.97), weathering (0.96) and joint orientation (0.96), with mining-induced stress and water set to 1.00 at the planned depths,

MRMR is 69.54 (63.86 at one standard deviation below the mean). Intact UCS is in the range 180 MPa to 300 MPa (very strong, brittle gneiss), and density is 2,750 kg/m³ to 2,754 kg/m³.

Table 16-1: Derived rock-mass classification and design rock-mass strength

Unit	RMR89	Q	Q'	MRMR	UCS (MPa)	DRMS (MPa)
Mineralized Zone	77.79	8.89	22.24	69.54	180-300	92.99
Mineralized Zone (±1 StDev)	71.44	6.96	17.40	63.86	180	82.77
Hanging Wall	77.79	8.89	22.24	69.54	180-300	92.99
Hanging Wall (±1 StDev)	71.44	6.96	17.40	63.86	180	82.77

Source: ABGM Geotechnical Department (2026).

Table 16-2: Key intact rock properties

Property	Value	Comments
UCS (Mean)	200 - 300 MPa	Very strong, brittle gneiss
Density	2,750 - 2,754 kg/m³	Standard for Canadian Shield gneiss
Point Load Index	Ongoing testing	Will refine UCS variability
Mineralized Zone Width	1.5 - 2.4 m	Narrow vein - favours selective mining

Source: ABGM Geotechnical Department (2026).

16.2.3 Open Pit Slope Design Parameters

Slope angles were derived empirically from the MRMR slope-design chart of Haines and Terbrugge (1991), the Ryan and Prior berm-width relationship and SME guidance, for a maximum pit depth of 120 m. The recommended base case uses 10 m benches, with 5 m benches where near-surface weathering or structural complexity requires better selectivity. The Ryan and Prior relationship (W = 0.2H + 4.5 m) gives a berm of about 5.2 m for a 10 m bench; a 6.5 m catch berm is adopted to allow for back-break and jointed ground. Stacks are limited to about 40 m, with a geotechnical berm of at least 10.3 m between stacks. Ramp widths of 15 m to 20 m at a 10% grade are proposed for haulage.

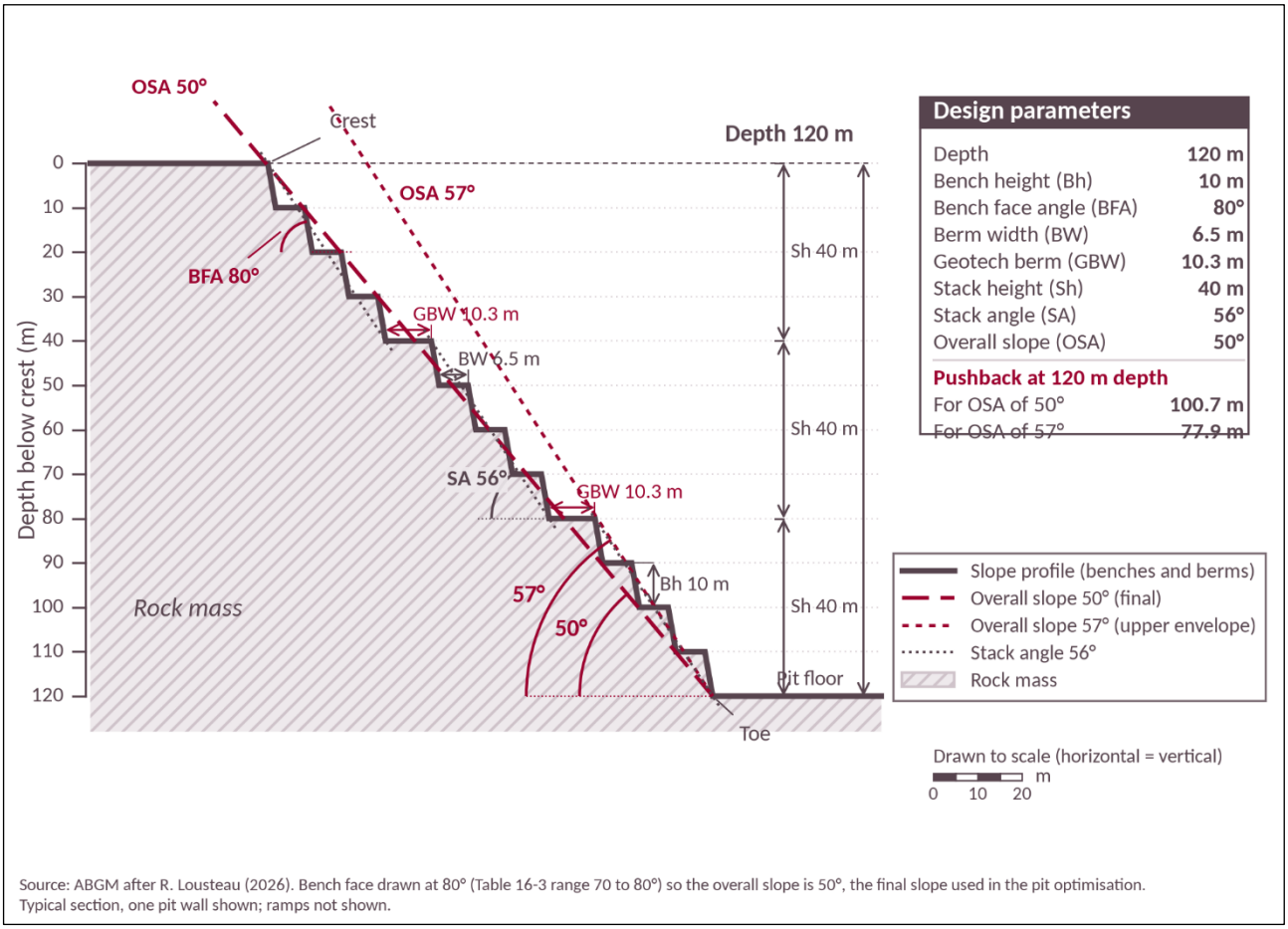
Table 16-3: Recommended slope geometry for a 120 m pit depth (base case)

Parameter	Value	Notes
Total Pit Depth	120 m	Maximum
Bench Height	10 m	Standard
Number of Benches	12	-
Bench Face Angle (BFA)	70-80° (target avg. 75°)	Trigonometric derivation; structural control
Catch Berm Width	6.5 m	Conservative
Stack Height (max)	40 m	Before geotech berm
Geotech Catch Berm	≥10.3 m	Between stacks
Overall Slope Angle (OSA)	52-65°	Domain dependent
Inter-Ramp Angle (IRA)	52-58°	Including ramps

Source: ABGM Geotechnical Department (2026).

Final walls are designed for a FoS of about 1.5, giving overall slope angles (OSA) of 50° to 57°. Temporary or interim walls designed for a FoS of about 1.2 may be steeper (OSA 60° to 65°). Inter-ramp angles, including berms, are expected to be 52° to 58°. Walls that contain faults, or where foliation dips out of the face, must be designed flatter than these ranges.

Slope envelopes are given for good rock and, separately, for faulted, wet or daylighting walls. On good, dry walls where joints do not daylight, inter-ramp angles are 52-58° and overall slopes are 50-62°. An overall slope of 65° is an upper bound for that class of wall only, not the pit default. A 15 m thickness of competent rock is left between the pit floor and the first underground stope. Soil and weathered rock do not count toward the 15 m. Stereonets, limit-equilibrium sections, pore-pressure grids and blast-damage calibration are pre-feasibility work, not part of this PEA.



Source: ABGM after R. Lousteau (2026). Bench face drawn at 80° (Table 16-3 range 70 to 80°) so the overall slope is 50°, the final slope used in the pit optimisation. Typical section, one pit wall shown; ramps not shown.

Figure 16-1: Schematic of the recommended overall slope angle and bench face angle.

The pit design parameters in the 2026 project description prepared for the Environment Act submission (AECOM, draft 2026) differ from these recommendations: footwall (western) slopes of 35°, hanging-wall (eastern) slopes of 70° with benches, 5 m benches, 20 m ramps, inter-ramp angles of 50°, 6.28 m catch berms and a North Pit depth of 65 m to 174 m. The 174 m depth exceeds the 120 m depth used for the geotechnical envelope.

The main potential failure modes are wedge and planar sliding on intersecting discontinuities and faults, circular failure in near-surface weathered or intensely fractured ground, and toppling where steep foliation or joints are adversely oriented. At PEA level these are mitigated by 3 m crest berms and surface drainage, controlled (low-damage) perimeter blasting, face scaling, slope monitoring (prisms, InSAR and crack gauges), and locally flatter slopes in weak domains. Groundwater pressure is not yet characterized and could reduce stability.

16.2.4 Crown Pillar between the Open Pit and Underground

The crown pillar between the final pit floor and the first underground stopes was sized with the empirical Scaled Span method of Carter (restated 2014). The scaled span Cs = S[γ/(T(1 + S/L)(1 − 0.4cosθ))]^0.5 is

compared with the critical scaled span Sc = 3.3Q^0.43, and a design margin of Cs = 0.7Sc is applied. The design Q is the mean minus one standard deviation (Q = 6.96); the minimum Q of 1.60 is treated as a sensitivity case.

where S is the mined crown span (m), T is the vertical thickness of the rock lid (m), γ is specific gravity (numerically equal to t/m³), θ is the dip of the orebody or controlling foliation, and SR = S/L is the span-to-strike ratio. A long, narrow stope has a small SR and is more onerous than a short stub of the same width. A steep stope does not receive the dip benefit that a flat opening receives.

If Cs < Sc, the case plots in the empirical stable field. If Cs ≈ Sc, the case is in transition. If Cs > Sc, the case plots in the caving field. Inverting the Cs equation for thickness gives the minimum T on the transition line:

Table 16-4: Geometry adopted for the crown-pillar assessment

Parameter	Value used	Basis
Orebody true thickness	1.5 - 2.5 m	Table 4-3 / previous reports
Design mined span, S	4.0 m (check 3.5 m and 6.0 m)	Vein width plus drive / dilution; 4.0 × 3.5 m mineralized drive
Unsupported strike length, L	35 m	2018 FS maximum level spacing
Span ratio, SR = S/L	0.11 at S = 4 m	Calculated
Dip, θ	70°	Steep shear assumption pending mapping confirmation
Unit weight / specific gravity, γ	2.75 t/m³	Density 2,754 kg/m³, Table 4-3
Stope method under the pit	First stope assumed empty	Backfill cannot be relied on for the first opening

Source: ABGM Geotechnical Department (2026).

Table 16-5: Required crown-pillar thickness by mined span

Mined span S (m)	T at mean Q, limit (m)	Design T at Q-1 SD and 0.7 Sc (m)	T at Q = 1.60, limit (m)	T at Q = 1.60 and 0.7 Sc (m)
3.0	0.4	0.9	1.6	3.3
3.5	0.5	1.3	2.2	4.4
4.0 (design)	0.6	1.6	2.8	5.7
5.0	1.0	2.5	4.3	8.7
6.0	1.4	3.5	6.0	12.3

Source: ABGM Geotechnical Department (2026).

On the empirical chart, a 4 m wide stope in rock with Q of about 7 needs only about 1.6 m of competent rock. The practical design thickness is governed instead by the pooled (unseparated) hanging-wall Q, blast damage and weathering of the pit floor, and possible coalescence of stacked shears into wider openings.

Underground stoping should not be scheduled into the 15 m zone immediately below the designed pit floor. That rock is a regional pillar. Recovery of any high-grade remnant inside the crown would be a separate, later

study and is not part of the base-case PEA extraction. If the pit is deepened after underground mining has started, the remaining lid must be re-checked before the floor is taken down.

Where several stacked shears will be mined as one opening, the span S used in the chart must be the full mined width, not the 1.5-2.5 m vein thickness. A 6 m opening at poor Q already requires a Carter thickness of more than 10 m before the practical pit-floor allowance is added.

For the PL open-pit-to-underground transition, a 15 m vertical thickness of competent rock should be left below the final pit floor as the design crown pillar. The Scaled Span calculation shows that a much thinner lid would plot in the stable field for mean Q and a 4 m span. That thin result is not adopted because the hanging-wall Q is averaged, the pit floor will be damaged, and the crown must carry surface operations. The following are recommended for the overall report and the PEA mine plan:

Table 16-6: Recommended crown-pillar thickness

Design case	Vertical thickness below final pit floor	Application
Carter chart only, Q confirmed ≥ 7, S ≤ 4 m, fresh unblasted rock	5 m	Absolute empirical minimum plus a blast-damage skin. Not for PEA base case.
Recommended PEA / FS design	15 m	Pit-to-underground transition until hanging-wall Q is measured separately and the first-stope span is confirmed.
Lower bound if hanging-wall Q ≥ 8 and pit floor is proven fresh	10 m	Only after new mapping and logging. Do not go below 10 m on the present dataset.
Poor hanging wall (Q near 1.6), stacked shears, or S = 5-6 m	20 - 25 m	Do not start underground mining under the pit on the mean-Q table alone.

Source: ABGM Geotechnical Department (2026).

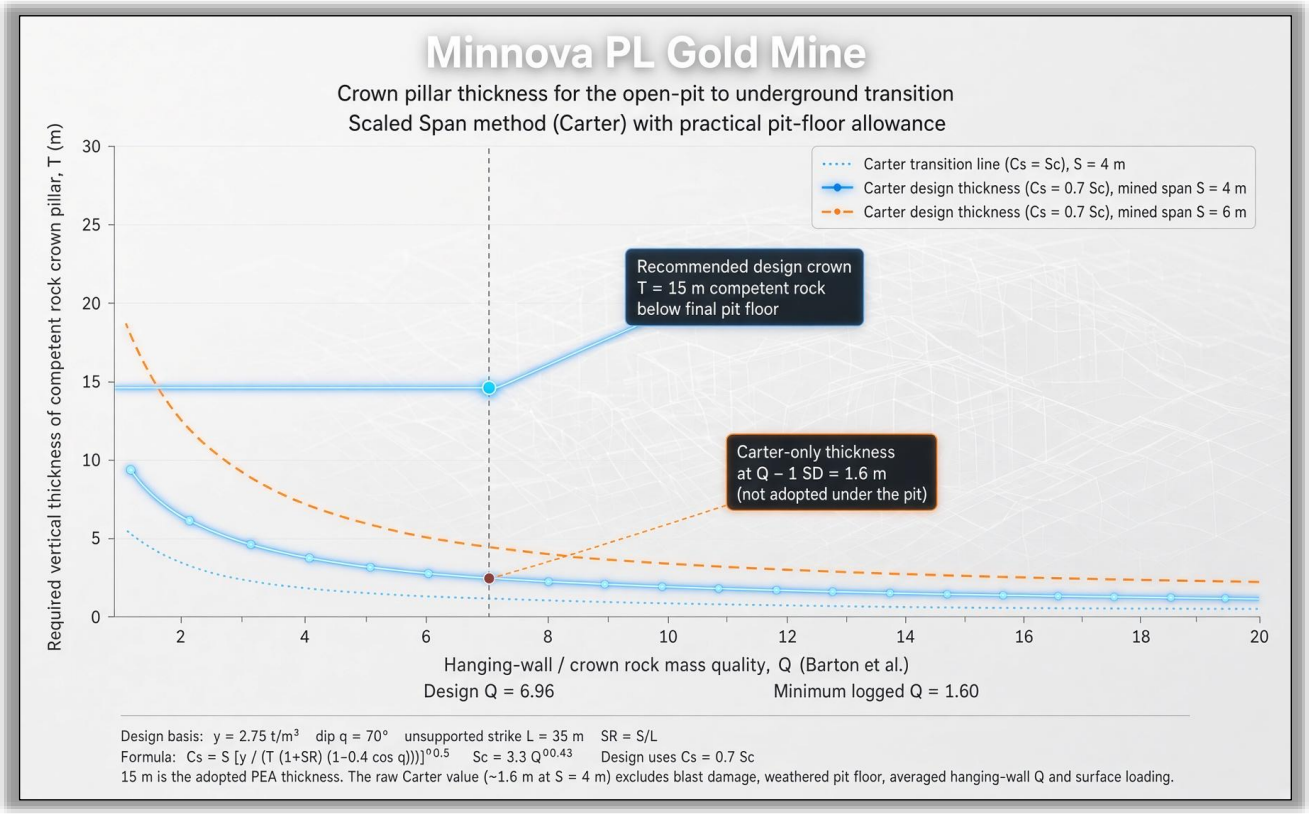


Figure 16-2: Required vertical crown-pillar thickness versus Q for the pit-to-underground transition

A 15 m vertical thickness of competent rock below the final pit floor is recommended as the base-case crown pillar, and no stoping should be scheduled inside that zone. The design span is S = 4 m, with a check at S = 6 m where stacked shears coalesce. The geometry table assumes a 70° dip, while the deposit dips at about 30°; a flatter dip increases the required thickness.

16.2.5 Underground Design Parameters

The proposed modified drift-and-fill (MDF) method develops a series of controlled drifts along strike on the vein horizon, each followed by backfill, to form a stepped, filled panel. The key design parameters are an optimum vertical level spacing of 17 m, a maximum strike panel length of 250 m and a footwall face of at least 50°. The footwall face reduces the tonnage mined by about 10% and raises the head grade by about 6% by limiting external dilution.

This modified Drift and Fill mining method does result in an increase in the effective stoping width compared to a single drift. A single drift typically measures between 5 m and 11 m in width and extracts only a portion of the mineralized vein. However, by developing successive adjacent drifts and applying backfill as required, these individual drifts are combined to form a larger composite extraction panel. This approach enables effective stoping widths of 20 m to 40 m (or greater), depending on the vein geometry and mine design parameters.

The deposit's competent rock mass (UCS 200-300 MPa, RMR 77.79 ±6.35, Q 8.89, MRMR 69.54) provides excellent stand-up time and self-supporting capability for the MDF geometry. The high DRMS (≈93 MPa) and low fracture frequency (average 8-12 per 3 m) mean that the stepped backfilled panels will remain stable even at the proposed 17 m vertical spacing.

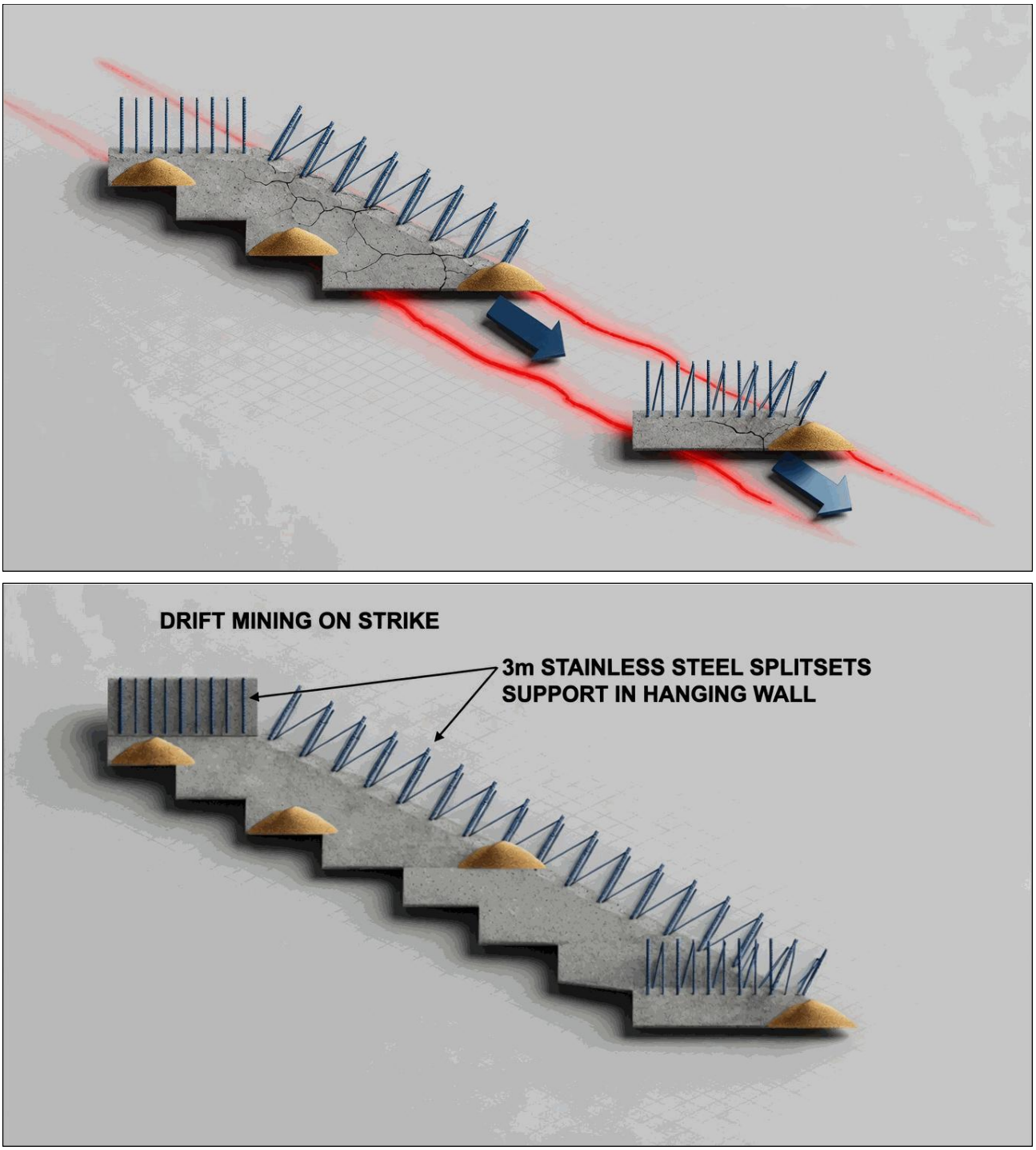


Figure 16-3: Schematic of the modified drift-and-fill mining method

Hanging-wall stability was assessed with the Potvin stability graph. The stability number N′ = Q′ × A × B × C is about 47.4 for the mean rock mass (about 37.1 at one standard deviation below the mean), corresponding to a critical hydraulic radius (HR) of about 9.5 m. The Laubscher MRMR approach gives a critical HR of 10 m to 12 m.

Table 16-7: Stability number N′ and critical hydraulic radius

Rock domains	Stability Number	Critical Hydraulic radius
Mineralized zone	47.36	9.46
Mineralized zone -1 St-dev	37.07	9.00
Hangingwall	47.36	9.00
Hangingwall - 1 St-dev	37.07	9.00
Rock domains	**MRMR**	**Critical Hydraulic radius**
Mineralized zone	69.54	10.00
Mineralized zone -1 St-dev	63.86	12.00
Hangingwall	69.54	10.00
Hangingwall - 1 St-dev	63.86	12.00

Source: ABGM Geotechnical Department (2026).

Table 16-8: Hydraulic radius and recommended support by drift width (drift height about 5 m, 30° dip)

Drift Width (m)	Approximate HR (m)	Stability Rating	Recommended Support
5-7	4.9 -7.0	Excellent (FoS >> 2.0)	Standard bolts + mesh
8-9	7.5 - 8.5	Stable (FoS 1.6-2.0)	Bolts + shotcrete + occasional cables
10-11	9.0 - 10.5	Marginal (FoS ≈ 1.3-1.5)	Enhanced cable bolting + fibrecrete + monitoring
>11	>10.5	Unstable without heavy support	Not recommended as primary drift

Source: ABGM Geotechnical Department (2026).

Primary drifts should be limited to 5 m to 8 m wide. Spans of 9 m to 11 m are acceptable only where RQD exceeds 80% and RMR exceeds 78, and only with enhanced cable bolting, fibre-reinforced shotcrete and monitoring. Drifts wider than 11 m are not recommended. A maximum strike span of about 240 m is suggested for the stope panels. A Voussoir beam analysis of the jointed hanging wall indicates that a competent beam about 1.2 m thick is needed for the planned panel spans.

Because the rock mass is competent and the workings are shallow, pillar requirements are modest: in-panel pillars of about 2.0 m to 3.5 m wide and regional pillars of about 6 m to 14 m wide are indicated between 100 m and 500 m depth. Historical in-stope crush pillars were 3 m to 5 m wide. The Hedley and Grant pillar-strength formula with tributary-area loading was used to size pillars for a width-to-height ratio greater than 2.5 and a FoS greater than 2.0, to 500 m depth, the approximate depth limit of the drilled resource.

Table 16-9: Pillar dimensions and extraction by depth (to 500 m)

Depth	Stope Span	Level Spacing	In Panel Pillar Width (m)	In Panel Pillar Length (m)	Regional Pillar Width (m)	Regional Pillar Length	Panel Span	Vent Holing	Extraction Ratio
100	200	200	2.0	3.0	6.0	18.0	20.0	2.0	95%
150	200	200	2.5	3.0	8.0	24.0	20.0	2.0	93%
200	200	200	3.0	3.0	10.0	30.0	20.0	2.0	92%
250	200	200	3.5	3.0	10.0	30.0	20.0	2.0	91%
300	200	200	3.5	5.0	10.0	30.0	20.0	2.0	89%
350	200	200	4.0	4.5	12.0	36.0	20.0	2.0	89%
400	200	200	4.0	4.5	12.0	36.0	20.0	2.0	86%
450	200	200	3.5	6.0	12.0	36.0	20.0	2.0	86%
500	200	200	3.5	6.0	14.0	42.0	20.0	2.0	86%

Source: ABGM Geotechnical Department (2026).

The recommended ground support is:

- Primary: 3.0 m grouted split sets or 2.4 m resin bolts on a 1.5 m × 1.5 m pattern with weldmesh.
- Secondary: 6 m to 8 m cable bolts in wider drifts and abutment zones.
- Shotcrete: 25 mm to 50 mm in all drifts, fibre-reinforced in 8 m to 11 m spans.
- Monitoring: seismic array, convergence pins and backfill pressure sensors, with targeted de-stress blasting in high-stress crown pillars where needed.

Table 16-10: Underground geotechnical risks and mitigation

Risk	Likelihood	Consequence	Mitigation
Excessive dilution	Medium	High	Strict footwall optimization + backfill QA/QC
Instability in wide drifts	Low-Medium	High	Limit width to 8 m + enhanced support
Sill pillar failure	Low	Very High	Minimum 8-10 m pillars + numerical modelling
Backfill stability	Low	Medium	Lab testing + curing monitoring

Source: ABGM Geotechnical Department (2026).

Underground access was planned in the geotechnical draft through rehabilitation of the existing portal and decline (about 7,000 m of historical development to about 135 m depth). The PEA instead uses a new portal in the north-east pit (373,663.6 E, 6,100,247.1 N, 292.0 m RL). The historical workings will be dewatered and rehabilitated. The underground mining study proposes a trial mine of about six months through the existing portal and decline, and an underground test-mining and bulk-sampling program of up to 20,000 t starting at a vertical depth of about 140 m; these programs are PFS recommendations and are not included in the PEA mine schedule or cash flow model.

16.2.6 Regional Numerical Modelling

A three-dimensional elastic boundary-element model of the PEA underground layout was built in Map3D from the supplied DXF files. It used a hybrid fictitious-force/displacement-discontinuity (FF/DD) approach and multi-step mining and backfill sequencing. The model was used to compare sequences and the effect of fill on stress, closure and surface displacement. The input parameters are summarized below. The UCS (140 MPa) and density (3,218 kg/m³) used in the model differ from the classification values (180 MPa to 300 MPa and 2,754 kg/m³).

Table 16-11: Input parameters used in the numerical analysis

Parameter	Value
Uniaxial Compressive Strength (UCS)	140.0 MPa
Young's Modulus (E)	90.0 GPa
Poisson's Ratio (υ)	0.31
Density (ρ)	3,218 kg/m³
K - ratio	1.0
Stress variations	
σxx	-0.027 MPa
σyy	-0.027MPa
σzz	-0.027MPa
Backfill Parameters	
Uniaxial Compressive Strength (UCS)	0.5-1 MPa (Ideal 3MPa)
Young's Modulus (E)	200 -600 MPa
Poisson's Ratio (υ)	0.21
Density (ρ)	1800-2,200 kg/m³
K - ratio	1.0

Source: ABGM Geotechnical Department (2026).

Table 16-12: Key results of the Map3D scenarios

Parameter	No Backfill	Partial Backfill	Full Backfill	Optimized Backfill
Max Stope Closure (m)	0.090	0.015	0.005	0.003
Max σ_1 (MPa)	73-80	45	30	25
Max Surface Subsidence (m)	0.005	0.0015	<0.0005	<0.0005
Minimum Regional FS (Hoek-Brown)	1.1	1.5	1.8	2.1

Source: ABGM Geotechnical Department (2026).

With empty stopes the model shows peak major principal stress (σ_1) of about 73 MPa to 80 MPa on abutments and hanging-wall contacts, maximum closure of about 90 mm and surface displacement of about 5 mm. Placing fill progressively, at about 30% of the extracted volume and targeted at the larger spans, with primary stopes filled before the adjacent secondary stopes are mined, reduces closure to less than 15 mm and σ_1 by 40% to 55%. Full or optimized fill reduces closure to 3 mm to 5 mm. The factors of safety of 1.1 to 2.1 from

these runs are elastic ranking indices that show the improvement once fill is placed; they are not equivalent to design factors of safety. Results were robust to ±10% changes in rock-mass modulus and ±20% changes in fill stiffness.

Hydraulic sand fill or development waste is the PEA fill basis (modulus of about 1 GPa), with cemented fill (modulus of about 8 GPa) in critical stopes. The model should be updated with as-built geometry and fill properties every five stopes.

16.2.7 Hydrogeology

The hydrogeological parameters are from the Water Hunters (2026) hydrogeological desktop study and gap analysis (draft version 1.0, October 19, 2026). No field hydrogeological investigation has been completed for the PEA: groundwater enters the geotechnical work only as a classification rating, and all geotechnical factors of safety assume dry conditions.

16.2.7.1 Hydrogeological Setting and Existing Data

Groundwater movement is expected to be governed primarily by fractures, faults, mineralized shear zones and quartz-carbonate veins in otherwise low-permeability crystalline bedrock. Shallow groundwater levels measured in historic boreholes ranged from approximately 0.005 m to 3.31 m below ground surface and closely followed local topography. Ragged Lake and the Fyre Lake swamp may provide recharge through hydraulically connected structures, particularly during spring snowmelt.

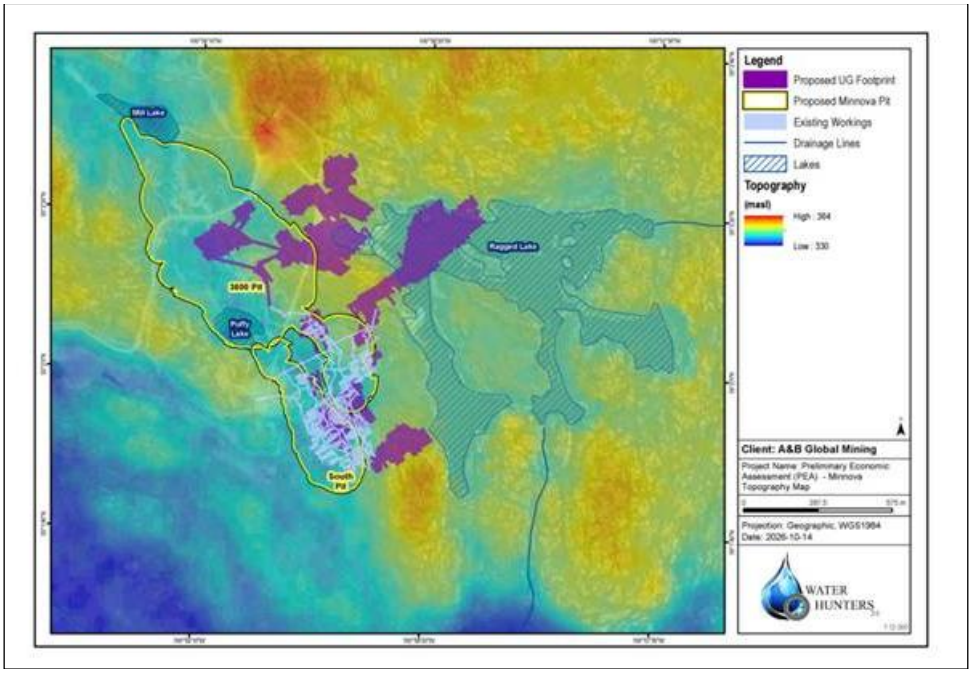


Figure 16-4: Topography and surface-water drainage in relation to proposed mine infrastructure

Source: Water Hunters, 2026, Figure 2-2

Historic aquifer tests were generally short and shallow relative to the proposed underground mining depth. Their bulk conductivity estimates do not isolate individual transmissive fractures. The deeper aquifer and potential hydraulic connectivity with surface-water features remain material uncertainties.

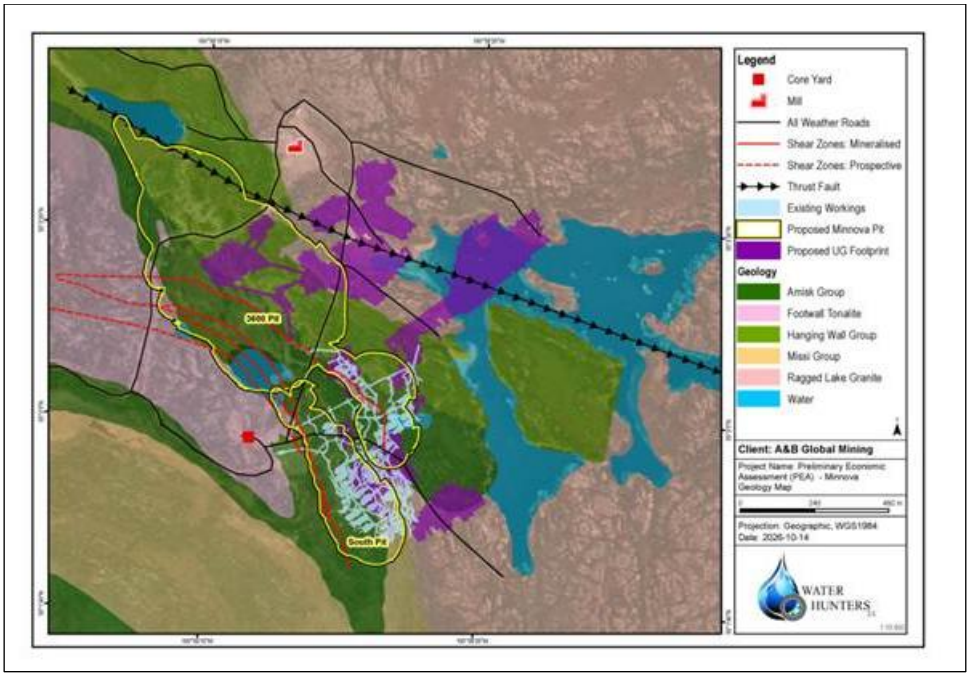


Figure 16-5: Geological units and interpreted structural features in the project area

Source: Water Hunters, 2026, Figure 3-1

Table 16-13: Historic groundwater monitoring boreholes

Borehole	Collar elevation (masl)	Hole length (m)	Static level (mbgs)	GW elevation (masl)
A3-06	342.320	87.48	0.27	342.050
A3-09	350.940	185.00	1.75	349.190
A3-32	347.800	38.71	1.96	345.840
A3-42	352.310	194.15	1.78	350.530
A3-45	345.000	63.09	0.49	344.510
A3-59	346.100	47.85	0.24	345.860
A3-60	347.500	57.00	3.31	344.190
A3-67	343.100	38.71	0.48	342.620
A3-87	341.800	41.76	0.005	341.800
A3-98	341.800	60.35	0.63	341.170

Source: Water Hunters, 2026, Table 3-1

Note: Borehole A3-42 was inclined; reported length is not vertical depth. Historical measurements date from 2011 to 2012.

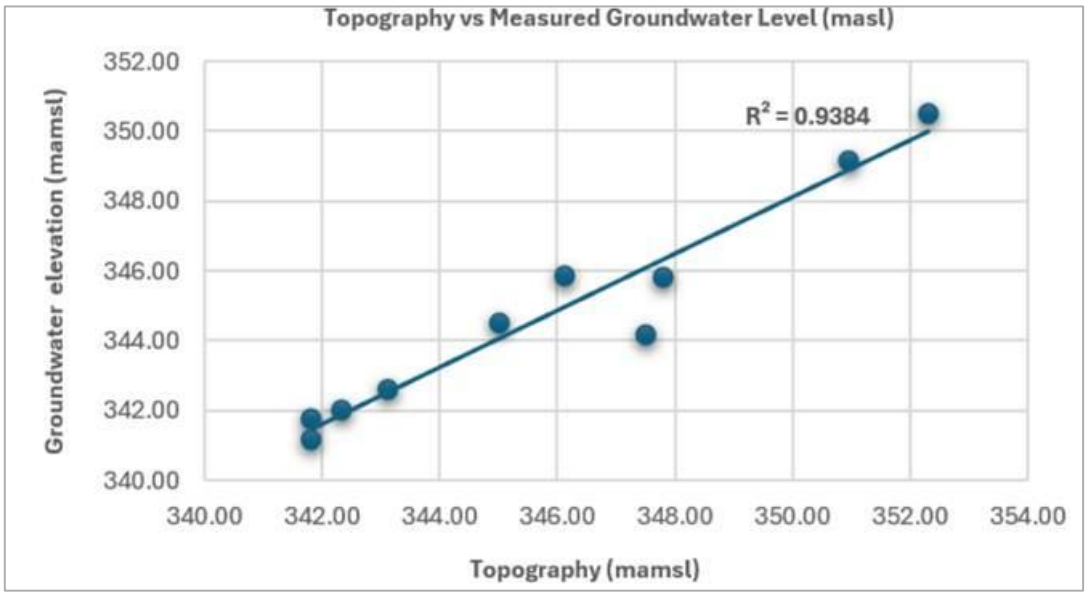


Figure 16-6: Relationship between topography and measured groundwater elevation

Source: Water Hunters, 2026, Figure 3-2

The strong relationship between ground and groundwater elevations describes the shallow monitored system and must not be interpreted as evidence of hydraulic continuity at mining depth.

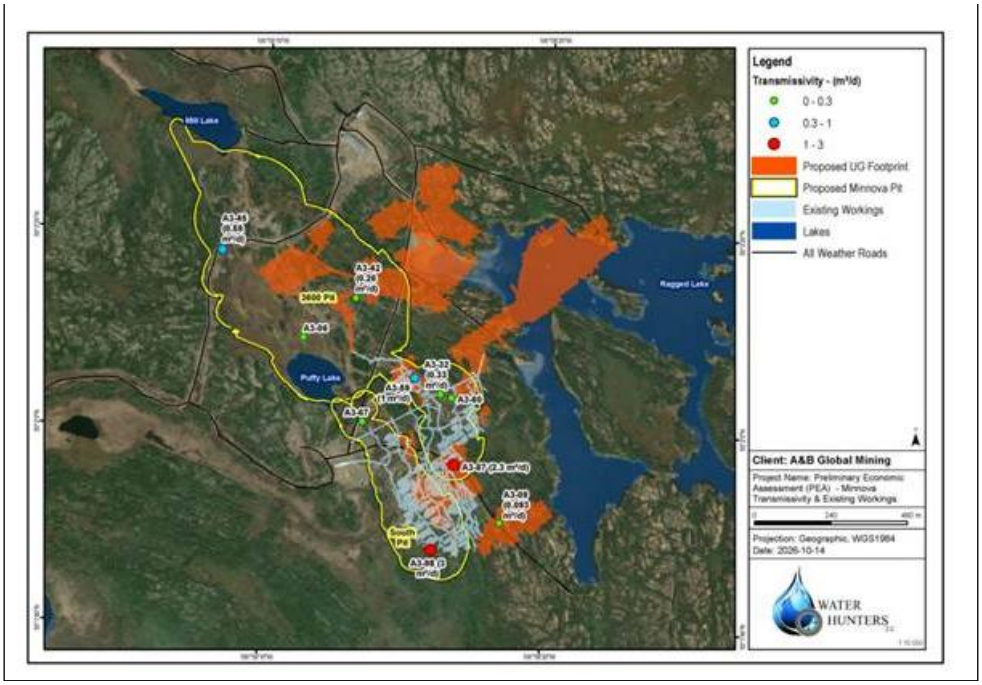


Figure 16-7: Spatial distribution of historical transmissivity observations

Source: Water Hunters, 2026, Figure 3-3

Table 16-14: Selected aquifer-test results

Borehole	Test type	Transmissivity (m²/d)	WESA K (m/d)
A3-06	Slug	n/a	2.6000
A3-09	Constant discharge	0.093	0.0012
A3-32	Constant discharge	0.330	0.0041
A3-42	Constant discharge	0.260	0.0032
A3-45	Constant discharge	0.590	0.0074
A3-59	Constant discharge	1.000	0.0130
A3-60	Slug	n/a	0.8600
A3-67	Slug	n/a	17.0000
A3-87	Constant discharge	2.300	0.0290
A3-98	Constant discharge	3.000	0.0370

Source: Water Hunters, 2026, Table 3-2

The relatively high slug-test conductivity values near historic workings may reflect locally connected fractures or mine-disturbed rock rather than representative regional rock-mass permeability.

16.2.7.2 Steady-state Underground Groundwater Inflows

Water Hunters evaluated historic workings using two analytical approaches. The Goodman tunnel method applied to approximately 13,373 m of historic drifts, ramps and raises, with an average tunnel cross-sectional area of approximately 8.5 m², yielded approximately 757 m³/d (low), 5,820 m³/d (base) and 33,176 m³/d (high). By opening type, the 13,373 m comprises about 6,914 m of level drifts, 6,416 m of ramps and inclines and 43 m of raise (Table 16-17). The method ignores interference between individual openings and is regarded as an upper-bound screening approach, rather than a whole-mine pumping prediction.

A Dupuit single-equivalent-well approximation, incorporating the footprint of the workings, provided a base-case whole-mine steady-state estimate of approximately 200 m³/d (2.3 L/s). The analysis assumes a mine base of 212 masl, initial groundwater elevation of 344 masl, equivalent mine radius of about 150 m and thickness-weighted hydraulic conductivity of 0.007 m/d.

Table 16-15: Dupuit steady-state inflow sensitivity for historic workings

Sensitivity scenario	Inflow (m³/d)	Inflow (L/s)
Base case: K = 0.007 m/d; R = 1,000 m	200	2.3
Low recharge: 10 mm/year	170	1.8
Radius of influence: 500 m	320	3.7
Radius of influence: 2,000 m	150	1.7
High upper-zone K: 0.057 m/d	1,150	13
Low upper-zone K: 0.0013 m/d	26	0.3

Source: Water Hunters, 2026, Table 4-1

The Dupuit sensitivity results show that estimated inflows are substantially more sensitive to hydraulic conductivity than to the assumed radius of influence. Recharge and leakage from adjacent lakes may sustain

inflows if the drawdown cone reaches surface-water features. Neither method accounts for the complete proposed underground mine; additional deep-mine inflow estimates require an updated mine design and site-specific testing.

Table 16-16: Goodman tunnel-inflow calculation by elevation: head and inflow scenarios

Opening type	Band (masl)	Radius (m)	Water table (masl)	Head (m)	Q low (m³/d)	Q base (m³/d)	Q high (m³/d)	Q base (L/s)
Level drift	320 to 340	1.66	341.6	14.1	14	111	630	1.3
Level drift	300 to 320	1.74	342.4	35.5	33	255	1 454	3.0
Level drift	280 to 300	1.66	346.8	51.9	53	405	2 309	4.7
Level drift	260 to 280	1.70	343.7	78.8	43	328	1 872	3.8
Level drift	240 to 260	1.79	345.6	100.9	94	720	4 106	8.3
Level drift	220 to 240	1.73	345.7	114.7	160	1 228	7 001	14.2
Level drift	200 to 220	1.93	344.9	130.8	4	28	158	0.3
Raise	280 to 300	0.44	341.5	43.8	3	22	127	0.3
Ramp/incline	340 to 360	1.53	342.9	0.0	0	0	0	0.0
Ramp/incline	320 to 340	1.49	341.9	12.1	31	238	1 356	2.8
Ramp/incline	300 to 320	1.54	342.1	34.5	88	677	3 860	7.8
Ramp/incline	280 to 300	1.54	345.0	56.8	131	1 010	5 758	11.7
Ramp/incline	260 to 280	1.52	344.5	77.9	45	343	1 952	4.0
Ramp/incline	240 to 260	1.68	345.9	100.0	35	267	1 520	3.1
Ramp/incline	220 to 240	1.49	344.0	110.7	21	160	911	1.8
Ramp/incline	200 to 220	1.85	343.6	128.8	4	29	163	0.3
Total		1.64			757	5 820	33 176	67.4

Source: Water Hunters, 2026, Table 4-2

Table 16-17: Goodman tunnel-inflow calculation by elevation: geometry and dimensions

Opening type	Band (masl)	Length (m)	Midpoint (masl)	Mean ground (masl)	Volume (m³)	Area (m²)
Level drift	320 to 340	353	327.5	342.6	3 074	8.7
Level drift	300 to 320	424	306.9	343.4	4 042	9.5
Level drift	280 to 300	514	294.9	347.8	4 424	8.6
Level drift	260 to 280	1 001	264.9	344.7	9 126	9.1
Level drift	240 to 260	1 789	244.7	346.6	18 057	10.1

Opening type	Band (masl)	Length (m)	Midpoint (masl)	Mean ground (masl)	Volume (m³)	Area (m²)
Level drift	220 to 240	2 778	231.0	346.7	25 986	9.4
Level drift	200 to 220	55	214.1	345.9	646	11.7
Raise	280 to 300	43	297.7	342.5	26	0.6
Ramp/incline	340 to 360	942	347.4	343.9	6 906	7.3
Ramp/incline	320 to 340	873	329.8	342.9	6 058	6.9
Ramp/incline	300 to 320	1 187	307.6	343.1	8 894	7.5
Ramp/incline	280 to 300	1 217	288.2	346.0	9 097	7.5
Ramp/incline	260 to 280	1 080	266.6	345.5	7 840	7.3
Ramp/incline	240 to 260	676	245.9	346.9	6 000	8.9
Ramp/incline	220 to 240	383	233.3	345.0	2 675	7.0
Ramp/incline	200 to 220	58	214.8	344.6	626	10.8
Total		13 373			113 477	8.5

Source: Water Hunters, 2026, Table 4-2

16.2.7.3 Water Quality, Operational Risks and Further Work

Historic groundwater quality generally exhibited near-neutral to slightly alkaline pH and low dissolved solids, but locally elevated metals and metalloids, including arsenic, iron and manganese. The treatment and storage strategy must account for water recovered during initial dewatering and potential changes in water quality following contact with exposed mineralized rock. Kinetic geochemical testing and source-term assessments are required to define long-term water-treatment needs.

The principal hydrogeological risks are unverified deep hydraulic conductivity, conductive shear zones beneath surface-water bodies, transient inflow spikes, high-pressure water intersections during underground development, and potential delays or slope instability from inadequate pressure relief. Analytical estimates assume simplified homogeneous conditions and should not be adopted as final design capacities.

For the pre-feasibility stage, Water Hunters recommends structural and surface-water mapping; site reconnaissance and a hydrocensus; inspection of existing monitoring and exploration holes; targeted packer and longer-duration pumping tests; deeper hydraulic testing; groundwater-quality and geochemical investigations; and development and calibration of a three-dimensional numerical groundwater model that represents seasonal recharge, pit and underground mine geometry, and surface-water connectivity.

16.2.7.4 Indicative PFS Investigation Program and Budget

Water Hunters presents an indicative program for the pre-feasibility phase. The proposal covers site reconnaissance and hydrocensus, structural investigations, supervision and interpretation of aquifer testing, numerical groundwater and geochemical transport modelling, reporting and project management. A site visit is required to confirm the condition of existing drillholes, monitoring wells and core before finalizing the scope. Client responsibilities include sample collection, handling, logistics and laboratory testing costs, which are not included in the quoted consulting budget.

Table 16-18: Indicative PFS hydrogeological work program and cost estimate

Work package	Indicative amount (US$)
Phase 2, Hydrocensus and site reconnaissance	13,767
Structural investigation	40,501
Supervision of fieldwork and data analysis	18,785
Numerical model for dewatering design	41,661
Geochemical transport model	20,830
Feasibility-level hydrogeological report	3,030
Project management (as stated in source)	15,291
TOTAL excluding VAT	153,865
VAT	23,080
TOTAL including VAT	176,944

Source: Water Hunters, 2026, Section 8; ZAR amounts converted to US$ at 0.0853 CAD/ZAR (November 5, 2026) and the PEA exchange rate of US$0.74 per C$1.00 (equivalent to US$0.0631 per R1.00). Each amount is converted directly from the ZAR source value, so totals may not add exactly because of rounding. US$ excluding VAT preferred.

Table 16-19: Detailed indicative PFS work-package rates and quantities

Activity	Rate (US$)	Quantity	Amount (US$)
Site visit (7 days, two people)	170/day	7 days	10,737
Data review	76/hour	40 hours	3,030
Structural mapping	1,302/day	21 days	27,340
Structural reporting	104/hour	90 hours	9,374
Leapfrog model	189/day	20 days	3,787
Fieldwork supervision	76/hour	168 hours	12,725
Packer/falling-head data analysis	76/hour	80 hours	6,060
Conceptual site model	104/hour	80 hours	8,332
Model construction and calibration	104/hour	120 hours	12,498
Dewatering scenario modelling	104/hour	80 hours	8,332
Groundwater sensitivity analysis	104/hour	40 hours	4,166
Groundwater modelling report	104/hour	80 hours	8,332
Contaminant plume modelling	104/hour	80 hours	8,332
Geochemical sensitivity analysis	104/hour	40 hours	4,166
Geochemical reporting	104/hour	40 hours	4,166
Kinetic testing consultation	104/hour	40 hours	4,166
Feasibility hydrogeological report	76/hour	40 hours	3,030

Source: Water Hunters, 2026, Section 8; ZAR amounts converted to US$ at 0.0853 CAD/ZAR (November 5, 2026) and the PEA exchange rate of US$0.74 per C$1.00 (equivalent to US$0.0631 per R1.00).

Note: Line-item rates, quantities and amounts are transcribed from Water Hunters (ZAR) and converted to US$ at 0.0853 CAD/ZAR (November 5, 2026) and US$0.74 per C$1.00. Certain displayed line-item calculations do not reconcile arithmetically; the source proposal totals have therefore been retained without adjustment.

The numerical groundwater model scope includes conceptual model development, model construction and calibration, dewatering scenarios, sensitivity and uncertainty assessment, and reporting. The geochemical scope includes surface-source plume modelling, sensitivity assessment, reporting and consultation on kinetic-test sampling and interpretation. All amounts above reproduce the source proposal totals; the detailed rates and subtotals are indicative and require commercial reconciliation before procurement.

16.2.8 Conclusions and Recommendations for the Pre-Feasibility Study

The rock mass at PL is strong and of good quality, and it supports the proposed hybrid open-pit and underground concept at PEA level, subject to the limitations above. Slope angles, stope spans, crown-pillar thickness and numerical factors of safety remain PEA indications until the following work is completed:

- Separate hanging-wall, mineralized zone and footwall classification from the existing core and from new oriented-core drilling.
- Laboratory testing: UCS (parallel and normal to foliation), triaxial, Brazilian tensile, direct shear on discontinuities, density and durability.
- In-situ stress measurement, and hydrogeological investigation (packer tests, piezometers and a groundwater model).
- Kinematic, limit-equilibrium and numerical slope analyses by domain for the full pit depth, with acceptance criteria to industry guidelines.
- Confirmation of the 17 m level spacing and drift spans with 3D numerical modelling, backfill (including paste-fill) strength testing, and a ground-support design matrix and geotechnical risk register.
- A numerical stability check of the first stopes beneath the pit, and a survey and visual monitoring line on the pit floor before underground production begins under the pit.

16.3 Open Pit Mining

16.3.1 Introduction

The September 2026 open-pit study evaluates the PL deposit through three geographic mining areas: South-East, North-East and North-West. This chapter describes the optimization input criteria, pit configuration, mining adjustments and production profiles. The study uses a gold-price assumption of US$3,000/oz, 90% milling recovery and a 50° pit-slope input.

The monthly schedules indicate adjusted run-of-mine (ROM) inventories of approximately 0.418 Mt at 6.125 g/t Au for South-East, 0.109 Mt at 4.686 g/t Au for North-East and 1.229 Mt at 4.181 g/t Au for North-West.

These inventories and grades are derived from the individual pit monthly summaries. North-West is reported on the 23-period adjusted-summary basis described in Section 16.3.6. The pit periods are local scheduling periods rather than a common project calendar.

16.3.2 Open Pit Optimization

16.3.2.1 Design Approach

The optimization and production work is organized by pit area. The schedule workbook contains NPV Scheduler (NPVS) outputs, including rock movement, mineralized material destinations, metal quantities and monetary value fields. The production summaries apply mining recovery and dilution adjustments to the source mineralized material quantities and express the resulting ROM tonnes, grade and strip ratio on a consistent reporting basis.

The technical and economic criteria comprise the gold price, mining and processing cost inputs, selling costs, royalty, metallurgical recovery, mining loss, dilution and pit slope. The full-cost and incremental cut-off calculations provide separate measures of mineralized material classification economics. The full-cost calculation includes the complete listed plant feed cost, while the incremental calculation deducts the waste-mining cost component.

16.3.2.2 Economic and Technical Inputs

Table 16-20: Economic inputs for pit optimization

Parameter	Value	Input unit
Gold price	3,000.00	US$/oz
Gold price per gram	96.45224	US$/g
Mineralized rock mining	6.70	US$/t mined
Waste rock mining	5.48	US$/t mined
Processing	26.81	US$/t*
Hauling to mill	2.44	US$/t*
General and administration	12.60	US$/t*
Reclamation	0.61	US$/t*
Selling cost	31.68	US$/oz produced
Selling cost per gram	1.01854	US$/g produced
Royalty	3	%

Table 16-21: Mining and recovery inputs

Parameter	Value
Pit slope	50°
Mining loss	10%
Mining recovery used in schedule adjustments	90%
Dilution	20%
Milling recovery	90%

The 50° slope is an optimization input. It is reported separately from bench-face, berm and ramp geometry. Mining recovery and metallurgical recovery have different functions: the former adjusts mineralized material and contained metal delivered by mining, while the latter represents recovery during treatment.

The final geotechnical study (R. Lousteau, 2026) specifies the pit slope geometry for design and scheduling: 10 m benches (5 m only in weathered or structurally complex zones), bench face angles of 70° to 80°, 6.5 m catch berms, stacks of up to 40 m separated by geotechnical berms of at least 10.3 m, and a final overall slope angle of 50°, which is the slope used in the optimization. Ramps are 15 m to 20 m wide at a maximum grade of 10%. A 15 m crown pillar of competent rock is left between the final pit floor and the first underground stope, with no stoping inside that zone.

16.3.2.3 Cut Off Grade

The input workbook calculates net revenue of US$92.54014/g after selling costs and royalty. The full plant feed cost input is US$49.16/t. At 90% milling recovery, the corresponding full-cost cut-off grade is:

Full-cost cut-off grade = 49.16 ÷ (0.90 × 92.54014) = 0.59025 g/t Au.

The incremental plant feed cost is US$43.68/t, calculated as US$49.16/t less US$5.48/t waste-mining cost. The corresponding incremental cut-off grade is:

Incremental cut-off grade = 43.68 ÷ (0.90 × 92.54014) = 0.52446 g/t Au.

These values are economic screening calculations from the September criteria. The production results in this chapter are taken from the scheduled material destinations, rather than generated by reapplying either cut-off to a block model.

16.3.3 Pit Optimization Results and Pit Design

The pit configuration comprises a large north-western excavation and smaller excavations extending southward and eastward along the mineralized system. The production presentation identifies the three scheduled areas using green outlines on the pit plan views. The overview illustrates the variation in excavation depth and the spatial separation of the mining areas.

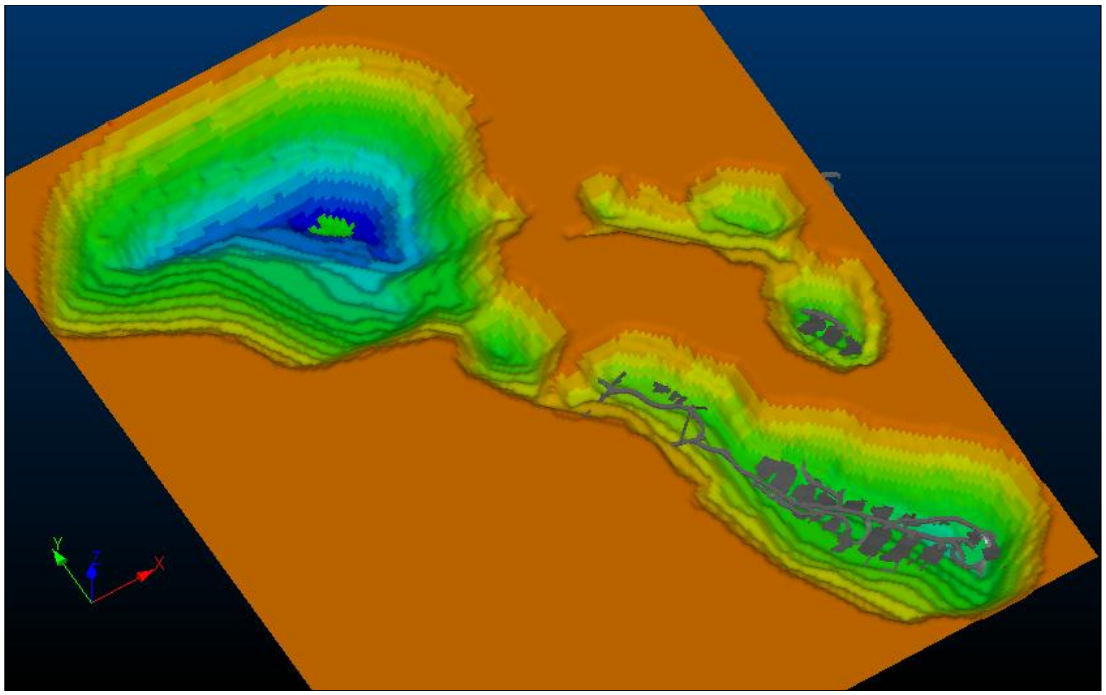


Figure 16-8: Pit configuration overview. Source: ABGM (2026).

The overview slide records local depth annotations between 35 m and 155 m, including 45 m, 55 m, 65 m and 100 m. These are local annotations on the three-dimensional illustration and are not assigned here as maximum depths for the individual schedule areas.

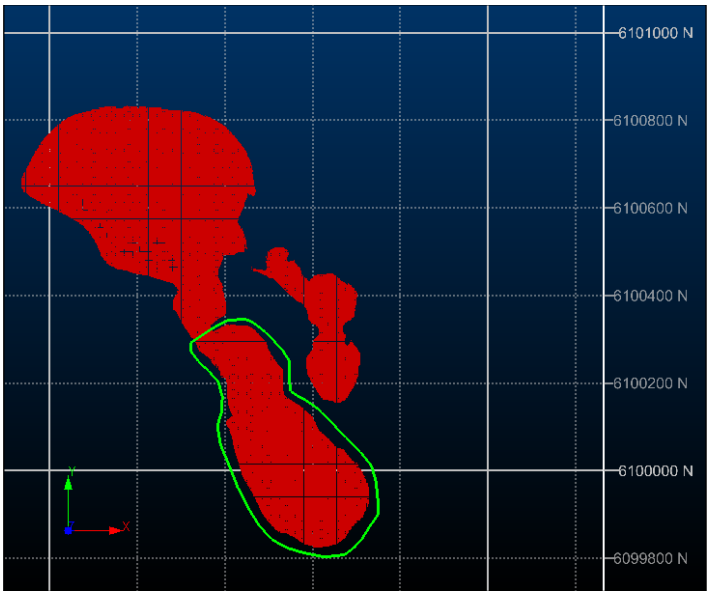


Figure 16-9: South-East pit area, outlined in green. Source: Production Schedules, slide 1.

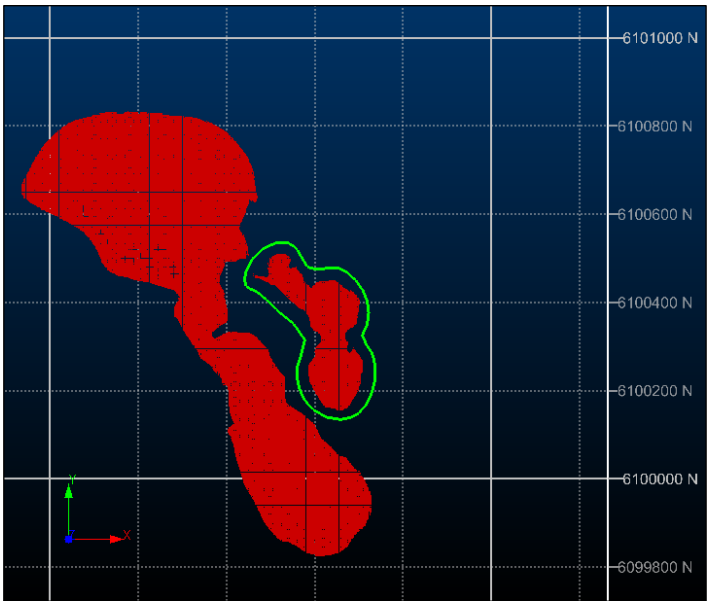


Figure 16-10: North-East pit area, outlined in green. Source: Production Schedules, slide 2.

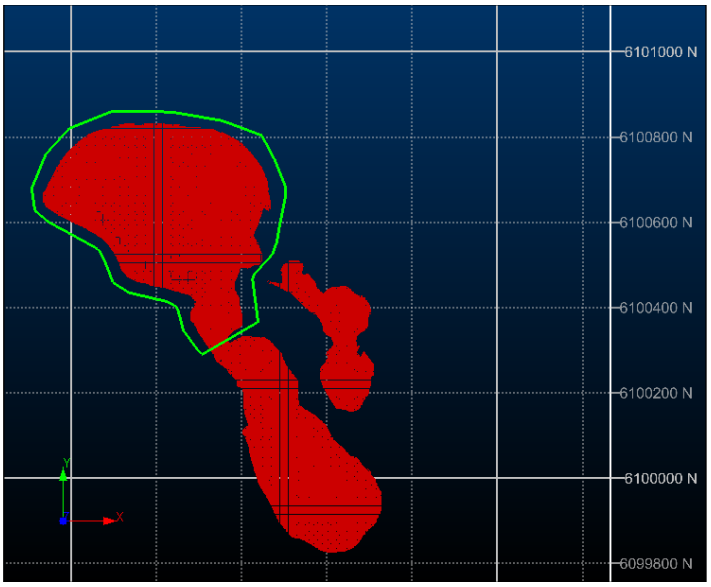


Figure 16-11: North-West pit area, outlined in green. Source: Production Schedules, slide 3.

16.3.4 Mining Sequence and Modifying Factors

Mining is represented through monthly rock movement and mineralized material exposure within each pit. South-East has 11 active periods, North-East has six and North-West has 23. The profiles describe the progression of each mining area independently. They are not summed into a concurrent fleet demand or added together to state a project mine life.

The schedule summaries apply 20% dilution and 90% mining recovery. The adjusted ROM tonnage factor is calculated as:

ROM tonnage factor = 0.90 × (1 + 0.20) = 1.08.

Adjusted ROM tonnes equal source mineralized material tonnes multiplied by 1.08. Adjusted contained gold equals source contained gold multiplied by 0.90. ROM grade is adjusted contained grams divided by adjusted ROM tonnes, giving a dilution grade factor of 1/1.20 for matching source quantities.

Waste in the adjusted summaries is the residual of total rock less adjusted ROM. The reported strip ratio is residual waste tonnes divided by adjusted ROM tonnes. This convention differs from the source scheduler's unadjusted mineralized material and strip fields. Contained ROM ounces are calculated using 31.1034768 g/oz, without a further metallurgical recovery deduction.

16.3.5 Concept Production Schedule and Mineralized Material Profile

Table 16-22: Pit production inventory from monthly schedule summaries

Pit	Active periods	Rock Mt	ROM Mt	Waste Mt	Au g/t	Strip t/t	ROM koz
South-East	11	9.516	0.418	9.097	6.125	21.75	82.37
North-East	6	2.045	0.109	1.935	4.686	17.72	16.46
North-West	23	27.105	1.229	25.876	4.181	21.06	165.17

16.3.5.1 South East Pit

South-East comprises 9.516 Mt total rock movement, including 0.418 Mt adjusted ROM and 9.097 Mt residual waste. The tonnage-weighted ROM grade is 6.125 g/t Au, equivalent to approximately 82.37 koz of contained gold. The overall adjusted strip ratio is 21.75 t waste per tonne ROM.

Rock movement increases from approximately 0.250 Mt in Month 1 to approximately 1.25 Mt/month during Months 7–10. Month 11 is a residual mining period of approximately 0.016 Mt. ROM production increases from 18.85 kt in Month 1 to approximately 50–53 kt/month through much of the middle schedule. Monthly ROM grade varies from 3.695 g/t to 9.964 g/t, with the maximum in Month 8. The grade profile is variable rather than a uniform progression with depth or time.

16.3.5.2 North East Pit

North-East comprises 2.045 Mt total rock, including 0.109 Mt adjusted ROM and 1.935 Mt residual waste. The weighted ROM grade is 4.686 g/t Au, containing approximately 16.46 koz. The adjusted strip ratio is 17.72 t/t.

The schedule begins at approximately 0.250 Mt rock in Month 1, maintains approximately 0.437–0.438 Mt/month through Months 2–5 and concludes with approximately 0.045 Mt in Month 6. ROM production peaks at 33.66 kt in Month 5. The highest monthly grade is 7.174 g/t in Month 1, and the lowest is 3.023 g/t in Month 4.

16.3.5.3 North West Pit

The North-West adjusted summary comprises 27.105 Mt total rock movement over 23 active periods, including 1.229 Mt ROM and 25.876 Mt residual waste. The weighted ROM grade is 4.181 g/t Au, containing approximately 165.17 koz. The strip ratio calculated from the monthly material balance is 21.06 t/t.

Rock movement increases from approximately 0.304 Mt in Month 1 to more than 1 Mt/month from Month 3, before falling to approximately 0.567 Mt in Month 23. The highest monthly rock movement is 1.502 Mt in Month 13. ROM production exceeds 50 kt/month from Month 5 and reaches a maximum of 64.84 kt in Month 20. Grade ranges from 2.844 g/t to 5.157 g/t. The first two months have high strip ratios of approximately 92 and 53 t/t, reflecting limited early mineralized material exposure relative to rock movement.

16.3.5.4 Monthly Production Profiles

Table 16-23: South-East monthly production profile

Month	Rock kt	ROM kt	Waste kt	Au g/t	Strip t/t
1	250.1	18.85	231.3	5.061	12.27
2	437.5	25.47	412.0	4.334	16.18
3	625.0	33.31	591.7	5.133	17.76
4	937.5	37.96	899.6	5.241	23.70
5	1124.9	51.87	1073.0	6.931	20.69
6	1125.3	53.11	1072.2	5.533	20.19
7	1250.0	51.04	1198.9	6.362	23.49
8	1249.8	50.32	1199.4	9.964	23.84
9	1250.2	52.77	1197.5	4.228	22.69
10	1249.8	42.33	1207.5	6.591	28.52
11	15.5	1.23	14.3	3.695	11.60

Table 16-24: North-East monthly production profile

Month	Rock kt	ROM kt	Waste kt	Au g/t	Strip t/t
1	250.2	7.90	242.3	7.174	30.67
2	437.3	20.66	416.6	5.236	20.17
3	437.5	24.13	413.4	5.350	17.13
4	437.6	21.54	416.1	3.023	19.31
5	437.5	33.66	403.8	4.332	12.00
6	44.5	1.33	43.2	5.213	32.48

Table 16-25: North-West monthly production profile

Month	Rock kt	ROM kt	Waste kt	Au g/t	Strip t/t
1	303.8	3.27	300.5	3.158	91.98
2	615.5	11.31	604.2	4.350	53.41
3	1047.9	28.41	1019.5	4.266	35.88
4	1222.1	44.05	1178.0	5.157	26.74
5	1307.1	62.34	1244.8	4.270	19.97
6	1235.6	55.50	1180.1	4.399	21.26
7	1299.9	56.02	1243.9	4.074	22.20
8	1301.2	54.36	1246.9	4.161	22.94
9	1277.1	60.30	1216.8	4.322	20.18
10	1164.2	58.58	1105.6	4.491	18.87
11	1444.5	59.79	1384.7	4.353	23.16
12	1294.1	63.53	1230.6	3.855	19.37
13	1502.4	57.70	1444.7	4.785	25.04
14	1316.5	60.19	1256.3	4.680	20.87
15	1328.9	59.27	1269.6	4.349	21.42
16	1351.6	56.72	1294.8	4.655	22.83
17	1301.7	55.66	1246.1	4.999	22.39
18	1372.9	62.85	1310.1	4.078	20.85
19	1221.2	62.63	1158.6	3.829	18.50
20	1306.3	64.84	1241.4	3.651	19.15
21	1177.9	64.74	1113.1	2.883	17.19
22	1144.9	64.64	1080.3	2.844	16.71
23	567.4	62.11	505.3	4.421	8.14

16.3.5.5 Plant Feed Profile

The base-case plant feed in the PEA cash flow model begins at 27,000 t in Month 3 (March 2028) and increases through 30,000 t, 36,000 t, 45,000 t and 54,000 t in Months 4 to 7, reaching 60,000 t/month (2,000 t/d) from Month 8. This is the total plant feed from all open-pit and underground sources and is distinct from the individual pit mining profiles.

16.3.6 Schedule Reporting Basis

The adjusted monthly summaries form the reporting basis for the production tables. Their monthly quantities and grades also match the chart data in the production presentation. Total rock, ROM, residual waste, weighted grade, strip ratio and contained gold are calculated directly from these rows to maintain an internally consistent material balance.

The North-West total-rock summary contains 9,515,561 t, compared with 27,104,641 t summed from the monthly rows. The summary waste and strip-ratio totals depend on that total-rock figure. The reported North-West inventory therefore uses the monthly sum, giving 25,875,821 t residual waste and a strip ratio of 21.05746. This is an arithmetic correction of the adjusted summary, rather than a revision to the mine design.

The North-West adjusted-summary source mineralized material totals 1,137,796 t, whereas the lower scheduler extract reports 1,227,632 t and contains 25 populated mineralized material periods. The 23-period adjusted summary and lower scheduler extract are separate data representations and are not combined. North-West quantities in this chapter are specific to the adjusted-summary basis and remain subject to reconciliation with the underlying scheduler case.

South-East monthly rock sums to 9,515,564 t compared with 9,515,561 t in its total row. Monthly adjusted ROM sums to 418,280.94 t compared with 418,284 t in that total row. The chapter consistently uses monthly sums. These differences are immaterial at the precision of the reported Mt inventories.

16.3.7 Mining Productivity Basis

The scheduled rock movements establish the pit-level production demand for equipment planning. Peak rock movements are approximately 1.250 Mt/month for South-East, 0.438 Mt/month for North-East and 1.502 Mt/month for North-West. Peak ROM movements are approximately 53.11 kt/month, 33.66 kt/month and 64.84 kt/month respectively.

These rates define material-handling requirements at the individual pit scale. Equipment numbers depend on the operating calendar, overlap between pits, haul profiles, material densities and unit productivity. The separate pit peaks are not added together as a simultaneous mining requirement.

16.3.8 Technical Assessment

North-West contains the largest adjusted ROM inventory and has the longest individual mining profile. South-East has the highest weighted ROM grade, while North-East has the lowest overall adjusted strip ratio of the

three areas. Early North-West production is dominated by waste removal, whereas the South-East schedule develops a higher-grade mineralized material profile during the middle periods.

The production results distinguish mining output from plant feed and preserve the separate timing basis of each pit. The monetary fields in the scheduler exports are not treated as project NPV, and no crown-pillar recovery benefit is included in the reported inventory. The current technical assessment is limited to the September open-pit input criteria, illustrated pit configuration and pit-level production schedules.

16.3.9 Open Pit Production Schedule

The open pit schedule follows the PEA cash flow model. Mining runs for three years, from 2028 to 2030, across the South-East, North-East and North-West pit areas and delivers 1.76 Mt of mineralized material at 4.68 g/t Au and 36.91 Mt of waste, a strip ratio of 21.0. The North-West pit provides 1.23 Mt of the mineralized material. The annual schedule is shown in Table 16-26. The pit inventories in Table 16-22 (0.418 Mt, 0.109 Mt and 1.229 Mt from the South-East, North-East and North-West pits, a total of 1.756 Mt and 264 koz of contained gold, with 36.91 Mt of waste) agree with this schedule. The pit months in Section 16.3.5 are local to each pit. The model starts the North-East pit in Month 1, the South-East pit in Month 4 and the North-West pit in Month 10 (where Month 1 is January 2028), which reproduces the annual mineralized material tonnes in Table 16-26.

Table 16-26: Open pit production schedule (PEA cash flow model)

Item	2028	2029	2030	Total
Mineralized material, South-East pit (kt)	375	44	-	418
Mineralized material, North-East pit (kt)	109	-	-	109
Mineralized material, North-West pit (kt)	43	692	494	1,229
Total mineralized material (kt)	527	735	494	1,756
Mined grade (g/t Au)	5.64	4.52	3.89	4.68
Waste (kt)	11,735	16,224	8,950	36,909
Total rock (kt)	12,262	16,959	9,444	38,665
Strip ratio (t:t)	22.3	22.1	18.1	21.0
Contained gold (koz)	95.5	106.7	61.7	264.0

The mined grade falls from 5.64 g/t Au in 2028 to 4.52 g/t Au in 2029 and 3.89 g/t Au in 2030 as mining moves from the South-East and North-East pits to the North-West pit.

The production strategy aims to maintain a consistent and reliable feed supply to the processing plant while ensuring balanced waste movement across all pushback phases, thereby reducing material rehandling. Additionally, the sequencing of pit development provides early exposure to the southern high-grade zones, enabling timely integration of surface infrastructure and access to the existing underground portal and decline.

16.3.10 Crown Pillar Mining Results and Economic Impact

The crown pillar optimization study evaluated the potential recovery of material beneath the final southern pit shell at a gold price of US$2,500/oz. The analysis compared a “no crown-pillar” scenario with a “crown-pillar mining” case to identify the additional benefit of extracting the near-surface zone before underground development. The two scenarios are illustrated in Figure 16-12 with a summary of the crown pillar mining in Table 16-27.

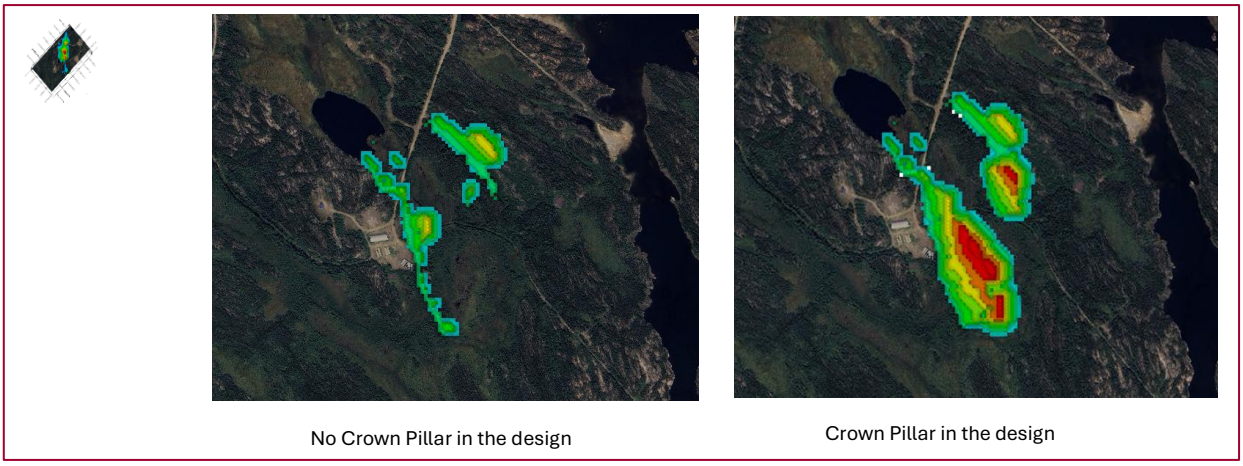


Figure 16-12: Pit comparison between including and excluding the crown pillar

Base Case, No Crown Pillar Mining

The pit without crown-pillar recovery yielded an NPV of US$29.78 million, based on 2.42 Mt of total movement, comprising 0.17 Mt of mineralized material at 5.14 g/t Au and 2.25 Mt of waste, with a strip ratio of 13.3. Contained metal amounted to approximately 430 oz of Au. This scenario maintained full geotechnical isolation between the pit floor and future underground workings but excluded any upper-level mineralized material beneath the pit base.

Crown Pillar Mining Scenario

Including the crown-pillar zone significantly increased the project's value. The model showed an NPV of US$73.46 million, exceeding the baseline. Total mineralized material extracted rose to 8.31 Mt, including 0.46 Mt grading 5.74 g/t Au, containing approximately 71,639 oz of gold. The strip ratio increased slightly to 16.98 t:t due to the larger shell footprint and ramp redesign. The higher value is due to the additional high-grade material immediately beneath the pit floor, which was recovered before the permanent geotechnical crown pillar was constructed to safeguard subsequent underground stoping.

Comparative Assessment

A direct comparison of the two cases shows that crown-pillar extraction adds 0.28 Mt of mineralized material and 48,000 oz of gold, significantly improving overall project economics. The combined shell, which includes both the southern pit and the crown-pillar extension, indicates a total of 41.97 Mt mined, with 1.89 Mt of mineralized material at 3.42 g/t Au, resulting in an in-situ inventory of approximately 264,500 oz of Au.

Crown-pillar recovery is not part of the PEA base case. The final geotechnical study recommends a 15 m crown pillar of competent rock below the final pit floor, with no stoping in that zone, and treats recovery of any high-grade remnant inside the crown as a separate, later study.

Table 16-27: Mining scenario comparison

	NPV (Millions)	Waste Rock (Mt)	Total Mineralized Material (Mt)	Au Content Ounces
No crown pillar mining	29.78	2.42	0.16	23,430
Crown pillar mining	73.46	8.31	0.45	71,639
Variance	+43.68	+5.89	+0.28	+48,209
Percentage differences	147%	244%	174%	206%

16.3.11 Surface Mining Fleet Selection and Productivity Basis

The open-pit mining fleet was selected through a concept-level equipment sizing study to support a steady-state waste movement of 350,000 bank cubic metres (BCM) per month and mineralized material movement of 30,000 t/month. The analysis assumed a material density of 2.8 t/m³, a swell factor of 25%, and two 12-hour shifts operating 29 days per month, resulting in 696 available hours. With 85% availability and 80% utilization, the effective operating time per unit is 473 hours per month.

Primary Loading and Hauling Fleet

The main excavation fleet comprises three CAT 395 hydraulic excavators (5.7 m³ nominal bucket, 5.13 m³ effective capacity, 35 s cycle time) working alongside CAT 740 EJ articulated dump trucks (ADTs) of 23 loose cubic metres (LCM) capacity. The haul system has an average cycle time of about 12 minutes, enabling roughly 82 loading cycles per hour and four passes per truck. A smaller CAT 349 excavator (2.9 m³ bucket) is used for mineralized material selectivity and ramp clearance in tighter work areas.

For the waste-movement target of 739 BCM/h, the calculated loading requirement is 2.25 excavators, resulting in a total of three active units with an average of four trucks per excavator. Including three spares, the fleet totals approximately 15 ADTs dedicated to waste haulage. Mineralized material movement, at approximately 700 m³/month, requires only one loading unit and a single dedicated haul truck to maintain mill feed continuity.

Drilling and Blasting

For cost modelling, drill-and-blast quantities were recalculated based on 5 m benches and the planned monthly material removal (350,000 BCM of waste and 30,000 t of mineralized material). Using a 3.0 m × 3.5 m pattern, 5 m benches, and 10% subdrill, the project requires approximately 40,000 m of 115 mm down-the-hole (DTH) drilling per month, which can be handled by one primary production drill plus one standby/pre-split drill for a two-shift, 29-day month. The final geotechnical study specifies 10 m benches, with 5 m benches only in weathered or structurally complex zones, so the drilling quantities are to be recalculated for the final bench height.

Ancillary and Support Fleet

The ancillary fleet includes:

- 2 × CAT D8 Dozers, ramp and bench maintenance, backfill shaping.
- 2 × CAT 140 Graders, haul-road surface control.
- 2 × CAT 740 Water Carts, dust suppression.
- 1 × CAT 966 Wheel Loader, mineralized material rehandling and stockpile blending.
- 1 × Fuel/Lube Truck and 5 × Lighting Plants, support and night operations.

These units ensure continuous production capability under the defined operating schedule and climatic conditions typical of Manitoba's mining season.

Productivity and Cost Linkage

The selected fleet configuration aligns with the production rates used in the pit-optimization and NPV schedule models. Equipment productivity factors were incorporated into the mining-cost model underpinning the US$6.70/t mineralized material and US$5.48/t waste operating-cost assumptions. The fleet size matches ABGM benchmark data for medium-scale gold open pits with 40 t to 45 t ADTs and 10 m benches, ensuring flexibility for future expansion or integration with underground haulage infrastructure.

16.4 Underground Mining Method

The underground concept was developed to extend mining operations beneath the southern open pit once economic limits are reached at the surface. The aim is to assess feasible underground methods that maintain selectivity, minimize dilution, and optimize recovery within the narrow, moderately dipping orebody of the PL deposit.

Design inputs were derived from the latest geological and geotechnical models. The orebody consists of a single prominent vein that trends approximately north–south and dips 20 to 30° eastward. The true vein thickness ranges from 1.5 m to 2.4 m. The deposit extends to a depth of about 250 m below the surface, with a 15 m crown pillar separating open-pit and underground operations.

Rock-mass quality is characterized by an RMR of 77.79, indicating good ground conditions. Both hanging- and footwall units consist of competent metavolcanics; water inflows are expected to be moderate and manageable. Access to the underground workings is proposed via a single footwall decline developed from the southern pit wall.

16.4.1 Mining Method Selection

The PL deposit is characterized by narrow veins with moderate dips (20° to 30°). These geological conditions complicate mechanization and necessitate careful evaluation of mining methods to balance dilution, production rates, and safety. The previous mining method used at the PL Gold Mine was up-dip stoping from

a central slusher gully; the advantage of this method is minimal dilution; however, it is labour-intensive with poor working conditions and low productivity.

Only mechanized mining methods were considered for this study, since they will provide improved safety and better working conditions than conventional methods; two mechanized methods were proposed for further investigation:

- Drift and Fill mining: this is a common mining method with reasonable production rates; however, there will be considerable dilution, and production rates will be hindered due to the placement and curing of backfill.
- Shallow Dipping Long-hole Open stoping: the average dip of the orebody is between 20° and 30°. This will not allow gravity to clean the stope faces, which is usual in Long-hole Open Stoping. Personnel may not enter the open stopes, so cleaning of the blast material into the bottom drive of each stope will require a remote-controlled dozer. Face lengths of up to 10 m are expected with narrow stoping widths; long-hole blasting is likely to result in slabs of mineralized material that may choke the face and require redevelopment to re-establish a mining face, thereby reducing mining extraction and productivity.

16.4.1.1 Underhand Drift Mining

The final geotechnical study recommends underhand drift mining with progressive backfill of about 30% of the extracted volume, targeted at the larger spans, with primary drifts filled before the adjacent secondary drifts are mined. Mineralized drifts will be developed on strike to a maximum length of approximately 100 m from stope access drives. Subsequent drifts will be excavated below the existing drifts in a top-down process, leaving the supported void above.

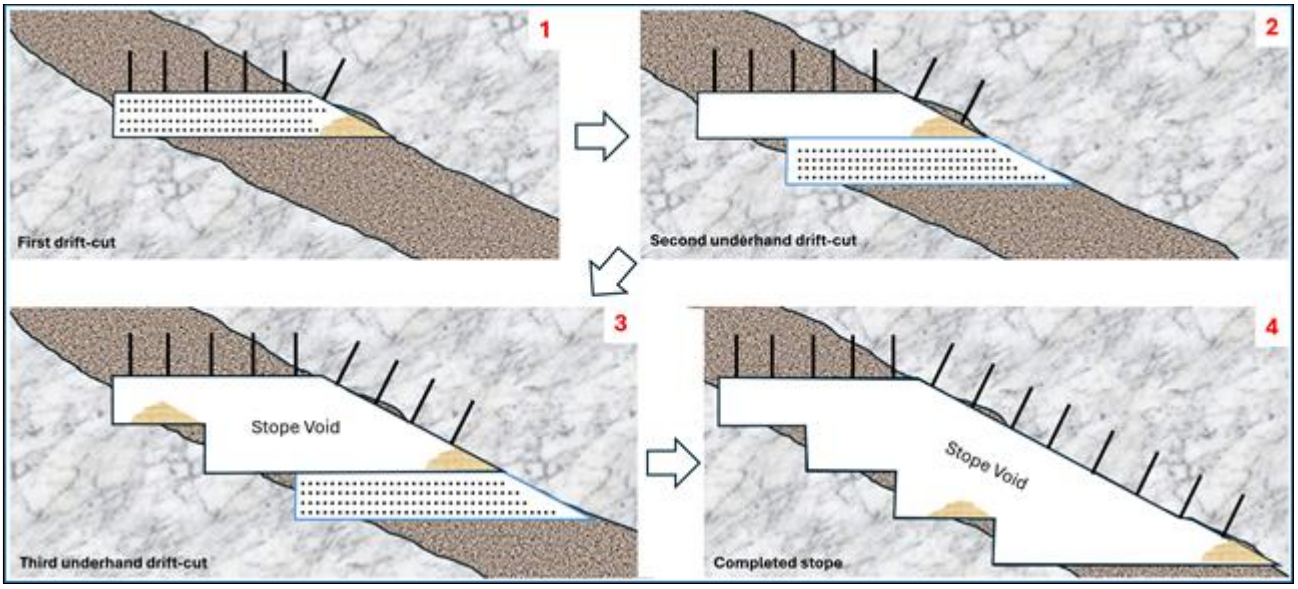


Figure 16-13: Vertical section looking in the direction of strike showing the underhand drift mining sequence

Sill pillars are strategically included in the mine design to ensure regional stability. While their implementation reduces the overall mineable resource, the resulting subdivision of the mining sequence enables the establishment of additional production faces beneath each pillar, thereby improving productivity and scheduling flexibility. The final geotechnical study recommends sill pillars of 4 m to 8 m between levels at the planned depths.

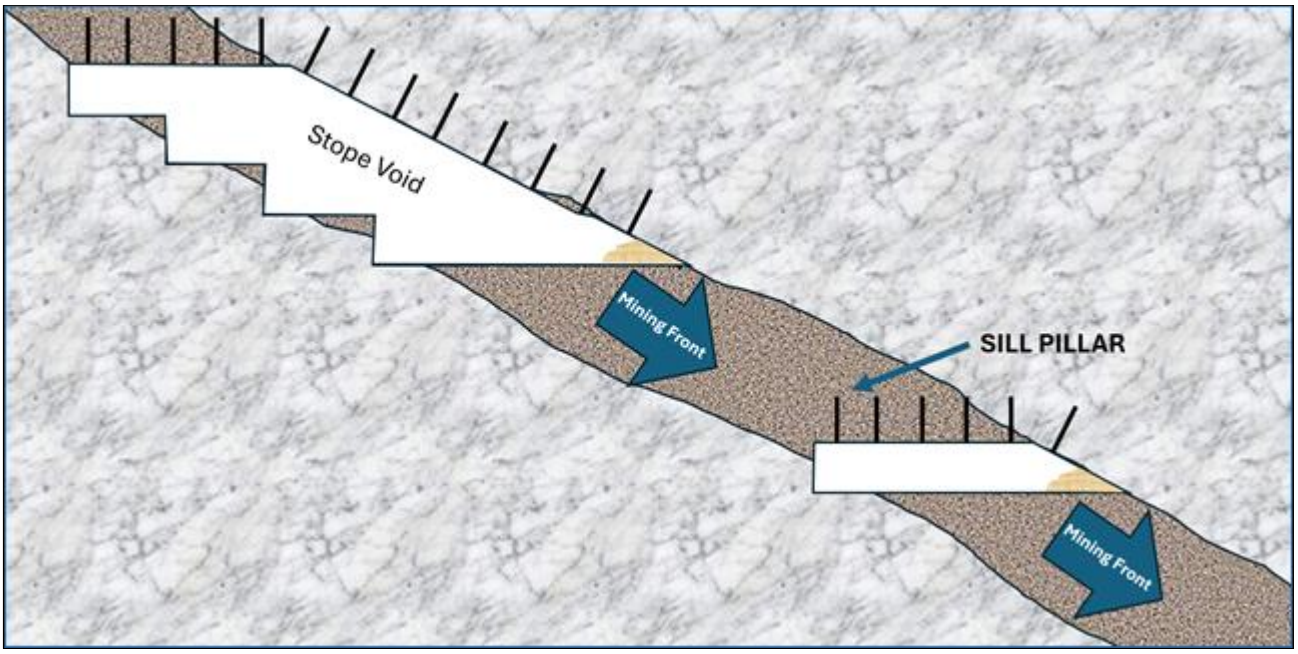


Figure 16-14: Schematic showing the relationship between sill pillars and mining fronts

16.4.2 Access Development

Although existing underground excavations are present at the PL Gold Mine, essential declines and access routes are within the planned open-pit shell; therefore, PEA underground access is through a new portal in the north-east pit (373,663.6 E, 6,100,247.1 N, 292.0 m RL) and a new decline system. The decline system will access the orebody from the hanging-wall side of the veins. The underground mining study proposes a trial mine of about six months through the existing portal and decline before the South-East pit reaches the historical portal area. This trial mine is a PFS recommendation and is not included in the PEA mine schedule or cash flow model, in which South-East pit mining starts in April 2028, underground development starts in January 2029 and underground mineralized material production starts in March 2029. The QPs recommend that the timing, scope and cost of the trial mine, and its sequencing with the South-East pit, be assessed in the PFS.

16.4.2.1 Declines and Ramp Systems

Three mining zones are planned (North, Central and South), each with a dedicated ramp system providing autonomous mining areas. Zones will be connected at regular intervals to provide a second means of egress from all areas of the mine.

Production levels are planned at 17 m vertical intervals.

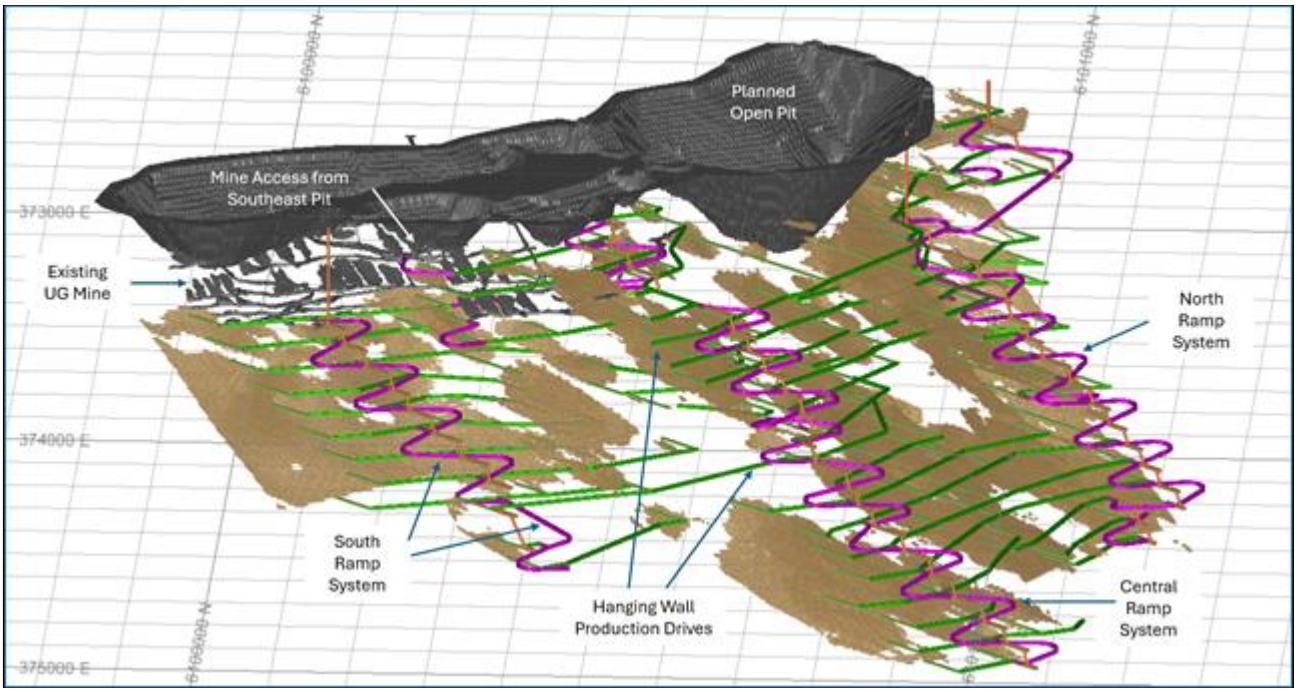


Figure 16-15: Isometric view looking West showing the underground access waste development.

16.4.2.2 Stope Access

Depending on the drift strike length in each zone stopes will be accessed via x/cuts either directly from the ramp system or from production drives developed in the hanging wall.

16.4.3 Break-even Grade Calculation

The mining cost was determined by applying recent mining contractor prices to a generic mine design and benchmarking of similar operations. The break-even cost per run-of-mine (ROM) tonne comprises US$80.98/t for production mining (drift) and US$41.16/t for mining development, with US$26.81/t for the treatment plant and US$12.60/t for general and administration, a total of US$161.54/t per ROM tonne (Table 16-28).

A break-even grade (BEG) of 1.92 g/t is achieved using the factors and costs shown in Table 16-28.

The cash flow model carries contractor rates per tonne of each material (US$82.05/t of underground mineralized material and US$72.44/t of underground waste development, excluding diesel), which is a different basis from the per-ROM-tonne break-even cost.

Table 16-28: Break-even Grade calculation

Factors	
Process Recovery	*90%*
Geological Recovery	*97%*
Unplanned Dilution	*0%*
Au Price (US$/oz)	***3,000***

Factors		
Royalties	*3%*	
Operating costs per tonne treated	**US$/t**	
Production Mining - Drift	*80.98*	
Mining development	*41.16*	
Treatment plant	*26.81*	
General & Admin	*12.60*	
Total cost per RoM Tonne	**161.54**	
Total cost per In-situ Tonne	**156.70**	
Au cut-off grade		
Au price (US$/g)	96.45	
Net Au price received (US$ per in-situ g)	81.68	
Break-even in-situ Au grade	**1.92**	**g/t**

16.4.4 Stope Shape Determination

Mining Shape Optimiser (MSO) was used to determine the optimized stope shapes using the BEG of 1.92 g/t as the cut-off grade.

Several scenarios were evaluated to determine the optimum drift height and sidewall shapes to minimize dilution and maximize gold content. The optimum design parameters are:

- Drift height: 2.5 m
- Minimum dip angle of sidewall (mineralized zone footwall): 45°
- Minimum dip angle of sidewall (mineralized zone hanging wall): 35° (shanty back)
- Minimum drift width (perpendicular to strike): 5.5 m (determined by equipment size)
- Maximum drift width (perpendicular to strike): 11 m
- Strike length: 2.5 m

Stope shapes were generated by MSO using a strike length of 2.5 m which is equivalent to the average planned face advance for each blast in mineralized drifts, this will minimize dilution and maximize mineralized material extraction.

The final geotechnical study limits primary drifts to 5 m to 8 m wide. Spans of 9 m to 11 m are acceptable only where RQD exceeds 80% and RMR exceeds 78, and only with enhanced cable bolting, fibre-reinforced

shotcrete and monitoring; drifts wider than 11 m are not recommended. MSO shapes wider than 8 m are to be checked against these limits.

16.4.5 Economic and Marginal Cut-off Grades

The BEG of 1.92 g/t was also used for the economic cut-off grade (CoG), because the MSO design parameters ensure that the in situ mining grade of each stope is maximized above the CoG. An average in situ grade of 3.53 g/t is achieved from mineralized drifts at this CoG.

Marginal drifts were also considered to remove gaps in the mining footprint and to maximize resource extraction. The marginal CoG is 1.43 g/t, which was derived by excluding the cost of waste development and sustaining Capex from the BEG calculation. It is assumed that marginal drifts will not require any additional waste development, and the cost of this development is already accounted for by stopes above the Economic CoG.

16.4.6 Modifying Factors

The following modifying factors were applied to determine the Run-of-Mine tonnage and grade.

- Geological loss 3%
- Planned dilution About 35% (in the MSO stope designs)
- Unplanned dilution* 0%
- Mining recovery (including geotechnical pillars) 83%
- Mine call factor 100%
- Tonnage discrepancy 0%

** Unplanned dilution is set at 0% because it is expected that the current 35% planned dilution in the MSO stope shape designs can be improved on during mining operations.*

16.4.7 Run of Mine Tonnage and Grade

The PEA cash flow model schedules 4.74 Mt of underground mineralized material at 2.80 g/t Au, containing 426,577 oz of gold, and 3.42 Mt of development waste from 2029 to 2037. Development comprises 150 m of decline, 32,902 m and 41,076 m of development labelled LED and LPD in the model, and 129 m of ventilation raising. The split of these tonnes by Mineral Resource category, and the dilution and mining recovery they include, are to be confirmed from the mine plan.

16.4.8 Rock Handling Philosophy

A fully mechanized methodology is proposed, using low profile trackless mining equipment for drift mining and stope access development, and standard trackless mining equipment for access declines and ramp

development. All the proposed mining equipment is diesel powered; however electrical options can be considered and evaluated at the next study phase.

Face drilling will utilize electro-hydraulic drill rigs; twin boom rigs in large-end development (LED) such as declines, ramps and hanging wall production drives. Drifts will be drilled using single boom low-profile rigs. Ground support will be installed using bolters and drill rigs when necessary.

Muck piles in blasted faces will be loaded into articulated dump-trucks in conveniently placed loading bays in the stope access x/cuts. The ADTs will then haul the mineralized material and waste rock to the respective stockpile or dump on surface.

16.4.9 Trial Mining

A trial mine area is proposed to confirm the extraction methods, validate geological models, and assess economic viability before committing to full-scale operations. It will reduce risk, improve safety planning, and ensure that grade assumptions hold true in real-world conditions. This trial mine is a PFS recommendation and is not included in the PEA mine schedule or cash flow model.

The proposed trial mine would be accessed from the existing underground infrastructure, and stoping would occur in the most southerly portion where gold grades are expected to be higher than average.

16.5 Production Schedule

The production schedule follows the PEA cash flow model. Open pit mineralized material is mined from 2028 to 2030 and underground mineralized material from 2029 to 2037. Mineralized material in excess of plant capacity is stockpiled and reclaimed, and the final year treats the remaining stockpile and underground mineralized material. The annual schedule is shown in Table 16-29.

Table 16-29: LOM production schedule (PEA cash flow model)

Year	OP mineralized material (kt)	OP grade (g/t)	OP waste (kt)	UG mineralized material (kt)	UG grade (g/t)	UG waste (kt)	Plant feed (kt)	Head grade (g/t)	Gold recovered (koz)
2028	527	5.64	11,735	-	-	-	492	5.71	73.9
2029	735	4.52	16,224	27	1.94	49	720	4.39	91.6
2030	494	3.89	8,950	195	2.92	299	720	3.71	79.4
2031	-	-	-	707	2.76	692	720	2.80	58.8
2032	-	-	-	728	3.05	495	720	3.05	61.4
2033	-	-	-	731	3.28	465	720	3.28	70.2
2034	-	-	-	729	2.46	462	720	2.46	52.2
2035	-	-	-	727	2.50	462	720	2.51	51.9
2036	-	-	-	729	2.79	437	720	2.80	57.9
2037	-	-	-	169	2.54	56	247	2.73	24.3
Total	**1,756**	**4.68**	**36,909**	**4,743**	**2.80**	**3,415**	**6,499**	**3.30**	**621.5**

Source: ABGM cash flow model (2026).

The 2028 plant head grade is higher than the open-pit mined grade because mineralized material in excess of plant capacity is stockpiled. The 34.9 kt remaining at year end grades 4.69 g/t, so the 492 kt treated grades 5.71 g/t against 527 kt mined at 5.64 g/t.

16.6 Equipment Fleet

16.6.1 Underground Mining Fleet

Developing drifts at 2.5 m high will require low-profile equipment (drill rigs, load-haul-dump units (LHDs) and utility vehicles (UVs)); however, waste access development (declines, ramps and production drives) is planned at 4.5mW x 4.5mH to provide sufficient ventilation. The access development can be excavated using normal sized drill rigs and ADTs. The LHD / ADT interface is critical, and it is proposed to utilize large capacity low-profile ADTs, this will eliminate the necessity of using two sizes of LHD and ADTs, one for low-profile headings and another size for Large headings.

The following fleet is proposed to support production rates of 2,000 t/d at steady state; Sandvik equipment has been suggested, but equivalent items from other manufacturers are available.

Table 16-30: Proposed underground mining fleet

Type	Size/Model	Number	Width	Height
Drill Rig	DD422 Twin Boom	2	2.3	3.2
Drill Rig	DD211L Low-profile single boom	12	2.3	1.9
Bolter	DS221L Cable Bolter	3		
LHD	LH209L: 9 Tonne	3	3.3	1.7
ADT	TH430L: 30 Tonne	3	3.6	2.8

16.7 Ventilation

The ventilation design is based on the Umoya Occupational Hygiene & Laboratory Solutions VENTSIM LOM study, which simulated the life-of-mine underground layout in VENTSIM Visual version 6.0.5.4 to determine the maximum main fan duty and a standby fan duty. The model represents peak ventilation and heating demand, at peak production and maximum depth.

16.7.1 Ventilation System and Layout

The mine is ventilated by a forcing system with four shafts: three downcast ventilation shafts (South, Central and North), each with its own main fan, and one upcast shaft. A forcing system was selected because of the large range in surface temperatures and the extreme cold, as it allows heating to be added to the downcast air without obstructing the mine access roads. The South, Central and North districts are interlinked and share airways and airflow.

Fresh air is forced down to the lowest level through a series of 3 m diameter vertical shafts connected to each level. When a new level is mined and connected to the downcast, the level above is closed off with a man-door access. Workings are ventilated from the downcast connection in tight circuit, using auxiliary fans and ducting. Each level has man-door access to the fresh-air downcast, which provides refuge on every level in the event of a fire or an irrespirable or toxic atmosphere. The layout is shown in Figure 16-16.

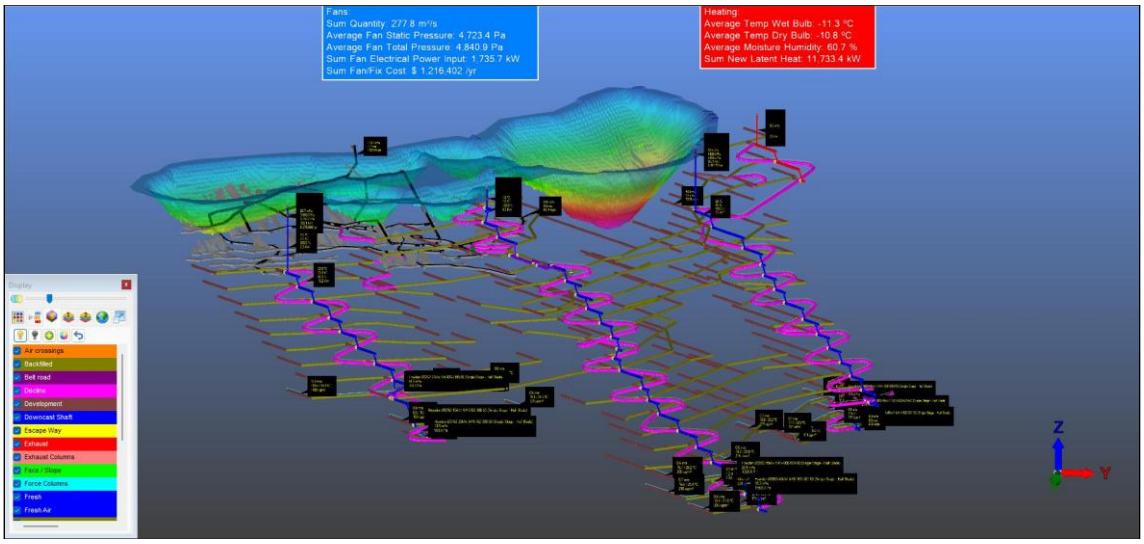


Figure 16-16: VENTSIM LOM model layout with main fan and heating data. Source: Umoya Occupational Hygiene & Laboratory Solutions VENTSIM LOM study.

The main design criteria used in the simulation are summarized in Table 16-31.

Table 16-31: Ventilation design criteria

Parameter	Design value
Development airways	4 m × 4 m, arched, rough blasted
Haulage airways	5 m × 5 m, arched, average blasted
Upcast shaft	4.5 m diameter
Downcast shaft velocity (equipped)	10 to 12 m/s
Maximum upcast shaft velocity (unequipped)	24 m/s
Maximum intake and return airway velocity (personnel)	6 to 7 m/s
Maximum dedicated return airway velocity	12 to 14 m/s
Leakage: surface / primary / secondary	5% / 10% / 15%
In-stope air utilization factor	75%
Development face quantity: force / exhaust (minimum)	0.25 / 0.30 m³/s per m² of face
Maximum duct distance from face: force / exhaust	15 m / 20 m
Diesel exhaust dilution	0.06 m³/s per kW rated power
Re-entry after blasting	Minimum of 8 air changes

Source: Umoya Occupational Hygiene & Laboratory Solutions VENTSIM LOM study, design parameter tables.

16.7.2 Main Fan Duties

The main fan duties at peak demand are shown in Table 16-32. The study recommends that the main and standby fans be ordered with variable speed drives (VSDs), which it estimates have a cost payback of about three to four years. A separate standby fan duty is not stated.

Table 16-32: Main fan duties at peak demand

Shaft	Airflow (m^3/s)	Total pressure at collar (Pa)	Electrical input power (kW)
South	82.6	3,750	395
Central	102.7	4,965	654
North	92.3	5,805	688
Total	**277**	**4,840 (average)**	**1,735**

Source: Umoya Occupational Hygiene & Laboratory Solutions VENTSIM LOM study, executive summary.

Auxiliary ventilation in the simulation uses a single 1.02 m diameter duct, which can serve more than one end through a Y-piece and a larger fan. Auxiliary fans range from 15 kW for short distances to 45 kW or 55 kW for longer distances or multiple ends. The study notes that this selection is not prescriptive and was made to estimate auxiliary costs.

16.7.3 Mine-Air Heating

Each downcast shaft requires separate mine-air heating. The peak heating demand is 3,500 kW at the South shaft, 4,200 kW at the Central shaft and 4,000 kW at the North shaft. The type of heating has not been specified, so the heater efficiency and the final input duty have not been calculated. Heating requirements and costs are to be determined at the next study stage.

16.7.4 Simulation Results

Heat. Temperatures remain stable throughout the mine at about 20 °C. Some areas are warmer because of vehicle activity; these hot spots are not constant and can be managed administratively.

Carbon monoxide. Carbon monoxide (CO) from diesel exhaust remains manageable at about 20 ppm in the declines. Only an increase in the diesel fleet would raise this level.

Diesel particulate matter. Diesel particulate matter (DPM) is a concern: concentrations in the declines are close to 500 µg/m³. This is the only disadvantage of the forced downcast system. The study recommends diesel particulate filtration, which has proven itself at several operations, and notes that retaining the warm diesel exhaust air in the accessible workings saves heating cost, which partly offsets the cost of the filters.

16.7.5 Conclusions

In the simulation, two or three levels per shaft were allocated a portion of the diesel fleet using an activity track, and development or production ends on the bottom three levels were ventilated with force fans and

flexible ducting. The fan duties above are the recommended design duties. The study states that the mining and ventilation layouts it describes should be followed, as deviations will give different results.

16.8 Dewatering

The dewatering assessment is based on the Water Hunters (October 19, 2026) hydrogeological desktop study and gap analysis. It provides preliminary planning estimates for the existing flooded workings and proposed open pit; it does not establish a final pumping-system design. Inflows to the deeper proposed underground workings remain unquantified pending further investigations.

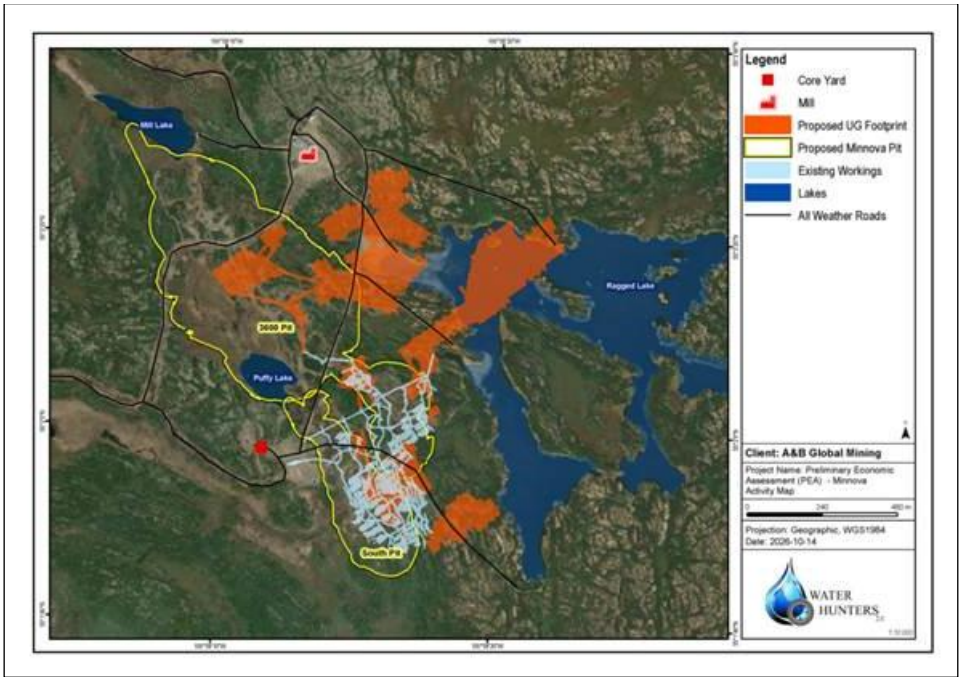


Figure 16-17: Mine activity map showing proposed pit, underground footprint, existing workings and surface-water features

Source: Water Hunters, 2026, Figure 2-1

16.8.1 Initial Dewatering of Historic Workings

A three-dimensional mesh of the historic PL Gold Mine workings produced a calculated void volume of 168,479 m³. Allowing for historic backfill and possible collapse, the actual flooded void volume was estimated at approximately 120,000 to 160,000 m³. Water Hunters indicated a preliminary pumping period of approximately two to six months at 10 to 30 L/s, excluding ongoing groundwater inflows and additional water released from fractured-rock storage. These are indicative estimates, not a confirmed pump duty or program.

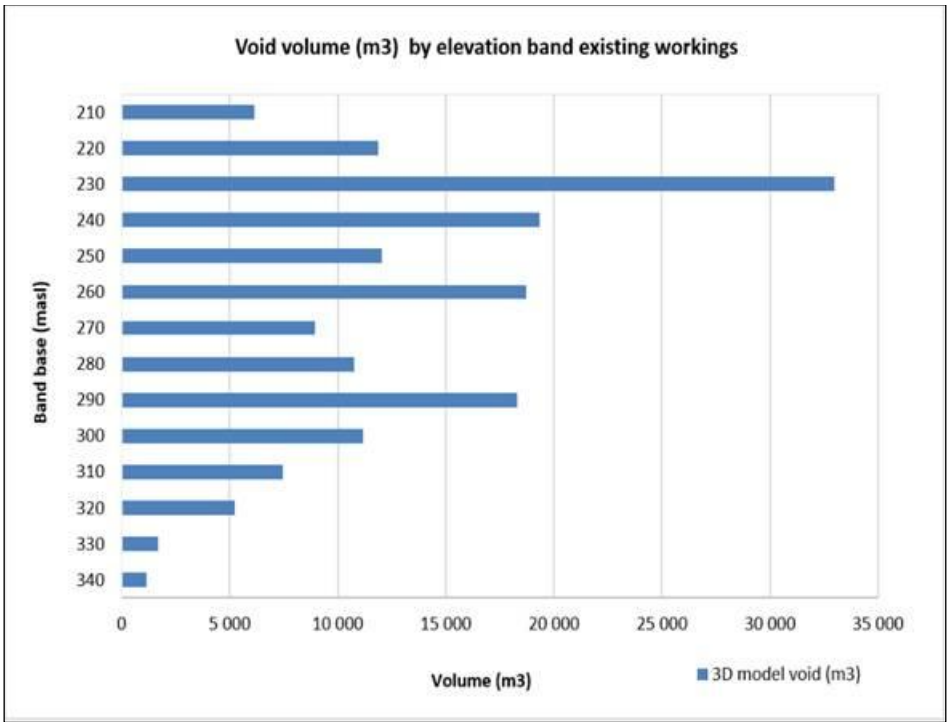


Figure 16-18: Calculated distribution of historic underground void volume by elevation

Source: Water Hunters, 2026, Figure 4-1

Dewatering of the existing workings is required to access the southern stopes through the historic portal. The proposed connection between old and new workings means that the historic workings would need to remain dewatered throughout underground operations.

16.8.2 Open Pit Dewatering and Mine Interaction

The proposed pit covers approximately 41 ha and extends to approximately 170 masl, about 40 m below the base of the historic workings. The pit overlaps flooded historic workings: approximately 4% of their void volume (around 6,400 m³) lies within the pit footprint, while approximately 29% is directly beneath the pit walls and benches. This overlap requires coordinated pit/underground dewatering and assessment of seepage and crown-pillar stability.

Preliminary Marinelli–Niccoli pit-inflow estimates are approximately 160 m³/d (low conductivity), 650 m³/d (base case) and 2,400 m³/d (high conductivity). A hydraulically connected lake in the high-conductivity case could increase inflows to approximately 3,500 to 6,100 m³/d (40 to 70 L/s). Direct precipitation over the 41 ha pit contributes an estimated average of approximately 540 m³/d, with higher seasonal peaks during snowmelt.

Table 16-33: Preliminary pit inflow scenarios

Scenario	Estimated inflow (m^3/d)	Comment
Low conductivity	160	Groundwater inflow
Base case	650	Groundwater inflow
High conductivity	2,400	Groundwater inflow
Lake-connected high case	3,500 to 6,100	Potential sustained lake leakage
Direct precipitation	~540	Annual-average contribution; seasonal peaks higher

Source: Water Hunters, 2026, Section 4.3

The base-case radius of influence was estimated at approximately 290 to 440 m, comparable with the distance to nearby surface-water bodies. Groundwater discharge is expected to be concentrated along conductive structures rather than evenly distributed around pit walls. Frozen pit faces can trap seepage and increase pore pressure; horizontal drains and spring-freshet pumping allowances require evaluation.

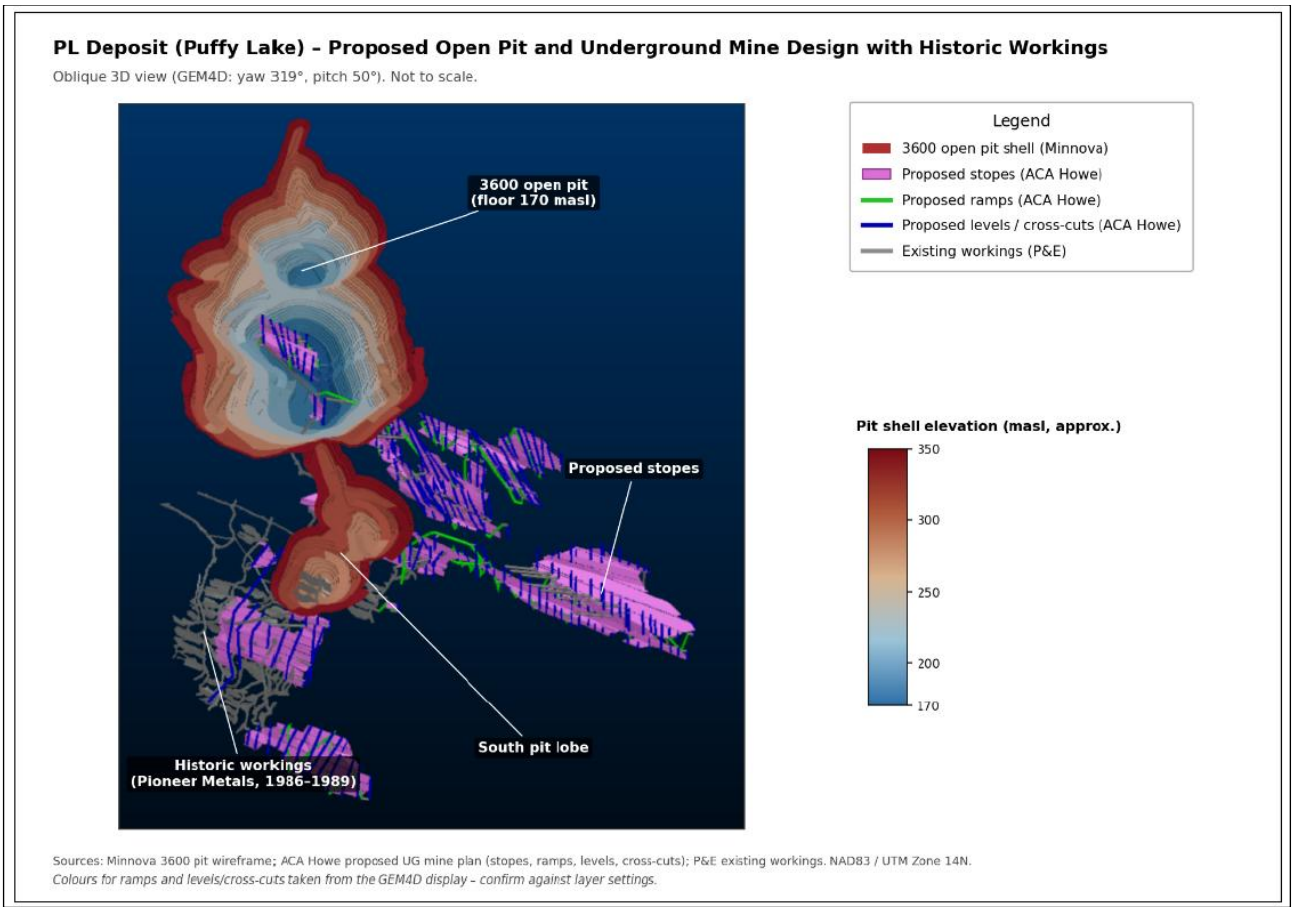


Figure 16-19: Proposed pit, flooded historic workings and planned underground development

Source: Water Hunters, 2026, Figure 4-2

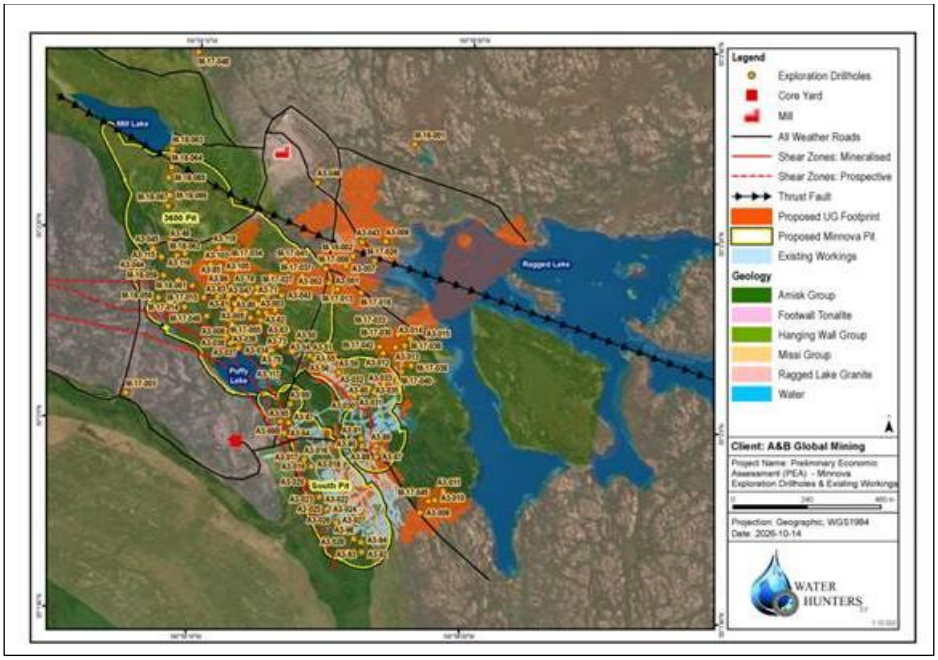


Figure 16-20: Exploration drillhole distribution and existing workings

Source: Water Hunters, 2026, Figure 5-1

The mapped shear zones and thrust structures intersect or approach both the proposed mine footprint and adjacent water bodies. The map evidence is relevant to the assessment of potential preferential groundwater pathways; the available desktop data do not establish hydraulic connectivity at depth.

Source: Water Hunters Pty Ltd (2026), Minnova: Hydrogeological Desktop Study and Gap Analysis, draft version 1.0 dated October 19, 2026, Sections 2 to 6. Estimates are preliminary and subject to verification during subsequent study phases.

16.9 Backfill

The final geotechnical study recommends progressive backfill, placed as mining advances, of about 30% of the extracted volume, targeted at the larger spans, with primary drifts filled before the adjacent secondary drifts are mined. Hydraulic sand fill or development waste is the PEA fill basis, with cemented fill in critical stopes. Regional numerical modelling shows that this reduces maximum stope closure from about 90 mm to less than 15 mm and peak major principal stress from about 73 MPa to 80 MPa to about 25 MPa to 45 MPa. The fill sources, preparation and placement system have not yet been designed. The PEA cash flow model carries no backfill capital or operating cost. The underground mining method requires further trade-off work at the PFS stage, which will include options that require no backfill or only partial backfill, and backfill costs will be estimated for the selected method.

16.10 Workforce

The PEA cash flow model costs the workforce through its cost lines rather than through a headcount. Mining labour is included in the contractor mining rates (US$5.48/t of open-pit waste, US$6.70/t of open-pit mineralized material, US$72.44/t of underground waste development and US$82.05/t of underground

mineralized material), and the LOM contractor mining cost is US$204.6M for the open pit and US$636.5M for the underground. Processing labour is included in the processing cost of US$28/t milled, which also covers plant power (US$182.8M over the LOM). Owner management and site staff are costed under management and general and administration (G&A): “Owner Costs - Senior Management” of US$1.5M per year and “Owner Costs - Jnr Management and Staff” of US$1.5M per year (US$14.75M each over the LOM), and the “G&A Cost” of US$10.60/t of ROM mineralized material, from which the two owner management lines are deducted so that they are counted once (US$40.76M over the LOM). The model reports these together as “Site G&A and owner management”, US$70.26M over the LOM, or about US$7.7M per year at full production.

Accommodation is costed in the “Personnel Lodge” lines at US$90 per person per day (catering and food service US$50, housekeeping and janitorial US$15, utilities US$15, and management and administration US$10), for an occupancy of 120 people from Month 4 (50, 80 and 100 people in Months 1 to 3), a total of US$38.41M over the LOM. The mining contractor expects up to about 60 pit contractor staff on site at any time (AECOM, draft 2026).

16.11 Life of Mine (LOM) Plan

The LOM plan in the PEA cash flow model is summarized in Table 16-34.

Table 16-34: LOM plan summary (PEA cash flow model)

Item	Value
Open pit mining period	2028 to 2030
Open pit mineralized material	1.76 Mt at 4.68 g/t Au
Open pit waste	36.91 Mt (strip ratio 21.0)
Open pit contained gold	263,992 oz
Underground mining period	2029 to 2037
Underground mineralized material	4.74 Mt at 2.80 g/t Au
Underground development waste	3.42 Mt
Underground contained gold	426,577 oz
Plant feed	6.50 Mt at 3.30 g/t Au
Contained gold in plant feed	690,570 oz
Metallurgical recovery	90%
Gold recovered	621,513 oz
Processing period	2028 to 2037

Source: ABGM cash flow model (2026).

17 RECOVERY METHODS

The PL Mine Gold Plant was constructed during 1986/87 and was commissioned with first gold production in late 1987. Ore treatment carried on until March 1989 with final gold production from inventory and plant 'clean-up' in May 1989 as detailed in Table 13-2.

17.1 Process Description

Historical testwork indicates that PL mineralization is non-refractory to cyanidation following fine grinding. Coarse gold is gravity-recoverable and finer sulphide-associated gold is amenable to flotation, regrinding and cyanide leaching; no sulphide oxidation pretreatment is required. There is no significant organic carbon content in the mineralized material and thus no preg-robbing component.

The installed process employed consists of two stage crushing to <15 mm, with grinding to <100 µm in a rod mill/ ball mill configuration incorporating hydro-cyclone classification. The cyclone underflow was directed to a two compartment Jig to recover the gravity gold. A Spiral plant was installed on the hydro-cyclone overflow but was discontinued and removed in 1988/89. The gravity recovery achieved is reported to be around 45%.

The correctly sized hydro-cyclone overflow is delivered to conditioning tanks ahead of the flotation plant. The flotation plant consists of a bank of conventional Outotec OK8 rougher and rougher-scavenger cells with the tailings being sampled for discharge to the Ragged Lake tailings facility. The scavenger concentrate is recycled to the flotation plant feed with the rougher concentrate being delivered to the concentrate thickener with the gravity concentrate.

The thickener underflow is pumped to hydro-cyclones with the coarse fraction delivered to the regrind mill and the combined finer fractions being pumped to the four-stage cyanide leaching circuit with a nominal residence time of 72 hours. The product from the leach is filtered in two stages with the cake from the second stage being repulped with detoxified barren solution and sampled and mixed with the flotation plant tails for discard to Ragged Lake. The filtrate from the rotary filters is clarified and stored ahead of the zinc cementation Merrill-Crowe gold recovery circuit. The cementation product is filtered in a filter press prior to being discharged into the gold room.

The gold room contained a calcining oven and a furnace to smelt the doré for dispatch to a refinery.

17.2 Process Flowsheet

17.2.1 Original 1987 Process Flowsheet

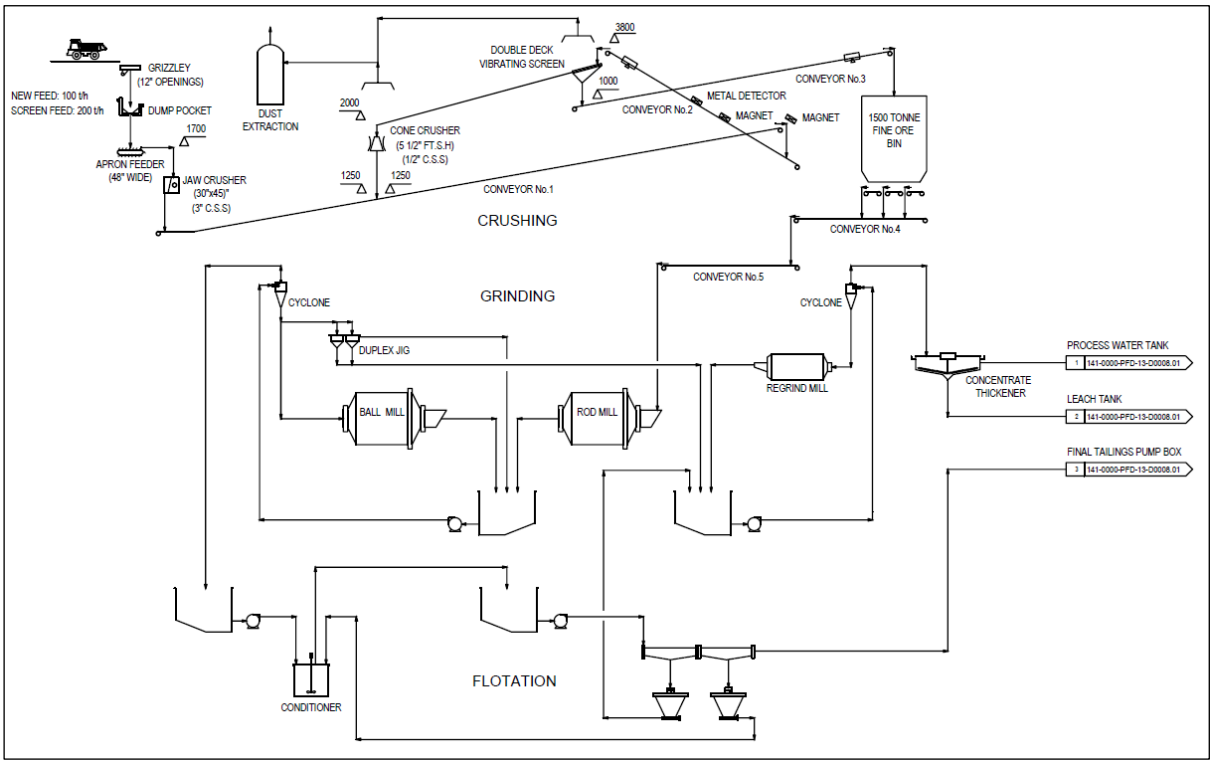


Figure 17-1: Historical Crushing, Milling and Flotation Circuit: 1,000 t/d

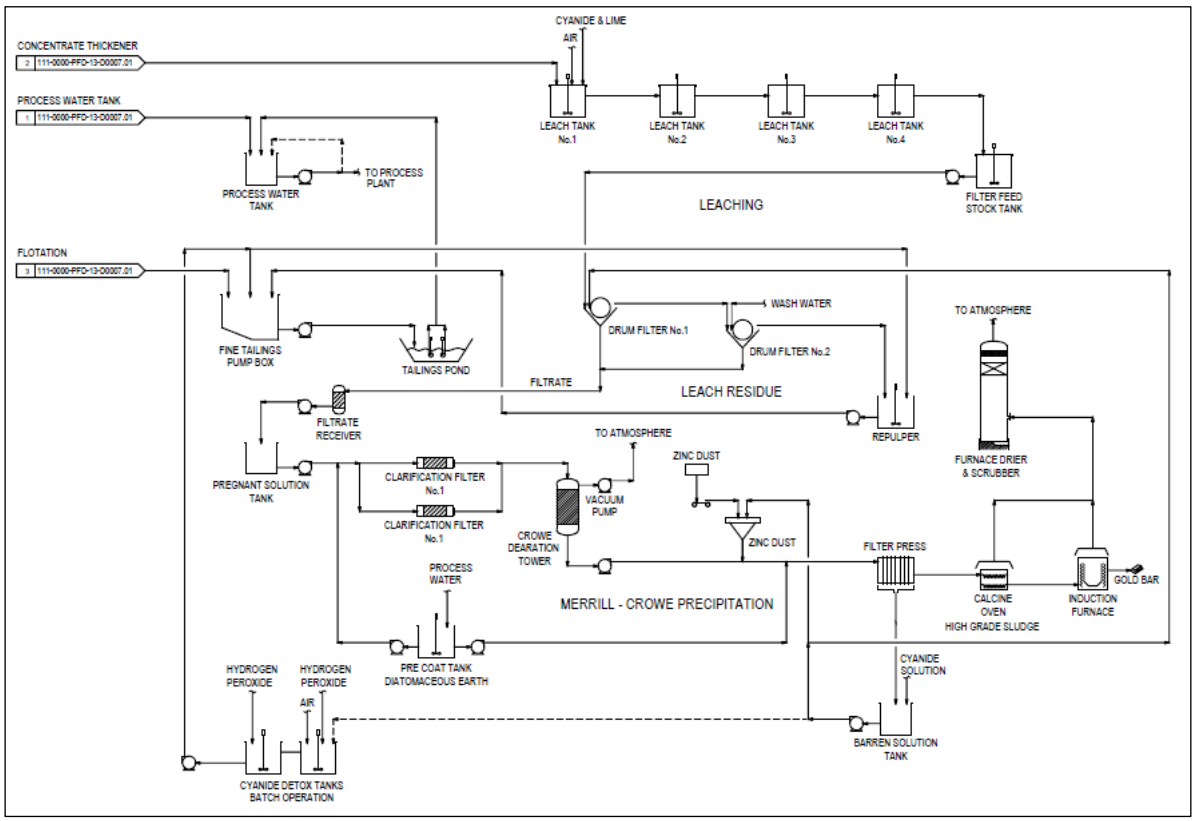


Figure 17-2: Historical Gold Recovery Circuit - 1,000 t/d

17.2.2 Operational problems identified: 1987 to 1989

Reviewing the available managerial monthly reports, it is apparent that the plant struggled to get out of the commissioning phase.

The primary crusher had a number of serious breakdowns during the early production period. The dust extraction system was reported to be inadequate with unacceptable levels of dust being generated. The gravity concentration Jigging Plant performance was never optimized to produce a consistent product at high gold recovery and the reliability was poor. The leaching circuit performance was compromised by the lack of oxygen required for leaching and at one stage, two of the four leach vessels were used for oxygenation rather than leaching.

There seemed to be a lack of constant feed to the plant considering the generally poor running time achieved.

There are also reports indicating that operator training was a concern with an impact upon plant performance and thus gold production.

The mineralogical composition of the ore proved to be variable with extreme fluctuations in gold head grade. It is unusual that other important elements in the ore such as sulphur, arsenic, copper, etc (with minerals such as pyrrhotite) were not analyzed daily and reported, as this would give an indication of the overall plant performance expectation.

The most serious problem was that the plant feed grade was not as expected from the mining plan and the plant recovery was not as anticipated from the metallurgical testwork due to the erratic plant performance.

17.2.3 Plant Condition as at January 2026

The PL plant has been inspected on a number of occasions by independent engineers and again in January 2026. A detailed assessment commissioned in 2013 by AECOM is the most recent review and is regarded as a good reflection of the current 2026 status of the crushing, milling, flotation, leaching and gold recovery plant.

The major equipment is in place, namely crushers, conveyors, mills, main pumps, flotation plant, concentrate thickener and leach vessels. The Merrill-Crowe gold recovery circuit with barren solution detoxification is available but the gold room has been stripped of all useful equipment. Numerous smaller pumps have been removed as well as interconnecting pipework.

The major equipment is summarized in the following mechanical equipment list:

- Jaw Crusher: 762 mm × 1067 mm Traylor type H
- Secondary Crusher: 1676 mm Symons SL:C5834 SHHD
- Vibrating Screen: 1524 mm × 3048 mm SALA Double Deck SH-RIPL-Flo
- Primary Rod Mill, HARDINGE 2.74 m diameter × 3.7 m long with 375 kW drive
- Secondary Ball Mill, HARDINGE 2.74 m diameter × 3.7 m long with 375 kW drive
- Flotation Cells: 3 x OK8 cells in rougher and 3 x OK8 in scavenger configuration
- Regrind Mill, MARCY 1.5 m diameter × 2.1 m long with 110 kW drive
- Thickener: 6.1 m diameter × 3.04 m high
- 4 x Leach Tanks with agitators: 2.6 m diameter × 4.0 m high: 20 m^3 each
- 2 x 1.80 m diameter × 1.50 m long EIMCO vacuum filters
- Pregnant & Barren Solution tanks: 2 off 4.6 m diameter × 9.0 m high
- Vacuum system for filter plant and Merrill-Crowe, filtrate receiver, moisture trap, vacuum tower, filter press, and various small tanks

The electrical cabling and much of the electrical switchgear has been removed and/or vandalized and the entire electrical system is to be replaced, this is being assessed to identify any components that can be refurbished.

The process control system and the plant control room have been vandalized and considered inoperable and is to be replaced. The plant was not well instrumented in 1987 and it is proposed to upgrade the process control system within the plant to optimize the milling and flotation performance.

Structurally, the crushing, conveying and main plant are in very good condition with minimal corrosion evident, apart from some minor surface corrosion. The mill bin shows some signs of corrosion and a detailed inspection will be required.

The analytical laboratory has been stripped of equipment, the general mine offices are devoid of all facilities and the warehouse has been repurposed as a maintenance facility.

17.2.4 Proposed Refurbishment and Repairs to Gold Plant

To be able to recommission the gold plant to achieve up to 1,000 t/d initially, the following refurbishment will be required:

- Non-destructive testing of the primary and secondary crushers, both ball mills and the regrind mill to confirm integrity
- Thickness testing of all tanks within the plant to confirm integrity
- Procurement of crusher and mill liners from the original manufacturer
- Overhaul/refurbishment of all mechanical equipment (e.g. crushers, mills, mill drives, main pumps, vacuum pumps, blowers, HVAC, compressors, agitator shafts, conveyor idlers and pulleys, etc), gearboxes and includes replacement of all bearings
- Overhaul and replacement of all pump liners for all pumps and identifying any missing pumps
- Detailed assessment of the electrical infrastructure
    - Refurbishment of all motors, especially motor bearings
    - Replacement of most of the electrical cables within the plant
    - Replacement/ refurbishment of electrical switchgear
    - Replacement and upgrade of the process control (SCADA) system and operator interface
- Detailed assessment of all pipelines within the plant and replace any that are suspect or missing plus all pipelines external to the plant
- The gravity recovery Jig is in a poor state of repair and it is proposed to replace this unit with a centrifugal gravity concentrator plus a leach reactor to increase the recovery of gravity gold as part of the initial refurbishment program
- Assessment and refurbishment of the Merrill-Crowe zinc precipitation system
- Complete rebuild of the gold room facility with new equipment

17.2.5 Gravity Recovery upgrade

The existing gravity Jig Equipment is to be removed to achieve the 1,000 t/d, and a centrifugal gravity concentrator installed being fed from a new pump installation located on the mill circulating sump. The gravity concentrator will incorporate a 3 mm vibrating screen with the oversize returned to a mill inlet with the undersize reporting to the concentrator. The discard from the concentrator will be directed to the mill sump (to maintain the water balance) whilst the gold concentrate will be collected for transport to the leach reactor.

The gravity concentrator facility is to be located in a new structure above the existing Jig Plant and will be fed from both milling circuits.

The gravity concentrate leach reactor is to be located on the roof of the gold room with the gold solution being pumped to a new pregnant solution tank and the washed tailings directed to the regrind circuit for size reduction and further gold extraction. The pregnant solution will be treated through the Merrill-Crowe plant in a batch for accounting purposes.

This gravity circuit upgrade is expected to improve the gravity recovery from the historic 47% to in excess of 60% as proposed from the initial testwork, thus reducing the 'load' on the flotation circuit recovery.

17.2.6 Proposed Flowsheet: 1,000 t/d

These proposed changes to the plant flowsheet when compared with the 1987 flowsheet are evident in the following Process Flow Diagram.

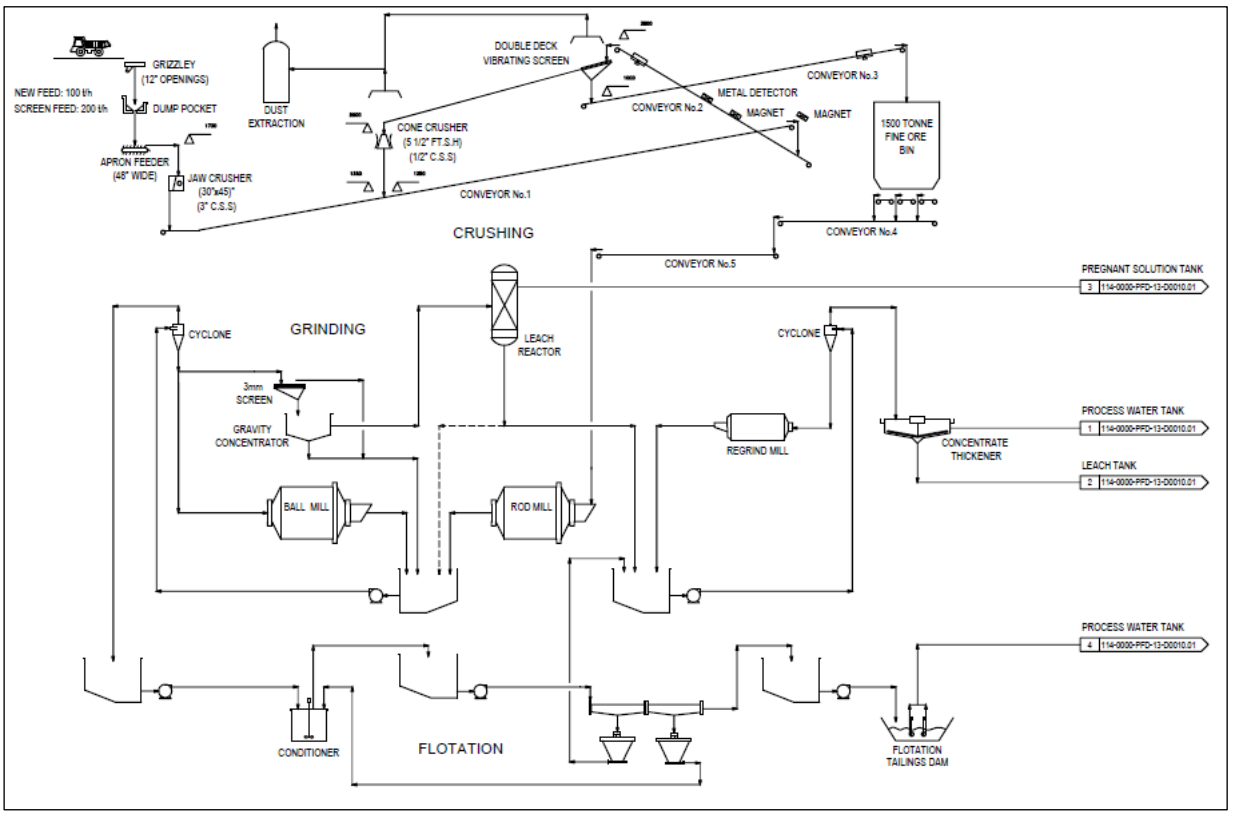


Figure 17-3: Changes to the plant flowsheet: 1,000 t/d

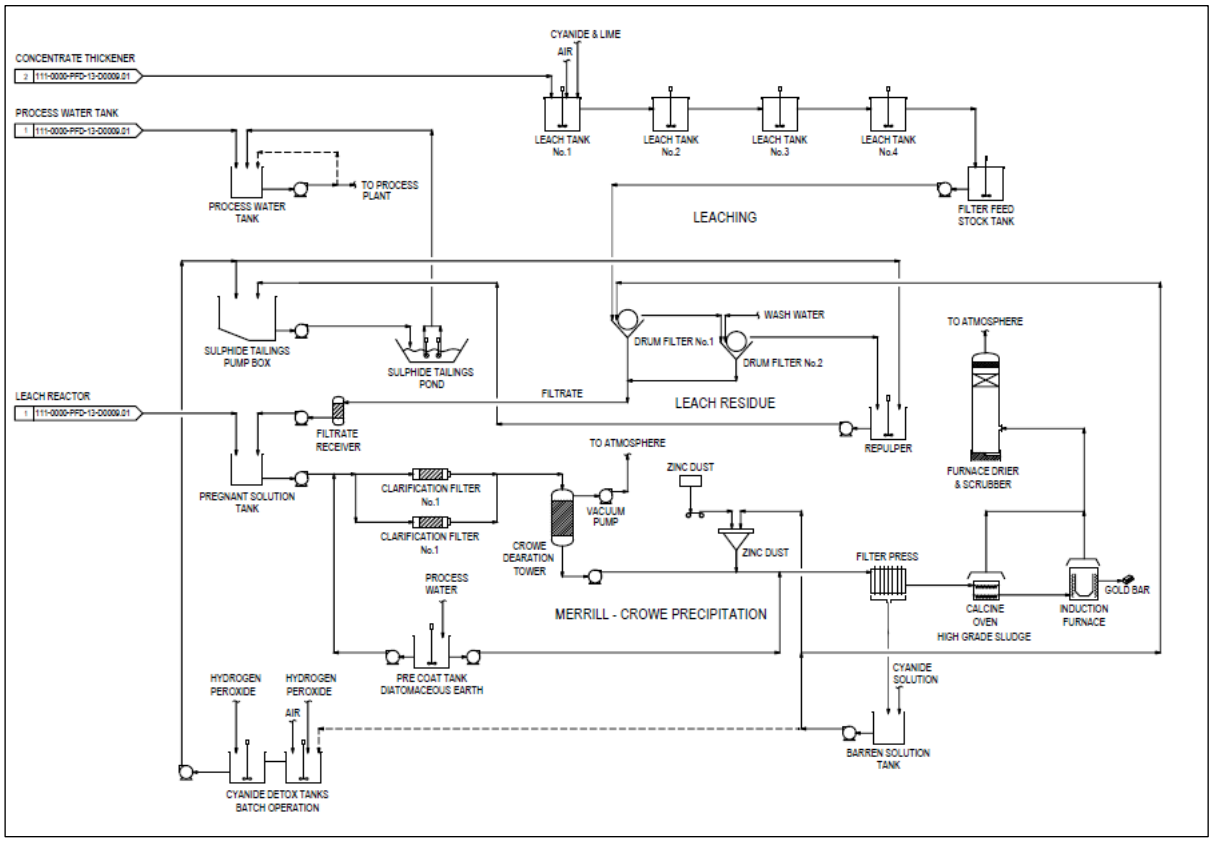


Figure 17-4: Changes to the plant flowsheet: 1,000 t/d

17.3 Proposed upgrades to the Flowsheet

To achieve an increased level of production to a targeted 2,000 t/d, plant upgrades will be implemented to a number of operating facilities within the gold plant as identified.

17.3.1 Crushing Circuit

The existing two stage crushing circuit can achieve a treatment rate of about 100 tph delivering the <15 mm product to the milling circuit. At an achievable 'up time' of 66%, a daily capacity is expected to be about 1,500 t/d. To increase this treatment rate to 125-130 tph (2,000 t/d), it is proposed to incorporate an additional secondary or tertiary crushing stage, external to the existing crusher building.

The tertiary crushing stage will also have the potential for finer crushing to <10 mm to improve the milling throughput, with the optimization of the screening capacity.

17.3.2 Milling Circuit

The current milling plant has a capacity of up to 50 tph or 1,100 t/d at an 'up time' of 92% at the required grind of <100 µm. A finer crush size would be expected to increase this capacity by about 10% to 1,200 t/d.

To increase the milling capacity to 2,000 t/d, an additional ball mill will be installed (estimated to be 3.6 m diameter × 7.5 m long with 1.4 MW drive) to deliver the extra 1,000 t/d, potentially within the mill building with additional pumps and an upgraded cyclone cluster. It is proposed that this new ball mill may receive crushed mineralized material from the mill silo via a new conveyor system rather than via the existing rod mill.

17.3.3 Gravity Recovery upgrade

The additional milling plant to achieve the 2,000 t/d treatment rate will incorporate an additional centrifugal gravity concentrator installed being fed from a new pump installation located on the mill circulating sump. The gravity concentrator will incorporate a 3 mm vibrating screen with the oversize returned to a mill inlet with the undersize reporting to the concentrator. The discard from the concentrator will be directed to the mill sump (to maintain the water balance) whilst the gold concentrate will be collected for transport to the leach reactor, installed with the 1,000 t/d refurbishment program.

This additional gravity circuit is expected to achieve the gravity recovery in excess of 60% as proposed from the initial testwork, thus reducing the 'load' on the flotation circuit recovery.

17.3.4 Flotation Circuit

The flotation circuit consisting of the two sequential banks of cells is to be modified with the incorporation of two tank cells as roughers and the existing cell banks modified to be parallel scavenger flotation cells. This will enable the increased throughput, with some minor modifications to the interconnecting slurry pipework

to be treated. The concentrate handling system is expected to be adequate with pump upgrades and modifications to the interconnecting pipework.

It is proposed that the rougher concentrate be further upgraded with the inclusion of a cleaner flotation circuit to upgrade the gold content of the material without loss of recovery. This will allow the leach circuit to remain substantially unchanged as well as the cementation circuit.

The concentrate thickener capacity is to be enhanced with the inclusion of concentrating cyclones with the coarser underflow directed to the regrind circuit and the finer overflow directed to the thickener. An enhanced flocculation system will be required to improve the settling characteristics.

17.3.5 Regrind System

The existing regrind ball mill is a typical installation from the 1980's. Modern regrind facilities incorporate vertical/tower mills with an improved performance.

It is proposed that the ball milling circuit be further evaluated during production and that consideration be given to replacing the circuit with modern grinding technology, subject to economic evaluation.

17.3.6 Concentrate Leaching

The existing four stage leach circuit has a nominal residence time of 72 hours. Reviewing the production reports from 1988, oxygenation of the slurry was inadequate, and the first leach vessels were used for pre-aeration rather than leaching, thus reducing the available leach residence time.

It is proposed that a shear reactor (e.g. Aachen Reactor or GoldOre MACH Reactor) system be incorporated to enhance the oxygenation level within the leach to improve the overall gold extraction. The same reactors can be incorporated to improve the cyanide leach kinetics with the existing tankage. To reduce the potential frothing in the leach tanks, an oxygen plant is proposed to deliver a about 90% purity oxygen supply to the shear reactor.

The proposed testwork to evaluate magnetic separation of pyrrhotite is likely to reduce the oxygen demand of the concentrate.

17.3.7 Gold recovery methodology

The existing Merrill-Crowe system is to be refurbished, and it is expected to have adequate capacity to recover the increased gold production with upgrades to pumps and piping.

The modification of the gold recovery circuit to incorporate carbon-in-leach has been considered as more modern technology but as the plant already has zinc cementation technology, the change is considered unwarranted. The carbon circuit would involve a major upgrade to the leaching circuit and additional carbon elution equipment and either the retention of the zinc cementation or the introduction of electrowinning facilities. This can be reviewed prior to 2 000 t/d upgrade, subject to prevailing economics.

In addition, the silver content in the mineralized material is a little lower than the gold content as can be seen with the doré quality and this aspect makes the carbon circuit less attractive, although within the typical Ag:Au ratio carbon recovery.

17.3.8 Gold room facilities

The gold room is to be re-equipped completely to process 1,000 t/d with provision to increase the gold quantity associated with up to 2,000 t/d. This will require calcining ovens with fume extraction, smelting furnace for the calcine and slag handling facilities.

There may be an opportunity to incorporate a gold refinery to produce 99.99% gold bullion on site plus silver ingots, subject to economic evaluation.

17.3.9 Tailings Disposal

Flotation tailings are the vast majority of the tailings to be generated from the gold plant, accounting for between 95% and 96% of the tonnage treated. This material is substantially sulphide-free and has not been exposed to cyanide leaching, with the sulphides having been recovered in the flotation concentrate. This material is expected to contain less than 0.2 g/t Au.

The sulphide concentrate, which contains arsenic plus other metals, has been exposed to cyanide leaching. The product is a filter cake which has been thoroughly washed and is expected to contain between 2 and 3 g/t Au. It is proposed that this material be stored in a lined facility for future reclamation and retreatment plus to contain the environmentally hazardous elements. The quantity of material will be 4% to 5% of the tonnage milled.

17.3.10 Proposed Flowsheet to 2,000 t/d

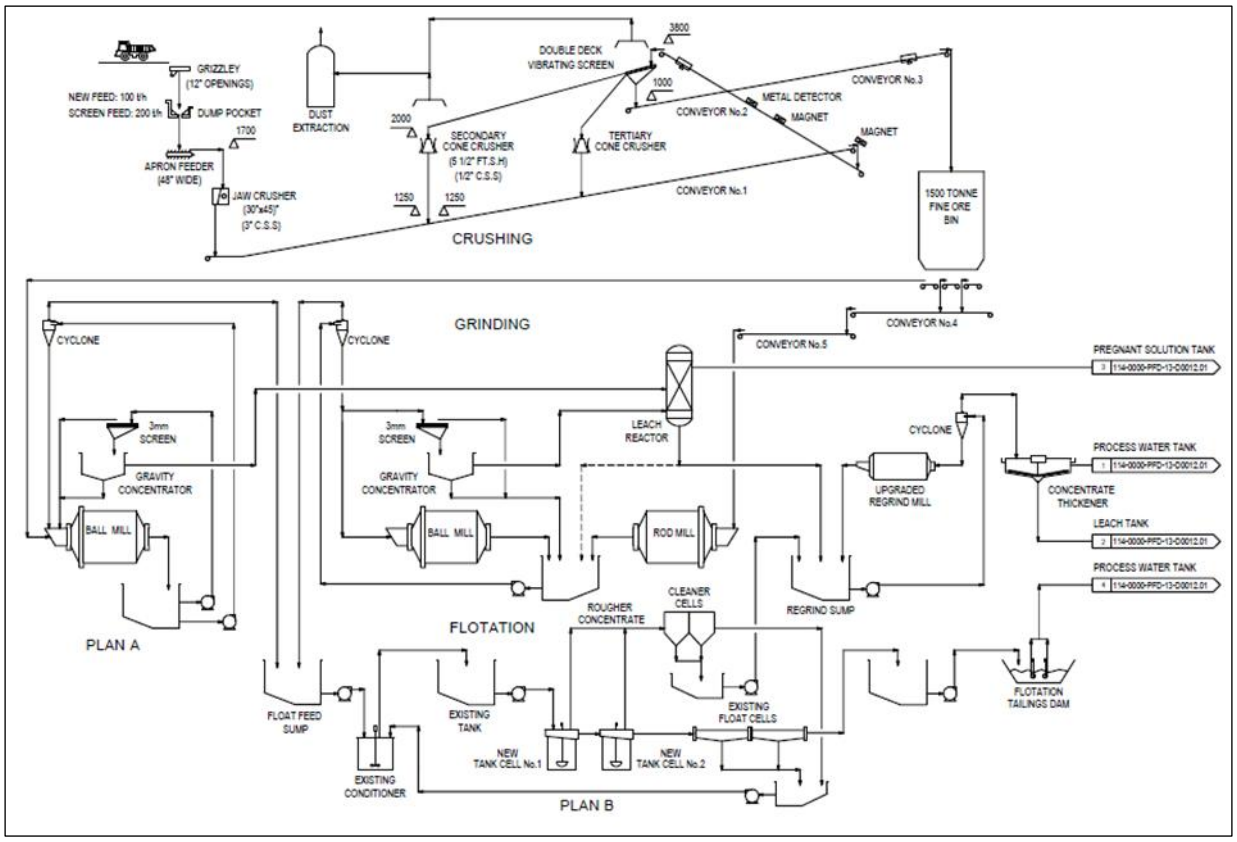


Figure 17-5: Proposed flowsheet for Crushing, Milling and Flotation: 2,000 t/d

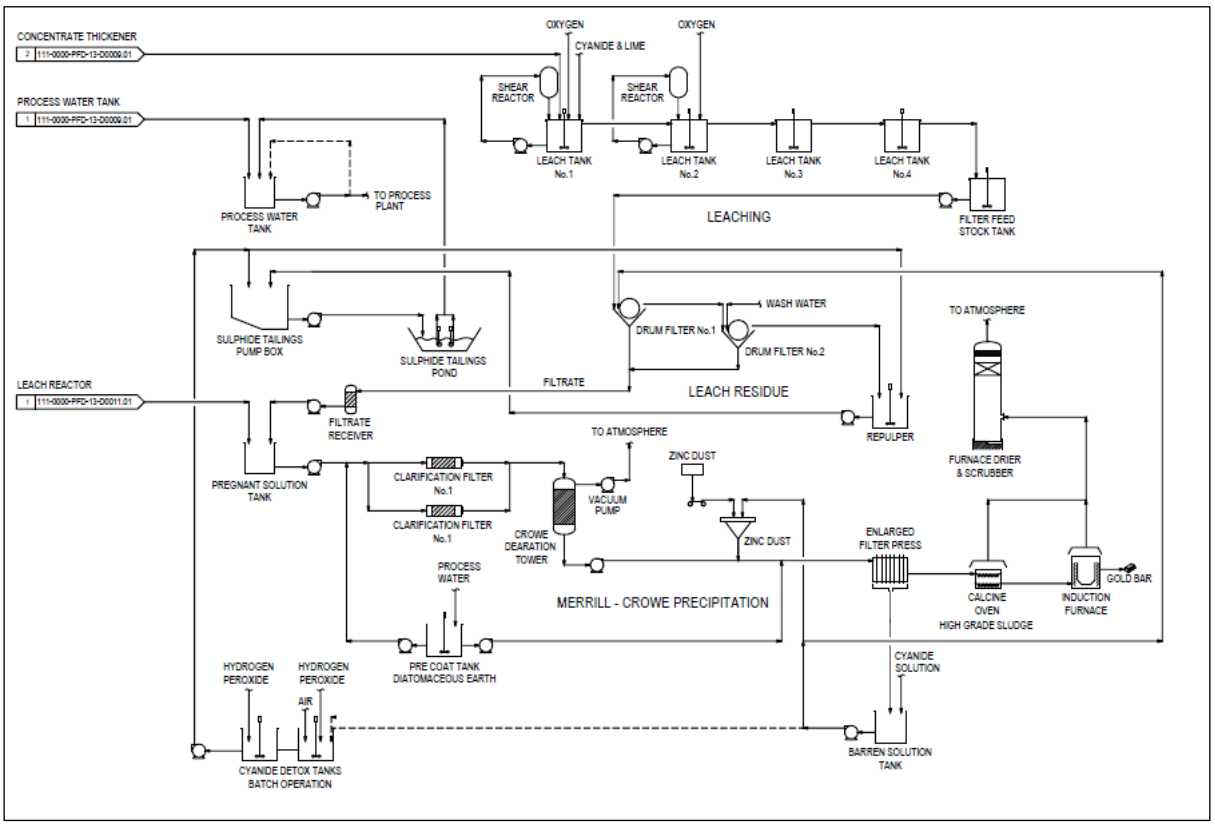


Figure 17-6: Proposed flowsheet for Leaching and Gold Recovery: 2,000 t/d

17.3.11 Analytical Laboratory

The analytical laboratory is to be refurbished and re-equipped fully. It is expected that a contractor will be appointed to equip and operate the laboratory. The laboratory is to be equipped to be able to evaluate the elements of interest to the management of the operation considering the mining and metallurgical requirements. The monitoring of the environmental requirements necessitates the incorporation of an environmental laboratory.

An oils laboratory is not required as all major mobile equipment will be procured and operated by mining contractors.

17.3.12 Design and Engineering

The design and engineering of the expansion to 2,000 t/d will require an owner's team plus a suitable engineering, procurement and construction management (EPCM) plant contractor to be able to incorporate the additional equipment into an operational plant.

The implementation schedule for the plant expansion is likely to be 12-15 months, dependent upon weather conditions and lead time for the mills.

17.4 Utilities Consumption

17.4.1 Power

Power consumption for the mine, based upon grid power consumed, varied from 65 kWh per tonne milled in summer to 90 kWh per tonne milled in winter, with a peak winter consumption of 115 kWh per tonne. The process plant is the major consumer of this power and it is estimated that the plant operations require 50 kWh to 55 kWh per tonne, excluding HVAC. This is considered a reasonable estimate based on industry norms for the employed flowsheet.

The plant expansion to 2,000 t/d will require a similar unit power consumption supplied partially from the grid with the balance being generated on site. The power required for the Gold Plant will be approximately 6.0 MW in winter peak with summer being approximately 4.5 MW, Gold Plant installed infrastructure (excluding HVAC) will be about 6.0 MW.

17.4.2 Water

Water is obtained from Ragged Lake (or Jay Lake depending upon quality from Ragged Lake) as make up for the plant operations and was historically dependent upon the moisture content being discarded to Ragged Lake as tailings. The tailings moisture was 35-40% solids or 1.75-2.00 tonnes of water per tonne of plant feed. At 1,000 t/d, the water consumption is 85 m^3 /hr and at 2,000 t/d, the consumption is likely to be 145 m^3/hr.

17.5 Plant Throughput

The current PL Plant has a nominal throughput of 1,000 t/d provided that the plant 'up time' exceeds 80-85%. The expansion to 2,000 t/d will require the addition of further processing equipment, namely crushing, milling and flotation with expansions and upgrades to the gold recovery circuit.

17.6 Gold Recovery

The gold recovery estimate for forecasting purposes is expected to be about 50% recovered through the gravity circuit with an 80% recovery through the leaching plant resulting in an overall gold recovery to doré of up to 90%. This includes any production efficiency losses in the discard solutions.

17.7 Reagents

The reagent consumption is estimated from 11 months of stable production data, serving as a reasonable forecast for the future.

17.7.1 Milling Circuit

Grinding Media consists of rods and balls for the main milling plant and balls for the regrind mill:

- Rod Mill Rods (89 mm diameter x 3.5 m): 0.49 kg/t

- Ball Mill Balls (38 mm): 0.79 kg/t
- Regrind Mill Balls (25 mm): 0.034 kg/t new feed or 0.80 kg/t leach feed

17.7.2 Flotation Circuit

Reagent consumption for flotation consists of collector, promoter and frother:

- Collector, Xanthate (PAX): 0.049 kg/t
- Promoter, Aerofloat 208: 0.027 kg/t
- Frother, Dowfroth 250 and MIBC: 0.035 kg/t

17.7.3 Leaching

Reagent consumption for leaching consists of lime and cyanide and hydrogen peroxide for oxygenation and cyanide destruction:

- Lime (CaO): 0.15 kg/t milled or 3.57 kg/t per tonne leached
- Lead Nitrate: 0.02 kg/t or 0.57 kg/t per tonne leached
- Cyanide: 0.24 kg/t milled or 5.56 kg/t per tonne leached
- Hydrogen Peroxide: 1.45 kg/t

17.7.4 Gold Recovery Reagents

The main reagent is zinc dust as used in the Merrill-Crowe process

- Zinc Dust: 0.01 kg/t or 4.61 kg/kg gold
- Filter Aide, Celatom FW50 - 0.036 g/t milled or 0.82 kg/t per tonne leached

17.8 Tailings Handling

Tailings are to be separated into sulphide and non-sulphide streams. The sulphide stream will be the residue from the concentrate leaching circuit and is to be stored in a lined tailings facility for potential future remining. The non-sulphide tailings will be from the flotation plant and it is currently proposed that this will be sub-surface deposited into the existing Ragged Lake facility. This facility was permitted for use for the original production in 1988.

17.9 Capital Estimate

The estimated capital cost for refurbishing the existing plant and expanding it to 2,000 t/d has not been precisely determined and is based on assessments from 2014.

17.9.1 Refurbishment of Plant Facilities to 1,000 t/d

The refurbishment of the plant facilities and the initial upgrades to the plant are summarized below:

- Design and management of the refurbishment program: US$0.50M
- Non-destructive testing (NDT) evaluation of all equipment: US$0.15M
- Refurbishment/ replacement of major crushing, conveying, belting, milling, flotation, leaching and gold recovery mechanical facilities: US$5.50M
- Refurbishment/ replacement of internal plant cranes, pumps and pipelines: US$0.50M
- Replacement of external pipelines for tailings and water supply with pumps: US$0.50M
- Necessary repairs to the tailings dam weir that was damaged previously and maybe again (as per 1988 report): US$0.15M
- Installation of the upgraded centrifugal gravity recovery circuit and leach reactor: US$1.25M
- Replacement of the electrical switchgear, cabling and testing: US$4.00M
- Replacement of instrumentation and SCADA control system: US$0.75M
- Upgrade of the plant sampling systems: US$0.15M
- Refurbishment/ replacement of plant HVAC system: US$0.50M
- Structural refurbishment as required (including the mill bin): US$0.15M
- Refurbishment of laboratory and general/plant offices: US$0.50M
- Procurement of operational spares for 1,000 t/d plant: US$0.40M
- Contingency Provision: US$5.00M
- Sulphide Tailings Storage Facility
    - Design - US$0.25M
    - Construction: US$2.50M
- **Estimate Total Expenditure for 1,000 t/d excluding Tailings Facility: US$20.00M**
- **Tailings Facility new: US$2.75M**

The analytical laboratory will be re-equipped, likely by an independent contractor, with costs charged as operating expenses and excluded from the capital budget.

17.9.1.1 New Facilities as Proposed to 2,000 t/d

To achieve the proposed 2,000 t/d plant expansion, the cost estimate is summarized below:

- Design, management and supervision (EPCM) of the upgrades (internal and external): US$1.00M

- Crusher plant upgrade including tertiary crusher, conveyors and screening - US$1.50M
- Additional ball mill with feed system, hydrocyclones & structure, pumps, pipelines, electrical switchgear and installation: US$5.00M
- Additional flotation capacity: US$0.70M
- Incorporation of a cleaner flotation module with pumps: US$0.35M
- Upgrade to thickener feed arrangement with cyclone and flocculant makeup: US$0.25M
- Replacement of the regrind milling circuit: US$1.5M
- Upgrade to plant pumps and piping as required: US$0.75M
- Installation of shear reactors in leach circuit: US$1.40M
- Installation and reticulation of oxygen plant system: US$1.00M
- Miscellaneous upgrades: US$2.0M
- Procurement of operational spares for upgraded plant: US$0.40M
- Building extensions including HVAC for additional plant: US$1.00M
- Contingency Provision: US$3.155M
- Phase 1 Ragged Lake tailings weir installation: US$0.50M
- Phase 2 Ragged Lake tailings weir upgrades and containment with appropriate lining of the containment wall: US$4.50M plus US$4.00M for lining and placement
- **Estimate Total Expenditure for expansion to 2,000 t/d excluding Tailings: US$20.00M**
- **Tailings Facility Upgrade: US$9.00M**

17.9.2 Process Plant and Tailings Capital: QP Comment

The capital estimates above were prepared by G. Cunningham from 2014 assessments updated for the January 2026 plant inspection. They differ from the allowances in the PEA cash flow model, which are used for the base case. On October 29, 2026 G. Cunningham provided the following comments as QP for this Item:

- Expansion to 2,000 t/d: the plant expansion estimate is US$16.8M plus US$3.2M contingency (US$20.0M), compared with US$14.0 million in the model (mill upgrade US$10.0 million and site and mill electrical expansion US$4.0 million, without contingency).
- Tailings: the US$9.0 million tailings facility upgrade (Ragged Lake weir phases 1 and 2, with lining and placement) is preferred to the US$4.0 million of sustaining tailings construction in the model.
- Contingency: the 15% model contingency should not be applied to the US$20.0 million refurbishment to 1,000 t/d, because that estimate already includes a US$5.0 million contingency. Removing it avoids double counting.

These comments are presented as a cost difference only. On October 29, 2026 G. Cunningham agreed that the PEA base case remains as published, using the model values, and that his estimates are tested as a sensitivity. The effect of his estimates is reported in Item 22 (sensitivity C2).

17.10 Operating Costs

The plant operating cost, excluding power, is estimated at US$22.18 per tonne treated, based on similar operations at 1,000 t/d with a fixed cost proportion of 40%. The model uses US$22.16/t excluding power, which is consistent.

The processing power consumption is estimated at 50 to 55 kWh per tonne. This covers processing only. The 75 kWh/t used in the model for the mill includes heating, ventilation and air conditioning (HVAC) and the offices, so the two figures are consistent. The 1,000 t/d plant is likely to be supplied from liquefied natural gas (LNG) generated power until grid power is reconnected (based on the Manitoba Hydro estimate for replacing the line to Sherridon). In the QP's opinion a grid power cost of US$0.06/kWh, as used in the model, is current and acceptable.

Including power at US$0.06/kWh, the plant operating cost is estimated at US$38/t at 1,000 t/d and US$27/t at 2,000 t/d. Over the life of mine (LOM) the average is close to the model processing cost of US$28.13/t milled (including mill power), so the model is consistent with the QP's estimate.

17.11 Construction Schedule

A detailed construction schedule has not been completed for either the refurbishment of the 1,000 t/d plant (Project 1) or the upgrade to the 2,000 t/d plant (Project 2).

It is anticipated that Project 1 will require about 12 months to evaluate, order the necessary replacement components from OEMs or others, install the electrical equipment and pipework, and install the mechanicals for the gold plant. The longest lead times are expected to be the transformers and some of the milling components.

Project 2 will need to be fully engineered, including complete Process Flow Diagrams with Mass Balances, and the selected equipment needs to be ordered no later than when Project 1 is commissioned. The longest lead item is expected to be the milling plant, with delivery from order placement at 9 to 10 months plus installation. It is anticipated that the Project 2 installation will require 12 months to construct following the completion of detailed design. The metallurgical and engineering design is proposed to commence only after Project 1 has been commissioned and additional testwork has been completed to confirm the plant capacity.

The 21-month period in this schedule covers operation of the refurbished plant at 1,000 t/d to confirm the process, followed by the construction and commissioning of the 2,000 t/d plant.

Project 2 will be executed whilst Project 1 is in operation, raising safety and security concerns, especially given the high-grade gold component in the mineralized material.

This construction estimate is regarded as a 'best case scenario' and does not include the delays that may occur due to extreme cold weather at the mine site or equipment supply interruptions due to lack of design skills and equipment availability constraints.

The PEA base case assumes that the plant reaches 1,000 t/d in Month 4 and 2,000 t/d in Month 8 (first feed is in Month 3, March 2028). This is faster than the construction sequence described above, and the schedule is a risk to the base case. The 2,000 t/d plant must be in place as soon as throughput exceeds 1,000 t/d, not only when 2,000 t/d is reached. The QP's ramp-up basis is assessed as sensitivities C1 and C3 in Item 22, and the detailed execution schedule is to be developed at the next study stage.

17.12 Production Ramp-up

The QP's ramp-up profile, set out in the ramp-up workbook, defines both a tonnage ramp-up and a recovery ramp-up. Project 1 (refurbished 1,000 t/d plant) treats 22,500 t, 25,000 t and 27,500 t in its first three months and 30,000 t a month (1,000 t/d) from the fourth month. Its recovery rises from 75% in the first month to 77.5%, 80%, 84% and 87%, reaching 90% in the sixth month.

Project 2 (expansion to 2,000 t/d) adds 25,000 t, 27,500 t and then 30,000 t a month, with recovery on the additional tonnage of 80%, 85% and then 90% in successive months. In the workbook, first feed is in Month 7 and Project 2 starts in Month 18. The Month 7 first feed is a placeholder: the project team expects first feed in about January 2029, rather than July 2028, to allow more time for plant refurbishment. For the Item 22 sensitivity, first feed is taken as January 2029, as agreed by the project team on October 29, 2026, with Project 2 starting its ramp in December 2029. The base case keeps the published schedule, with first feed in March 2028.

The base case is not changed by this ramp-up profile. The base-case plant feed rises to 60,000 t a month in Month 8 at a flat 90% recovery.

17.13 Gold Inventory and Doré Shipment

The QP notes the following operating considerations, which are not costed separately in the PEA:

- Gold lock-up: a gold inventory equivalent to about two weeks of plant production will be locked up in the circuit at start-up and recovered at final clean-up.
- Doré shipment: at 2,000 t/d doré shipments are expected to be about 2,000 oz to 2,800 oz of doré a week. Shipment frequency, security and helicopter transport from site are to be planned with the refiner and insurer.

17.14 Process Plant Management

It is proposed that senior plant operations and maintenance management be instituted as early as possible (after funding has been secured) to oversee the requirements of Projects 1 and 2 and to engage the appropriate Canadian-based EPC Process company to design and implement the PL strategy.

It is unlikely that trained staff will be readily available to operate the plant; therefore, a training program is to be implemented several months prior to plant re-commissioning.

18 PROJECT INFRASTRUCTURE

The PL Gold Mine is a past-producing operation (1987 to 1989), and much of the original surface and underground infrastructure remains on site under care and maintenance. Later in 1989, after the operation had been placed on care and maintenance, forest fires damaged the power line and the freshwater and tailings pipelines (2018 FS, pp. 43 and 47; AECOM, 2014; Manitoba Mineral Inventory File 0984). Parts of the site have since been vandalized, and the 2025 wildfires caused further damage to the power line. The condition descriptions in this Item are based on site information from Minnova and ABGM. The PEA infrastructure concept reuses previously disturbed ground and existing facilities as far as practical, refurbishes the processing plant (1,000 t/d, with an upgrade to 2,000 t/d), and adds new facilities for open-pit mining, tailings expansion, power generation and worker accommodation. PEA underground access is through a new portal in the north-east pit, as described below. The historical portal and decline will be dewatered and rehabilitated for the recommended trial mine, which is a PFS recommendation and is not in the PEA schedule. Unless stated otherwise, the descriptions below are based on the 2017 Feasibility Study (2018 FS), the 2026 project description and Intent to Apply form for the Environment Act submission (AECOM, draft 2026; Minnova, 2026), and the PEA cash flow model (ABGM, 2026). Where these sources conflict, the 2026 documents take precedence.

Figure 18-1 shows the PEA general site layout. The open pits lie on Mining Lease ML065. Underground access is from a new portal in the north-east pit at 373,664 E, 6,100,247 N (292 m RL). The process plant is at 373,357 E, 6,101,191 N (55°02’28.95” N, 100°58’55” W; AECOM, 2026), with the run-of-mine (ROM) pad to the north. Flotation tailings go to the Ragged Lake TMA east of the pits, closed by a conceptual non-acid-generating (NAG) waste rock embankment (perimeter B, crest 352.4 m, pond 351.4 m, enclosing 124.8 ha), and sulphide tailings go to a lined Sulphide TMA outside the embankment (toe about 6.05 ha, centroid about 373,613 E, 6,100,881 N, with a 193 m pipeline from the plant). Waste rock is stored in two WRDs: a south-west WRD on ML065 and a north WRD on claim MIT-045 (MB12260). The south-west WRD is sited from imagery and elevation data and will be confirmed by site inspection and geotechnical investigation at the PFS stage. Surface infrastructure shown on Figure 18-1 includes the process plant and services building, a truck shop, the emergency genset, a temporary 70-person camp, the LNG/CNG pad and power plant, a 525 kL fuel farm, the contractor yard, the effluent treatment plant (ETP) and discharge, a security gate, explosives and detonator magazines, and the 25 kV line to Sherridon. A 100 m separation between the detonator and explosives magazines is achieved with an earth mound (barricade); the unbarricaded separation would be 305 m. The worker lodge, topsoil and organics stockpile and contractor laydown follow the 2026 draft project description (Figure 18-2); the two waste rock pads south of the pits shown in Figure 18-2 are replaced in this PEA by the two WRDs. Coordinates are NAD83 / UTM Zone 14N.

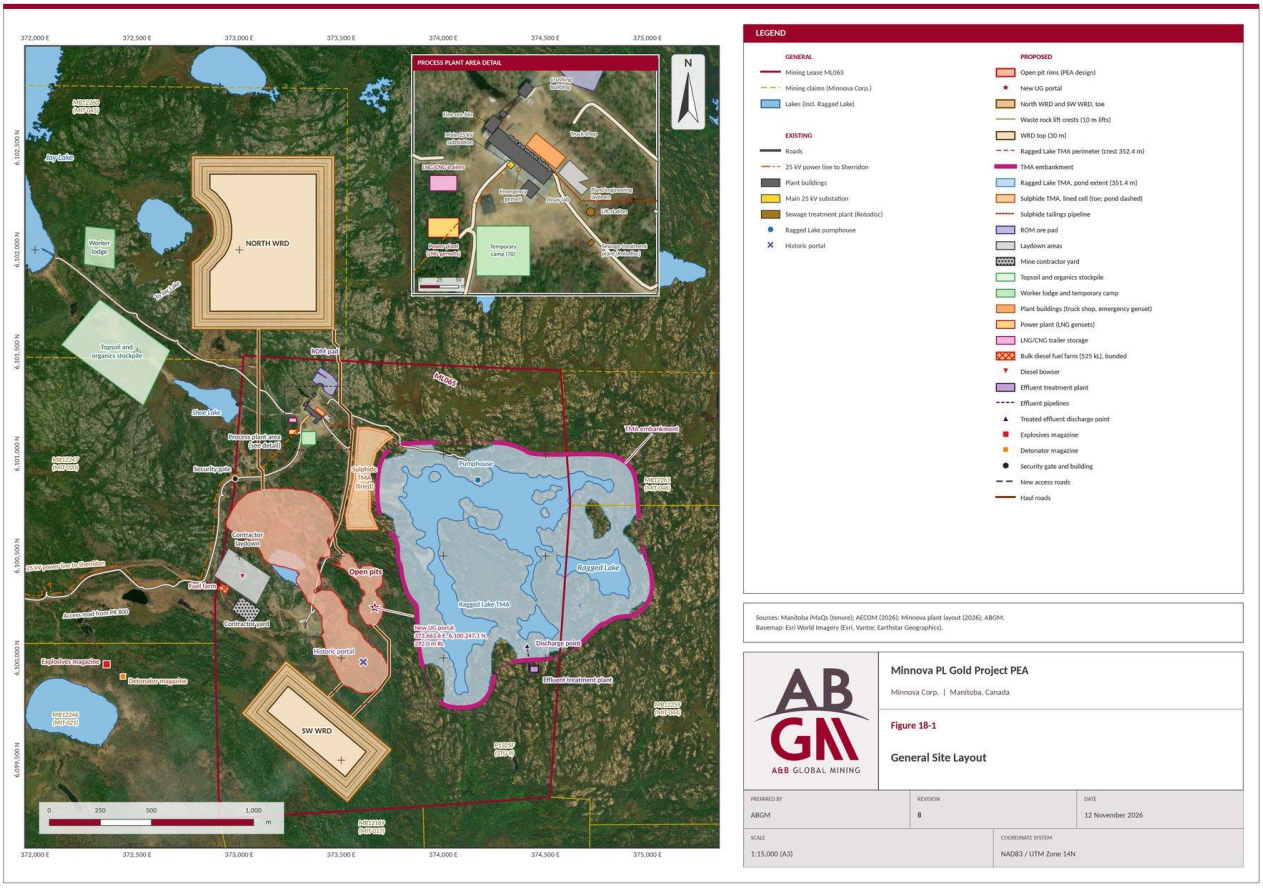


Figure 18-1: General site layout

Source: ABGM, November 12, 2026 (block plan rev8). Pit rims: PEA design (November 11, 2026); Ragged Lake TMA perimeter B and split waste rock dumps: ABGM (November 11, 2026); new underground portal: ABGM (November 12, 2026); Sulphide TMA, ROM pad, laydowns, lodge, topsoil stockpile and other facilities after AECOM (2026), Figure 1-2, and the Minnova plant layout; claims and lease from Manitoba iMaQs; basemap Esri World Imagery.

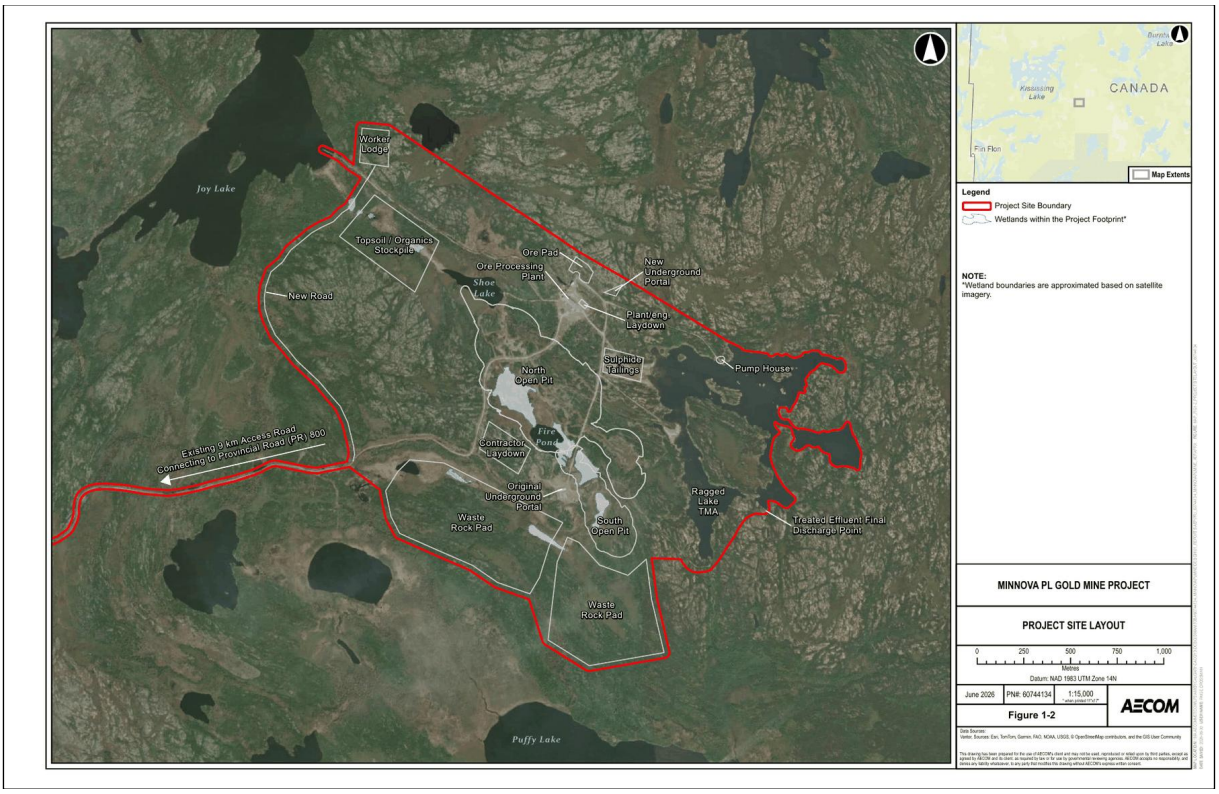


Figure 18-2: Project site layout in the 2026 draft project description

Source: AECOM, Figure 1-2, June 2026

18.1 Existing Infrastructure Condition Assessment

The infrastructure available on site comprises:

- the crushing plant and the gold processing plant (flotation, concentrate leach and gold room), which need refurbishment, upgrades and replacement of the main power cabling;
- the processing/services building, which houses a maintenance shop and a warehouse (each about 24 m × 10 m × 10 m; the shop has a 10 t overhead bridge crane) and the office and dry complex;
- the assay laboratory building, which is serviceable but needs complete re-equipping;
- the historical decline portal, ramp and about 7,000 m of underground development to about 135 m depth, currently flooded;
- a core-logging and cutting facility with open-air core storage;
- a small exploration camp of 16 rooms (connected, propane-heated trailers, with kitchen and recreation room), and an unused larger camp area near the portal;
- the historical Ragged Lake tailings management area (TMA), with about 350,000 t of tailings, the reclaim pumphouse and pipelines (fire-damaged);

- the 9 km access road from Provincial Road (PR) 800, which crosses the Keewatin Railway line about 6 km from the mill, and the 25 kV overhead powerline and on-site distribution lines to Sherridon (damaged by the 1989 forest fire and further by the 2025 wildfires).

Siemens tested the main 25 kV substation (one 25 kV/4,160 V and one 25 kV/600 V transformer, each 1,500 kVA) in 2017 and found it to be in good condition with minor refurbishment needed; the 4,160 V unit showed 8 ppm acetylene and must be re-sampled. The crusher-building substation and the main MCC room were vandalized and need replacement. An updated condition assessment of the existing buildings and the electrical and mechanical plant will be carried out at the PFS stage. The PEA refurbishment allowances are based on the 2017 and 2026 site inspections and the financial model allowances.

18.2 Access Road and Site Roads

The site is reached by PR 800, an all-season gravel road between Provincial Trunk Highway (PTH) 10 and Sherridon, and then by the 9 km gravel main access road east from about km 65.5, which is controlled by a security gate. The existing road is about 4.5 m wide and has few working culverts. It is passable year round, although poorly drained sections become muddy and rutted in spring. The 2026 plan is to widen the main access road by about 2 m (to about 6.5 m), with pullouts, to replace culverts and to resurface with non-acid-generating (NAG) rock and aggregate. The road will have safety berms along the sides and will be shaped to drain water away from the road surface (Figure 18-3). A mobile aggregate crushing plant will be used for road building and for any terracing needed on site. It also includes a new single-lane gravel road of about 2 km to the new worker lodge and the relocation of short site roads around the open-pit footprints. About 40,000 t of rock and 40,000 t of sand and gravel will be sourced from former borrow areas along the access road under dispositions under The Mines and Minerals Act. The model carries US$0.5M for road upgrades (Years -2 and -1).

Estimated project traffic is about 38 one-way trips per day (up to 6 waste and overburden hauls, up to 2 mineralized material hauls, 10 site deliveries and 20 light vehicles or buses), plus about 18 construction vehicles on site during open-pit mining and closure (AECOM, draft 2026).

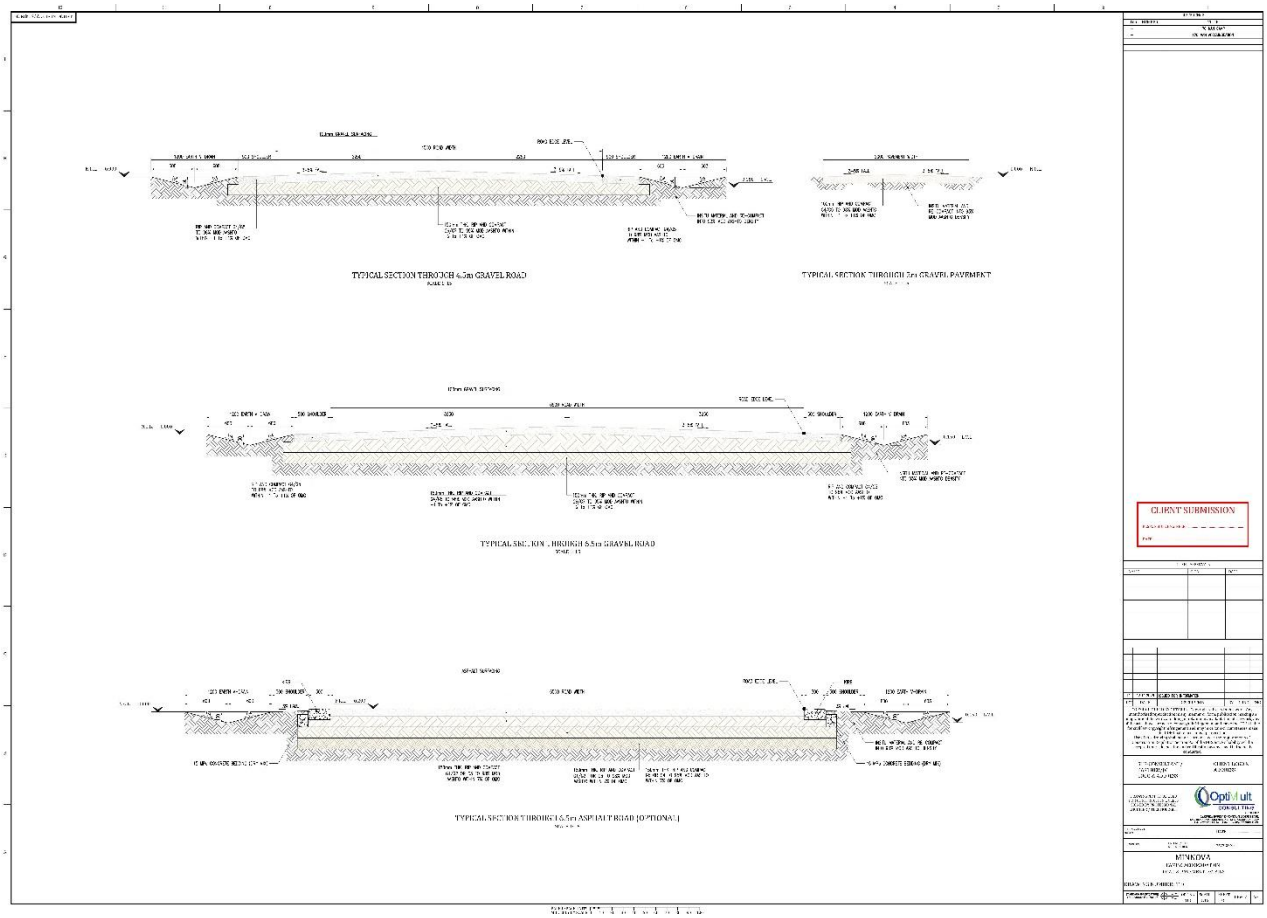


Figure 18-3: Typical access road cross-section

18.3 Power Supply and Distribution

Historically, Manitoba Hydro supplied grid power from its substation at Sherridon over a 22 km, three-phase 25 kV line (about 300 poles), stepped down on site to 4,160 V and 600 V. The original supply of about 4 MW was adequate for 1,000 t/d. Minnova bought the line from Manitoba Hydro and completed the purchase in December 2021, so Minnova owns the line between the mine and the Sherridon substation. The line needs to be refurbished before it can be used: about 10 km of the line lacks conductor (the section along the access road was destroyed by the 1989 forest fire; 2018 FS, p. 43), and the 2025 wildfires caused further damage, including the loss of several poles. The Project power demand is estimated at 4 MW to 5 MW for 1,000 t/d in Year 1, rising to 8 MW to 10 MW at 2,000 t/d.

On January 30, 2026, Minnova's consultant, AMPS, applied to Manitoba Hydro to restore the historical service and upgrade the line to 25 kV. Minnova also submitted a large-load request to Manitoba Hydro for 9 megavolt-amperes (MVA), with an intake form dated May 13, 2026. Manitoba Hydro has advised that the 115 kV Snow Lake to Sherridon transmission line was damaged by the 2025 wildfires, that restoration may take about two years (up to three years), and that no new commercial load will be connected to the Sherridon station until the line is restored. The capacity available after restoration may be about 3 MW to 3.5 MW, which is below the Project demand. A tap from the 230 kV transmission line was also assessed, but Manitoba Hydro has set

a minimum load of 30 MVA for such a connection, so it is not viable for the Project. The easement for the Minnova power line is still to be resolved (Minnova, emails from G. Glenn dated October 21 and 22, 2026).

The PEA therefore assumes on-site generation from start-up. Diesel generation is used initially, transitioning to liquefied natural gas (LNG) supplied by truck and stored in trailer-mounted tanks next to the processing plant, with grid power assumed to become available from Month 37 (2031) in the model. Installed generating capacity in the model totals 17.2 MW: 1.2 MW (Year -2, testing), 2 MW (Year -1), and 5 MW and 9 MW units (Year 1). The 9 MW unit is needed in Year 1 because the site load including the mill exceeds 8.2 MW from Month 8. The estimated site load including ventilation is about 14.6 MW. The LNG supply terms are described in Item 19. Because of Manitoba Hydro's advice, grid power at the Project demand from Month 37 is not assured, and on-site LNG-fuelled generation may be needed for the life of mine. Grid connection is to be assessed at the next study stage. The base case uses LNG genset power until Month 37 (2031) and grid power at US$0.060/kWh after that. A life-of-mine LNG generation case, the power study, the load list and the genset and LNG facility layout will be completed at the PFS stage.

The ventilation design in the Umoya VENTSIM life-of-mine study uses three main fans: South (395 kW), Central (654 kW) and North (688 kW), a total of 1,735 kW for 277 m³/s of airflow, with variable speed drives and auxiliary fans of 15 kW to 55 kW. The main fan power is consistent with the operating ventilation load of about 1.7 MW carried in the model within the allowance of 60 kWh per tonne of underground mineralized material. Other unit power allowances in the model are 75 kWh/t for the process plant (including HVAC and offices), 10 kWh/t for the open pit, 25 kWh/t for other site uses and 15 kWh/t for the lodge. The ventilation study estimates peak mine-air heating demand of 3,500 kW, 4,200 kW and 4,000 kW at the South, Central and North downcast shafts. Waste heat recovered from the LNG gensets is the planned primary heat source. Heat recovery and distribution are not costed in the PEA and will be designed and costed at the PFS stage.

18.4 Fuel Storage

The model includes a bulk diesel fuel farm of 7 × 75,000 L double-walled tanks (525 kL; US$0.97M including contingency). The 2026 project description also provides for a mobile diesel bowser at the 3.8 ha contractor laydown area, trailer-mounted LNG/CNG storage next to the plant, and an emergency diesel generator. Explosives and process reagents are to be supplied as just-in-time deliveries by the contractor and suppliers based in Flin Flon. The PEA provides a 525 kL fuel farm, explosives and detonator magazines separated by an earth mound, and reagent storage at the process plant. The fuel tank configuration and the magazine and reagent storage designs will be completed at the PFS stage.

18.5 Water Supply, Site Water Management and Effluent Treatment

Fresh water was historically pumped from Jay Lake, about 2 km north of the plant, and process water was reclaimed from Ragged Lake. The pipelines were largely destroyed by the August 1989 forest fires, after the operation had been placed on care and maintenance (2018 FS, p. 47). Water for the worker lodge will be drawn from Jay Lake under an updated Water Rights Act licence. The existing industrial water licence (No. 2017-116) covers the 1988 requirement of about 4,000 m³/d for 1,000 t/d and will need renewal and

amendment for 2,000 t/d. Reclaimed Ragged Lake water becomes cloudy in winter and may need clarification (settling and polishing filtration) before reuse. There is no potable-water treatment on site.

Pit dewatering water, contact water and runoff will flow overland to the Ragged Lake TMA, as under the historical licence. The open pits, the ROM pad and the Ragged Lake TMA lie within the Ragged Lake drainage basin. The drainage of the south-west WRD (on ML065) and the north WRD (on claim MIT-045) is to be confirmed; runoff and seepage from both WRDs will be collected and, where needed, pumped to the Ragged Lake TMA. The 2018 FS sized a lime/ferric-sulphate metals-removal effluent treatment plant (ETP), with ion exchange for ammonia, at an average 7,855 m^3/d for 1,000 t/d, pumped over a 180-day season. A water treatment facility may be needed to meet the discharge limits in the 2012 licence update. A pond of about 2 ha ("Fire Pond") must be drained for the North Pit. The model carries US$2.0M for water handling and reticulation, US$1.0M for underground dewatering and US$0.5M for water treatment and discharge.

18.5.1 Stormwater Management

The PL Project is located within the Ragged Lake and Puffy Lake catchments in an area characterized by numerous lakes, wetlands, low-lying terrain and seasonal snowmelt. Surface water management will be incorporated into the project design to minimize contact between clean water runoff and mine-affected water (dirty water), reduce peak flow events, and protect downstream receiving environments. The Project area is situated within typical Canadian Shield terrain where elevations range from approximately 340 m to 350 m above sea level and drainage naturally reports toward adjacent lakes and low-lying wetland areas.

The proposed stormwater management strategy is based on the separation of clean and dirty water. Clean runoff from undisturbed catchments will be diverted around operational areas using diversion berms, perimeter cut-off drains and natural drainage corridors. Dirty water generated from the open pits, WRDs, mineralized material stockpiles, plant areas, haul roads, tailings facilities and other disturbed areas will be collected and directed to sedimentation and containment systems prior to treatment, reuse or controlled discharge. The strategy is intended to limit sediment transport, reduce the volume of water requiring treatment and improve overall water management efficiency.

Open pit stormwater and groundwater inflows will be managed through pit dewatering systems and perimeter cut-off drains. Water collected within the pits will be pumped to the site water management system and directed to the Ragged Lake TMA and associated water management facilities for reuse in the process plant where practical. The existing site water management philosophy assumes that pit dewatering water, dirty water and surface runoff from disturbed areas will report to the Ragged Lake TMA prior to treatment or release.

Runoff from WRDs will be managed using perimeter cut-off drains, sediment-control structures and collection ponds. Potentially acid generating (PAG) and non-acid generating (NAG) waste rock will be managed separately in accordance with the site's mine waste management strategy, reducing the risk of contaminated runoff and facilitating long-term water quality management.

The expanded Ragged Lake Tailings Management Area (TMA) will continue to serve as the primary collection and storage facility for site dirty water and flotation tailings. Stormwater reporting to these facilities will form part of the overall site water balance and will be managed to maintain adequate freeboard and operational storage capacity during major precipitation and snowmelt events.

At the PEA level, stormwater infrastructure sizing is conceptual and will be refined during the Pre-Feasibility Study through development of a detailed site-wide water balance, hydrological assessment and hydraulic modelling. These studies should confirm diversion requirements, pond capacities, spillway designs and regulatory discharge criteria under the applicable permitting framework.

This stormwater management approach is intended to support regulatory compliance, maintain operational reliability and minimize impacts on surrounding water resources throughout the life of mine.

18.6 Tailings Management Facilities

The Ragged Lake TMA (Environment Act Licence 1207E, October 13, 1989; status re-confirmed by Manitoba on May 17, 2012) received about 350,000 t of blended flotation and detoxified leach tailings from 1987 to 1989. It was designed as a closed system. A 1.5 m weir at the southern end prevents discharge towards Puffy Lake, and a 1988 assessment recorded weir damage. The 2026 plan changes tailings handling in three ways:

- Sulphide tailings: the leached sulphide-concentrate tailings, which contain elevated arsenic and make up about 4% to 5% of milled tonnage, will be stored in a new lined Sulphide TMA outside the Ragged Lake embankment, with a toe of about 6.05 ha, a centroid at about 373,613 E, 6,100,881 N, and a 193 m pipeline from the plant.
- Ragged Lake TMA: the substantially sulphide-free flotation tailings, up to 1,900 t/d, will go to the Ragged Lake TMA, which in this PEA is closed by a NAG waste rock embankment on perimeter B (crest 352.4 m, pond 351.4 m), enclosing 124.8 ha and storing the 6.2 Mt of life-of-mine flotation tailings.
- The PEA life-of-mine flotation tailings of 6.2 Mt are stored within this enclosure. Underground (paste) backfill remains a secondary deposition option and is not required for the PEA tailings balance.

The model carries US$0.25M for tailings design and US$6.5M for tailings construction (US$2.5M initial in Year -1, with the balance as sustaining capital), plus ongoing tailings construction at 5% within sustaining capital. Continued use of Ragged Lake requires a Schedule 2 listing under the federal Metal and Diamond Mining Effluent Regulations (MDMER).

The QP for Recovery Methods estimates US$2.75 million for the new lined sulphide tailings facility and US$9.0 million for the Ragged Lake upgrade (US$0.5 million for the phase 1 weir, US$4.5 million for the phase 2 weir and US$4.0 million for lining and placement), against US$6.75 million in the model (US$0.25 million design, US$2.5 million initial construction and US$4.0 million sustaining). The base case uses the model values.

Ragged Lake TMA embankment design basis. Manitoba has no stand-alone dam safety legislation, and Manitoba mining projects (for example, the Lynn Lake Gold Project) design tailings dams to the Canadian Dam Association (CDA) Dam Safety Guidelines (2007, 2013 edition) and the CDA Technical Bulletin Application of Dam Safety Guidelines to Mining Dams (2014, revised 2019), with tailings governance following the Mining Association of Canada (MAC) guide (version 3.2, 2021). The PEA assumes a High consequence classification until a dam-breach and inundation study is completed. For a High classification the inflow design flood is one-third of the way between the 1-in-1,000-year flood and the probable maximum flood, the earthquake design ground motion has an annual exceedance probability of 1 in 2,475, and the minimum factors of safety are 1.3 at the end of construction, 1.5 for long-term steady seepage, 1.0 pseudo-static and 1.2 post-earthquake.

The conceptual embankment is a NAG waste rock dam with a 10 m crest, a 3H:1V upstream slope lined with an HDPE geomembrane on bedding and transition zones (or a low-permeability core if suitable till is available) and a 2H:1V downstream slope, founded on bedrock after removal of organics and soft soils. At least 1.0 m of freeboard is kept above the maximum operating pond, and raises are by the downstream or centreline method only. A pond level of 351.4 m gives a crest of 352.4 m. On public terrain data (Natural Resources Canada MRDEM-30, CGVD2013), the present lake surface is at about 345.0 m to 345.5 m, about 0.5 m below the PEA weir level of 346 m, which is within the accuracy of the data. Along the perimeter B alignment in Figure 18-1 (4,459 m long, enclosing 124.8 ha), a wall of 3,380 m is needed, up to 7.36 m high, mainly across the low ground between the pits and the lake, the north drainage valley and the south weir. This needs 237,186 m^3 of placed NAG rock (579,558 t, about 0.58 Mt), taken from the north WRD.

The enclosure stores the 6.2 Mt of life-of-mine flotation tailings at a pond level of 351.4 m (pond area 114.8 ha). The Ragged Lake TMA, sized for the 6.2 Mt PEA life-of-mine tailings, will be advanced at the PFS stage. That work covers bathymetry and existing tailings levels, a lidar or ground survey, a foundation investigation, staging, the water balance and stability, a dam-breach study and consequence classification, and the sulphide TMA liner design. The 10 Mt in the Intent to Apply allows for possible future mine life beyond the PEA.

The dams need a licence for water control works under The Water Rights Act. Under the Operation of Mines Regulation (M.R. 212/2011), section 3.10, the plans, elevations, retained volumes, a professional engineer's report (site investigation, stability, seepage and closure) and a risk assessment must be given to the mines inspector at least 60 days before a new dam or a major modification is built, and section 3.11 requires an engineer-approved quality assurance and quality control program during construction. The Mine Closure Regulation (M.R. 67/99) requires the closure plan to leave the tailings dams and waste rock piles stable without ongoing maintenance. The TMA expansion and dams are part of the Notice of Alteration to EAL 1207E (Item 20).

The PEA specifies non-acid-generating (NAG) waste rock for all embankment construction, consistent with the ARD findings in Item 20, rather than the potentially acid-generating (PAG) rock proposed in the 2026 draft project description. The selected rock will be confirmed by acid-base accounting at the PFS stage.

18.7 Waste Rock, Mineralized Material and Soil Stockpiles

The PEA mine plan (Item 16) produces 36.9 Mt of open-pit waste and 3.4 Mt of underground waste. At an in-situ density of 2.81 t/m³ and a swell factor of 25%, this is about 17.9 Mm³ of loose rock, of which 257,810 m³ loose (237,186 m³ placed, 579,558 t, or about 0.58 Mt) is used in the Ragged Lake TMA embankment. The 2026 draft project description (AECOM, draft 2026) gives about 76.2 Mt of open-pit waste and 3.1 Mt of underground waste, with two waste rock management areas of about 38 ha and 28.5 ha, up to 20 m high, south of the pits; these figures predate the PEA mine plan and are replaced in this PEA by the two waste rock dumps described below. Other planned stockpiles are a run-of-mine (ROM) pad north of the plant sized for about two months of feed, and a 13.5 ha topsoil and organics stockpile for reclamation (Figure 18-1). The 2026 draft project description gives a total new footprint of about 643 ha, excluding 19 ha of existing roads and disturbed areas. The PEA waste rock dumps described below have a combined toe of about 80.5 ha. The total project footprint will be confirmed at the PFS stage.

Waste rock dumps. Waste rock is placed in two WRDs (Figure 18-1), both designed to a height of 30 m in three 10 m lifts, with lift faces at the angle of repose (about 1.4H:1V) and 24 m berms, giving an overall slope of 3H:1V, and with a 100 m setback from lakes where it applies.

The south-west WRD sits on ML065, with a toe of about 23.1 ha (about 340 m by 680 m). It has a capacity of 4.53 Mm³ (loose), or 10.19 Mt at 2.81 t/m³ and a swell of 1.25. It takes all of the south-east pit waste (9.10 Mt) and about 0.90 Mt from the north-east pit.

The north WRD sits on claim MIT-045 (MB12260), with a toe of about 57.4 ha and a capacity of 13.27 Mm³ (loose), or 29.74 Mt. It takes the north-west pit waste (25.88 Mt), the remaining 1.03 Mt from the north-east pit and the 3.42 Mt of underground waste, less the 0.58 Mt used in the TMA embankment. The toe is 100.6 m from the nearest lake, about 128 m from ML065, about 418 m from the process plant and about 185 m from the ROM pad. MIT-045 is a mineral claim and carries no surface rights, so the north WRD needs Crown surface rights (a Crown land permit or lease) and must be included in the Notice of Alteration.

The split reduces the weighted one-way haul for open-pit waste by about 35%, from 1,522 m to 983 m. Potentially acid-generating (PAG) rock (about 25% of the open-pit waste samples; Item 20) will be placed in separate cells with seepage collection. The south-west WRD covers about 23.1 ha and holds about 10.19 Mt, and the north WRD about 57.4 ha holding about 29.74 Mt, about 80.5 ha in total. The south-west WRD location, the foundation and drainage of both WRDs, any PAG cell liner, and Crown tenure for the north WRD will be confirmed at the PFS stage.

18.8 Surface Buildings

The existing office complex next to the plant (mine office, dry and change house with a lamp room and proto room, and assay laboratory office) will be refurbished. The 2018 FS identified the work needed: replacement of roll-up doors, repairs to heating, refurbishment of the overhead crane, pallet racking and a secure store in the warehouse, rebuilding of the office interiors, HVAC, electrical and fire protection, water reticulation, and complete re-equipping of the assay laboratory. A new 3.8 ha contractor laydown area will be established

south of the North Pit. The model carries US$1.0M for refurbishment of the on-site laboratory, US$0.2M for fencing and security, US$0.85M for underground contractor site establishment and US$0.28M for open-pit site establishment. Plant electrical and mechanical refurbishment and upgrades are covered in Item 17. The building condition survey and refurbishment scope will be completed at the PFS stage.

18.9 Camp and Accommodation

A temporary mobile work camp for about 70 workers, with kitchen, dining and car wash, will be installed on previously disturbed land next to the plant in Year 1 and will use the existing septic system. A new 120-worker lodge will then be built about 1.4 km west (by road) of the plant, near Jay Lake. It will have a central building (kitchen, dining, administration, utilities and recreation) on concrete foundations, two two-storey modular dormitories, a dedicated wastewater treatment facility and a water conditioning plant. The model carries US$4.0M for camp construction (US$0.3M in Year -2, US$1.7M in Year -1 and US$2.0M in Year 1), and lodge operating costs based on an occupancy of 100 persons at US$90 per person-day (US$3.3M per year). The lodge is planned for 120 beds, as in the 2026 project description. The financial model allows for 100 persons on site, and the extra beds give room for contractors and peak shifts.

18.10 Sewage and Solid Waste

In Year 1, wastewater from the temporary camp will go through the existing Rotodisc/pumphouse and a holding tank to a septic field. The current Environment Act Licence (EAL) requires treated sanitary effluent to be discharged to Ragged Lake. The new lodge will have its own wastewater treatment facility. Domestic and commercial waste will be collected in wildlife-proof bins and removed by a licensed contractor to the licensed facility in Flin Flon. Recyclables will be segregated, and hazardous materials (waste oil and lubricants) removed by a licensed handler. A packaged wastewater treatment plant will serve the lodge, and its design and discharge point will be confirmed at the PFS stage.

18.11 Communications and Security

Site operations will use a two-way VHF/UHF land-mobile radio system (AECOM, draft 2026). The 2018 FS proposed a rental satellite system for voice, data and internet, because there is no fibre or telephone connection point near the property. Site access is controlled by a security gate on the main access road. A prefabricated building will serve as the security and access control facility at the site entrance. Site communications and data capacity and security staffing will be confirmed at the PFS stage.

18.12 Underground Infrastructure

PEA underground access is through a new portal in the north-east pit (373,663.6 E, 6,100,247.1 N, 292.0 m RL). The historical portal and decline will be dewatered and rehabilitated for the recommended trial mine, which is a PFS recommendation and is not in the PEA schedule (Item 16). The portal will have a roadway designed for the haul trucks, with services supported on the sidewall, and a reinforced concrete covering over the entrance. The Umoya VENTSIM study uses a forced ventilation system with four shafts (three downcast

and one upcast). The model carries US$15.0 million for ventilation shafts and fans in 2030 (sustaining capital), which is considered to cover the main fan requirement (benchmark of about US$4.3 million to US$8.6 million installed), and US$1.0 million for refurbishment of the historical underground workings. The three ventilation shafts are planned for 2030. The PFS will confirm whether they need to be brought forward by a year to support the start of underground production in 2029.

Underground services will include process water, potable water and compressed air, as well as any dewatering pipelines that may be needed. The underground electrical reticulation and communication system will also enter through the portal. Modular refuge chambers equipped with essential services will be provided underground (Figure 18-4).

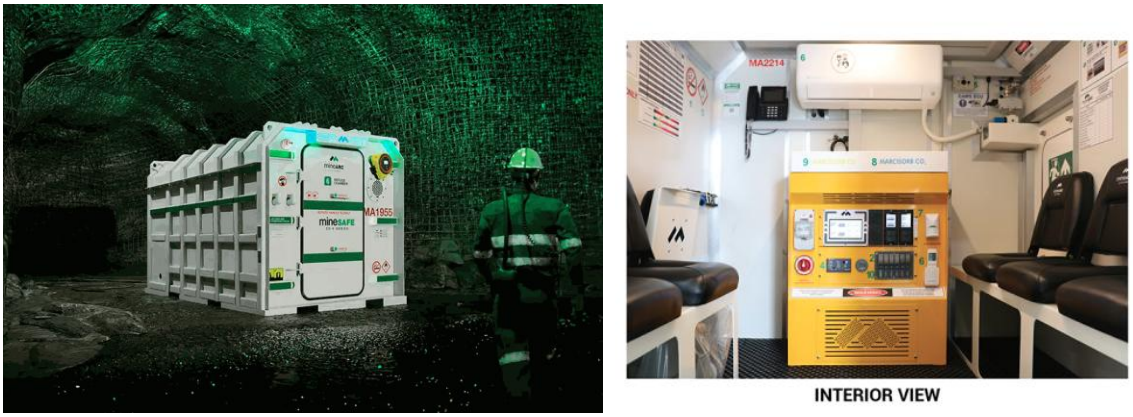


Figure 18-4: Typical portable refuge chamber, exterior and interior

Source: MineARC Systems product images, provided by the engineering consultant (Optimult), November 2026

18.13 Summary of Infrastructure Capital

Table 18-1: Infrastructure-related capital items in the PEA model (US$M, before the 15% contingency on pre-production capital)

Item	Amount (US$M)	Timing
Site and mill electrical refurbishment	10.00	Year -2 to Year -1
Site and mill electrical upgrades (expansion)	4.00	Year 1
Gensets: 1.2 MW / 2 MW / 5 MW / 9 MW	0.30 / 1.75 / 4.16 / 7.43	Year -2 / Year -1 / Year 1 / Year 1
Bulk diesel fuel farm (525 kL)	0.97	Year -1
Road upgrades	0.50	Year -2 to Year -1
Water handling and reticulation	2.00	Year -1
Camp construction	4.00	Year -2 to Year 1
Fencing and security	0.20	Year -2
On-site laboratory refurbishment	1.00	Year -1
Tailings design / construction	0.25 / 6.50	Year -2 / Year -1 onwards
Ventilation shafts and fans	15.00	Year 3 (sustaining)
UG dewatering / treatment and discharge / refurbishment	1.00 / 0.50 / 1.00	Year 1

Source: ABGM cash flow model (2026). The full capital estimate, including contingency and timing, is presented in Item 21.

19 MARKET STUDIES AND CONTRACTS

19.1 Gold Market

Gold is a freely traded commodity with a deep, liquid global market, and the PL Project will produce gold doré. No market study is required for the sale of doré, which is sold to refiners at prices referenced to the London Bullion Market Association (LBMA) Gold Price, set in twice-daily electronic auctions in London at 10:30 and 15:00 (ICE Benchmark Administration, 2026). Gold reached an all-time high of US$5,405/oz in January 2026. In early October 2026 it traded between about US$4,070/oz and US$4,226/oz, with a spot price of about US$4,150/oz on October 14, 2026 (GoldPrice.com, 2026; GramRate, 2026).

For comparison, a major producer in the same district, Hudbay Minerals, used gold prices of US$4,472/oz (2026), US$4,200/oz (2027) and US$4,000/oz (2028), declining to US$3,600/oz long term, in its September 2026 Snow Lake mine plan, and US$3,600/oz for its reserve estimates (Hudbay, 2026). The 2018 FS used a long-term price of US$1,250/oz, based on the three-year and five-year moving averages to September 2017.

19.2 Gold Price Assumptions

The PEA base case uses US$3,000/oz, consistent with the price used for the 2026 Mineral Resource Estimate. It is about 28% below spot at the effective date and below published long-term peer assumptions, and is therefore considered conservative. Two further cases are presented in Item 22: US$3,500/oz and a long-term price of US$4,000/oz. No escalation is applied. Model sensitivities cover ±5% to ±35% around the base price.

Table 19-1: Gold price cases and headline results (after tax)

Case	Gold price (US$/oz)	NPV @ 5% (US$M)	NPV @ 8% (US$M)	IRR (%)
Base case	3,000	180.6	151.4	111
Upside case	3,500	310.8	260.1	152
Long-term price	4,000	438.9	367.2	187

19.3 Product, Refining and Realization Costs

The plant will pour gold doré on site. As is common practice in Canada, the doré will be sold to the Royal Canadian Mint or another accredited refiner. Payment is made on receipt of refiner and referee assays, at prices referenced to the spot market. The model applies a combined realization, freight and marketing charge of US$31.68 per ounce of gold sold and treats all recovered gold (621.5 koz LOM) as payable. Silver and other by-product credits are not included. The refining terms used in the financial model, covering payable gold, treatment and refining charges, insurance and transport, are typical of doré sales and will be confirmed by refinery quotation at the PFS stage. No silver credit is assumed in the PEA.

G. Cunningham notes that at 2,000 t/d doré shipments are expected to be about 2,000 oz to 2,800 oz of doré a week, transported from site by helicopter. Shipment frequency, security and insurance are to be agreed with the refiner.

19.4 Royalties

The PL Property is subject to a sliding-scale net smelter return (NSR) royalty held by Franco-Nevada Corporation, under the NSR Royalty Agreement dated October 8, 2010 and the Royalty Deed and Assignment effective November 1, 2013: 3.0% at gold prices at or above US$1,000/oz, 2.5% from US$750/oz to below US$1,000/oz, and 2.0% below US$750/oz. The cash flow model applies a flat 3% of gross revenue, consistent with the filed MRE cut-off calculation. At the base-case gold price, and at all PEA price cases, the 3.0% rate applies, and applying it to gross revenue rather than NSR is a slightly conservative simplification. The Nokomis Property, which is not part of this PEA, carries an underlying 2% NSR royalty, listed in the 2018 FS as held by Hudbay Minerals Inc. and Placer Dome Inc. (now Barrick). Minnova advises that Hudbay has notified it of its interest, but that the 1% attributed to Placer Dome was never registered on title and has not been confirmed by Barrick. Manitoba mining tax and federal and Manitoba income taxes are described with the economic analysis.

19.5 Contracts

No offtake, streaming, hedging, forward-sale or other sales agreements are in place. Doré is assumed to be sold at spot to a refiner at the model's refining and selling terms (a combined charge of US$31.68 per ounce of gold sold, with all recovered gold payable). The PEA assumes the following contractual arrangements:

- Contract mining: open-pit and underground mining by contractors at US$5.48/t of open-pit waste and US$6.70/t of open-pit mineralized material (from the pit optimization, including diesel), and US$72.44/t of underground waste and US$82.05/t of underground mineralized material. The contractor will provide supervisors, operators and maintenance personnel (up to about 60 pit contractor staff on site at any time) and just-in-time explosives supply.
- LNG supply: the model uses indicative LNG supply terms, priced in C$: AECO gas plus fees of C$2.60/GJ, liquefaction of C$5.50/GJ on gas burned, take-or-pay liquefaction of C$5.50/GJ on the nominated volume, transport to site of C$10.00/GJ, and a facility charge of C$48,000/month (2028 and 2030 onwards) and C$96,000/month (2029). Nominated volumes are 31,209 GJ/month (2028), 62,418 GJ/month (2029) and 25,997 GJ/month (2030 onwards), with an engine efficiency of 36% and no boil-off. LNG supplies all load until grid arrival in Month 37. No LNG supply contract is in place. LNG supply, delivery, storage and on-site generation are assumed at these indicative costs.
- Power generation equipment: genset pricing is based on vendor budget quotations (2 MW, 5 MW and 9 MW units; C$ quotes converted at US$0.74 per C$1.00), with alternative purchase and rental options in the model.

- Grid power: an Electrical Service Agreement with Manitoba Hydro was signed in 2012 (Auriga Gold, now Minnova). Minnova owns the 22 km 25 kV line from Sherridon. In 2026 Minnova applied to Manitoba Hydro to restore service and submitted a large-load request for 9 MVA. Manitoba Hydro has advised that no new commercial load will be connected to the Sherridon station until the 115 kV Snow Lake to Sherridon line, damaged by the 2025 wildfires, is restored, which may take about two years (up to three years), and that the capacity after restoration may be about 3 MW to 3.5 MW, below the Project demand (Item 18). The model assumes grid power from Month 37 at a blended tariff; that timing and capacity are not assured.
- Site services: catering, housekeeping and lodge management (US$90 per person-day in total), waste disposal, and fuel and reagent supply will be contracted.
- Community agreements: the model includes an allowance for community agreements and community support as an operating cost. This is a planning allowance only, and the terms remain to be agreed. No impact benefit agreement (IBA) or other agreement with the Mathias Colomb Cree Nation (MCCN) or the Kiciwapa Cree Nation (KCN) is in place. Minnova keeps a log of its engagement with both communities and will continue that engagement through the PFS and permitting.

In the QP's opinion, the gold price assumptions and the realization and royalty terms are reasonable for a PEA, and the contract assumptions are consistent with industry practice for a small Canadian gold operation.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Regulatory Framework

The PL Project is regulated mainly by the Province of Manitoba. A mine and milling facility is a Class 2 development under the Classes of Development Regulation (M.R. 164/88) and may not be constructed, altered or operated without a licence under The Environment Act (C.C.S.M. c. E125). A licensee who intends to alter a licensed development, so that it no longer conforms to the licence or is likely to change its environmental effects, must notify the Director by a Notice of Alteration (NoA). A major alteration is processed as an Environment Act Proposal (s. 14(3)), and no alteration may proceed until it is approved (s. 14(5)) (Manitoba, 2024).

The Mine Closure Regulation (M.R. 67/99) under The Mines and Minerals Act requires a closure plan with a certified schedule of closure costs and financial security, and notice of any expansion or alteration at least 30 days in advance with a revised closure plan. Federally, continued tailings disposal in Ragged Lake requires its listing on Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER, formerly MMER) under the Fisheries Act, together with an approved fish-habitat compensation plan and an irrevocable letter of credit. Effluent must meet the MDMER limits and Environmental Effects Monitoring (EEM) requirements. At

2,000 t/d the Project is below the 5,000 t/d ore-production threshold for a new or expanded metal mine under the federal Physical Activities Regulations (SOR/2019-285), so it is not expected to be a designated project under the Impact Assessment Act.

20.2 Environmental Licence and Current Permitting Status

The mine operated from late 1987 to April 1989 under Environment Act Licence (EAL) No. 1207E. Manitoba confirmed in May 2012 that the licence remained in good standing for a restart, subject to compliance with the federal effluent regulations. The current licence authorizes underground mining. The open pits, the expansion to 2,000 t/d, the TMA expansion and dams, the lined Sulphide TMA, the north waste rock dump on claim MIT-045, the south-west waste rock dump on ML065 and the new lodge require an alteration to EAL 1207E. Minnova has filed an Environment Act Intent to Apply form for a major NoA for a Class 2 open-pit and underground mine and milling facility, with an intended Environment Act Proposal submission date of July 16, 2026 (Minnova, 2026). AECOM is preparing the environmental impact assessment and the project description. The Project schedule assumes that a revised EAL is issued by early 2027 (Minnova, 2026 schedule). At October 24, 2026, Minnova reported that EAL 1207E is in place and that the Intent to Apply for the major NoA has been filed. The MDMER Schedule 2 listing application for the Ragged Lake TMA was first submitted to DFO in May 2017. AECOM has completed a gap analysis of the earlier Alternatives Assessment report and will update it for the development plan, and Minnova is targeting filing by the end of 2026. In October 2026 Minnova applied for casual quarry permits for overburden aggregate and for a hard-rock quarry on the mining lease within the Pit 1 outline (G. Glenn, email of October 22, 2026). The status of each permit, as reported by Minnova, is summarized in Table 20-1.

Table 20-1: Permits and approvals: status as reported by Minnova Corp. to the QP, updated to October 24, 2026

Permit / approval	Authority	Status / comment
Environment Act Licence 1207E (underground mine, mill, Ragged Lake TMA)	Manitoba Environmental Approvals Branch	In place. Minnova has filed an Environment Act Intent to Apply form for a major Notice of Alteration. The major alteration (NoA / Environment Act Proposal) is required for the open pits, 2,000 t/d, TMA expansion and new facilities; revised EAL assumed early 2027
Mining Lease ML065 (378 ha)	Manitoba Mines Branch	In good standing. Maintained by an annual rental payment due on April 1 each year; current anniversary date April 1, 2027
Crown land permits GP0002799, GP0003758, GP0004038, GP0004134 (roads, water pipe right of way, tailings dam)	Manitoba	Renewed annually; current permits expire December 31, 2026
Water Rights Act licence 2017-116 (industrial water)	Manitoba Water Stewardship	Issued September 2017; expires September 5, 2027; renewal and increase for 2,000 t/d and lodge supply from Jay Lake required
Water Rights Act drainage licence (Fire Pond dewatering)	Manitoba	Required unless authorized in the revised EAL
MDMER Schedule 2 listing of Ragged Lake TMA, with fish-habitat compensation plan	Environment and Climate Change Canada / DFO	Application first submitted to DFO in May 2017. AECOM has completed a gap analysis of the earlier Alternatives Assessment report and will update it for the development plan; Minnova is targeting filing by the end of 2026
Closure plan and financial assurance (Mine Closure Regulation)	Manitoba Mines Branch	C$75,000 letter of credit provided to the Government of Manitoba under the closure plan, and site assets pledged as security of up to C$5,000,000. Closure and reclamation provision of C$3,718,583 at June 30, 2026. Revised closure plan required for the expansion
Casual quarry permit / quarry dispositions (borrow areas)	Manitoba	Required for about 40,000 t of rock and 40,000 t of sand and gravel. In October 2026 Minnova applied for casual quarry permits for overburden aggregate and for a hard-rock quarry on the mining lease within the Pit 1 outline
Other operating permits (septic, fuel storage, explosives, work permits)	Various	To be obtained as required

Source: Minnova Corp. (2026); 2018 FS; AECOM, draft 2026.

The model carries US$1.5M for approvals (NoA, water and regulatory; US$0.25M in Year -2 and US$1.25M in Year -1).

20.3 Environmental Baseline Studies

Approval of the original licence was supported by baseline terrestrial and aquatic studies in May 1987 and a 1987 Environmental Impact Statement by Ilam Associates. Re-activation studies completed by Minnova include an AECOM environmental baseline assessment (May 2014), the WESA preliminary assessment of pit inflows and water quality (December 2011), the Parks Environmental ARD/ML study (May 2012), the North/South Consultants status report on the Ragged TMA (December 2011) and an AECOM fish population study (August 2014). The 2026 environmental impact assessment by AECOM is in progress. Wetland boundaries in the project footprint have been approximated from satellite imagery. Vegetation clearing is to avoid the breeding-bird season (25 April to 12 August). An updated environmental baseline program will be carried out to support the PFS and permitting. It will cover surface water, groundwater, aquatics, wetlands, wildlife, species at risk, air, noise and a GHG inventory.

20.4 Existing Environmental Liabilities

The site has already been affected by the historical operation: the portal, ramp and more than 7,000 m of underground workings, the 1,000 t/d plant, historical waste rock, and more than 350,000 t of tailings in the Ragged Lake TMA. Minnova has provided a letter of credit of C$75,000 to the Government of Manitoba under the closure plan, and has pledged all assets, goods and personal property involved in the operation of the PL Property as security of up to C$5,000,000 for the performance of the closure plan and the rehabilitation program. Its closure and reclamation provision was C$3,718,583 at June 30, 2026 (Minnova, 2026), compared with C$2,900,324 at September 30, 2017 (future liability of about C$3,073,160) (2018 FS). A revised closure plan, cost estimate and financial assurance for the expanded Project are required (see Mine Closure and Reclamation).

20.5 Geochemistry: Acid Rock Drainage and Metal Leaching

Previous studies (Parks 2012; AECOM 2014) concluded that:

- About 25% of open-pit waste-rock samples are potentially acid-generating (PAG), with a neutralization potential to acid potential ratio (NP/AP) below 2, and arsenic was the most mobile contaminant in humidity-cell tests.
- Exposed pit walls that contain residual mineralization, sulphide-bearing mineralized material stockpiles and tailings are likely to generate ARD/ML. A water cover should be kept over the tailings, and TMA water should be treated before reuse or discharge.
- Historical waste rock and tailings are likely to generate ARD/ML and will be managed with the new material.

The Project will segregate PAG and NAG waste rock and will store the arsenic-bearing sulphide-concentrate tailings in the lined Sulphide TMA. The 2026 project description proposes using PAG rock for road and tailings-dam upgrades. That proposal conflicts with the ARD findings and must be reviewed; NAG rock should be

specified for construction. ARD/ML testing of bench-scale tailings from the upgraded flowsheet, and confirmation of the material specification, will be completed at the PFS stage.

20.6 Tailings, Waste Rock and Water Management

The tailings, waste rock and water management concepts are described in Item 18. In summary: flotation tailings go to an expanded Ragged Lake TMA (perimeter B, enclosing 124.8 ha, crest 352.4 m) and sulphide tailings to a lined Sulphide TMA outside the embankment; waste rock goes to a south-west WRD on ML065 and a north WRD on claim MIT-045 (Figure 18-1); contact water and runoff report to the Ragged Lake TMA; and treated effluent is discharged at a single final discharge point that meets the EAL and MDMER limits. The MDMER authorized limits (monthly mean) include arsenic 0.50 mg/L, copper 0.30 mg/L, cyanide 1.00 mg/L, lead 0.20 mg/L, nickel 0.50 mg/L, zinc 0.50 mg/L, total suspended solids 15 mg/L and radium-226 0.37 Bq/L, with effluent pH 6.0 to 9.5 and non-acutely lethal to rainbow trout (2018 FS; MDMER Schedule 4). The effluent limits will be confirmed against the current Metal and Diamond Mining Effluent Regulations (MDMER) schedule, including the un-ionized ammonia limit, during ETP design.

20.7 Social and Community

The Project lies entirely on provincial Crown land, mostly within Mining Lease ML065. Access to the site is controlled for public safety. The nearest communities are Sherridon and Cold Lake (a Métis community administered under The Northern Affairs Act, about 12 km northwest) and the Mathias Colomb Cree Nation (MCCN) at Pukatawagan, about 73 km north of Sherridon. Other First Nations at greater distances include the Opaskwayak Cree Nation (86 km), Nisichawayasihk Cree Nation (150 km), Mosakahiken Cree Nation (150 km), Cross Lake First Nation (195 km) and Norway House Cree Nation (240 km) (AECOM, 2014). Minnova has engaged with MCCN since 2011 and with the Kiciwapa Cree Nation (KCN) since July 2024. Its engagement plan includes technical briefings (Minnova, 2026). The Crown's duty to consult will apply to the provincial licensing decisions.

The PEA cash flow includes an allowance for community agreements and community support as an operating cost. This is a planning allowance only, and the terms remain to be agreed with the communities. Contractor mining is planned to supply most of the operational workforce, with up to about 60 pit contractor employees on site at any time and about 40 further personnel visiting each day (AECOM, draft 2026). The 2018 FS expected up to 50% of the workforce to come from the region. No impact benefit agreement (IBA) or other agreement with the MCCN or the KCN is in place. Minnova keeps a log of its engagement with both communities and will continue that engagement through the PFS and permitting. Minnova intends to prioritize local and Indigenous hiring. Targets will be set with the communities as engagement progresses.

20.8 Mine Closure and Reclamation

Closure will remove the surface infrastructure (mill, buildings, underground systems and services), rehabilitate the open pits progressively each year during mining, and use the stockpiled topsoil and organics for revegetation. The TMA will be closed with a water cover, and the effluent treatment plant (ETP) will

operate until contact water from the TMA and waste rock meets direct-discharge criteria. Post-closure monitoring will be completed as required by the Mines Branch and the revised EAL. Under the Mine Closure Regulation a revised closure plan, with a certified cost schedule and financial security, will be filed for the expanded Project. A US$1.0M closure allowance is carried as an input not yet linked into the cash flow model, to be refined when the closure plan is completed. The cash flow includes closure capital of US$2.6M in Year 11 (2038). The closure capital in the financial model is a PEA-level estimate. A costed closure plan, covering the about 643 ha of new disturbance, the pits, the WRDs and the expanded TMA, will be prepared for the PFS and the Manitoba closure plan submission, and the closure cost may increase.

20.9 Greenhouse Gas and Other Environmental Considerations

Power will initially come from diesel generators, followed by LNG/CNG generation, with the aim of running the Project entirely on Manitoba Hydro grid power once it becomes available (assumed from Month 37). This sequence lowers GHG intensity over the mine life. Manitoba Hydro's advice on the Sherridon supply (Item 18) means that grid power may not be available at the Project demand, in which case LNG generation, and its emissions, would continue for longer. A GHG inventory is required for the Environment Act submission. A GHG inventory, a fire protection and wildfire response plan, and a noise and dust assessment will be prepared at the PFS stage.

20.10 Environmental and Permitting Risks

- The timing of the revised EAL (assumed early 2027) and of the MDMER Schedule 2 listing for Ragged Lake determines the start of open-pit mining in 2028.
- The water licence 2017-116 expires on September 5, 2027 and must be renewed and increased before production.
- Capacity and design of the Ragged Lake TMA: the PEA stores 6.2 Mt of life-of-mine flotation tailings at a pond level of 351.4 m (crest 352.4 m, enclosing 124.8 ha); the 10 Mt in the Intent to Apply allows for possible future mine life beyond the PEA; dam foundations and consequence classification, and the effluent treatment needed to meet the discharge limits, remain to be confirmed.
- Crown tenure for the north WRD on claim MIT-045, outside ML065, and Notice of Alteration coverage for both WRDs, will be confirmed at the PFS stage.
- Closure liability and financial assurance for the expanded footprint, which may exceed the model allowances.
- Agreements with MCCN and KCN, and the Crown consultation process.

21 CAPITAL AND OPERATING COSTS

This Item summarizes the capital and operating cost estimates used in the preliminary economic assessment (PEA) cash flow, the basis on which they were prepared and their expected accuracy. All costs are in real (unescalated) United States dollars (US$). The project re-uses and refurbishes the existing PL mill and site infrastructure, which is why pre-production capital is low relative to the scale of production. All values are taken from the PEA cash flow model (the model), and the source of each table is given below it. **The cost base date of the estimate is Q3 2026.**

21.1 Capital Cost Estimate

21.1.1 Basis of estimate

The capital estimate is a line-item schedule in the model, phased by year. Each line is classified as Initial, Expense (pre-production costs that do not create a depreciable asset), Sustaining or Closure. The sources are:

- Owner and consultant allowances for mill electrical and mechanical refurbishment, tailings, water handling, camp, roads, laboratory, approvals and studies, entered directly in US$.
- Vendor budget prices in Canadian dollars (C$) for the generator set (genset) packages (2 MW, 5 MW and 9 MW) and the 525 kL bulk diesel fuel farm, converted at an exchange rate of US$0.74 per C$1.00.
- Contractor open-pit (OP) site establishment and disestablishment (C$375,000 and C$125,000 and C49) and an underground (UG) contractor establishment allowance of US$0.85M.
- Open-pit pre-strip: the contractor cost of OP waste mined in the first four months of 2028 (US$9.42M at the pit-optimization waste rate of US$5.48/t, which includes diesel) is capitalized as Initial capital and removed from operating cost.
- Mining equipment is supplied by the mining contractors and recovered through contractor rates; no owner mining fleet capital is included.

21.1.2 Estimate class and accuracy

The estimate is a PEA-level (scoping) estimate based on factored allowances and budget prices, with limited engineering. Consistent with industry practice for a PEA, the expected accuracy is in the range of approximately ±30% to ±50%, broadly equivalent to an Association for the Advancement of Cost Engineering (AACE) International Class 5 estimate. No tenders have been obtained.

21.1.3 Capital cost summary

Total life-of-mine (LOM) capital, including sustaining capital and closure, is US$120.9M. Pre-production capital in 2026 and 2027 is US$53.4M, including a 15% contingency of US$7.0M. Total initial (non-sustaining) capital is US$55.4M, which adds the US$2.0M balance of camp construction in 2028. Sustaining capital is US$62.9M,

made up of named sustaining items of US$47.9M (mainly the 2028 mill upgrade, gensets and UG start-up, and the 2030 ventilation shafts and fans) and a general sustaining allowance of US$15.0M. Closure totals US$2.6M.

Table 21-1: Capital cost summary by class and period (US$M)

Class	2026	2027	2028	2029 to 2037	2038	LOM
Initial: direct (including pre-strip)	2.0	38.9	2.0	-	-	42.9
Initial: capital contingency (15%)	-	7.0	-	-	-	7.0
Pre-production costs (Expense class)	2.0	3.5	-	-	-	5.5
Total initial (non-sustaining) capital	**4.0**	**49.3**	**2.0**	**-**	**-**	**55.4**
Sustaining: named items	-	-	29.9	18.0	-	47.9
Sustaining: 3% annual allowance	-	-	1.5	13.5	-	15.0
Total sustaining capital	**-**	**-**	**31.4**	**31.5**	**-**	**62.9**
Closure and disestablishment	-	-	-	0.1	2.5	2.6
Total capital	**4.0**	**49.3**	**33.4**	**31.6**	**2.5**	**120.9**

Source: ABGM cash flow model (2026). Expense-class pre-production costs (studies, owner's costs and engineering, procurement and construction management (EPCM), approvals, metallurgical testing) are non-sustaining capital in the cash flow and all-in cost but are not depreciated. Totals may not add because of rounding.

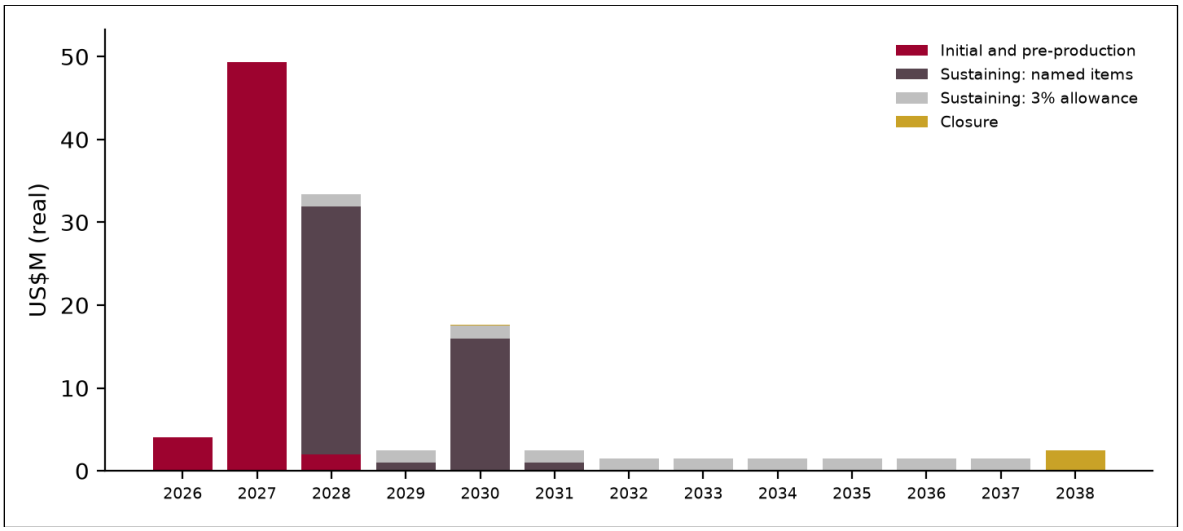


Figure 21-1: Capital expenditure by year and class (US$M)

Source: ABGM cash flow model (2026).

21.1.4 Initial capital

Table 21-2: Initial (non-sustaining) capital by line item (US$M)

Item	2026	2027	2028	Total
Initial capital (direct and contingency)				
Capital contingency (15% of 2026 and 2027 capital)	-	6.96	-	6.96
Tailings Construction (Initial)	-	2.50	-	2.50
Site Plus Mill Electrical Refurbishment	1.00	9.00	-	10.00
Mill Mechanical Refurbishment	-	10.00	-	10.00
1.2 MW test genset	0.30	-	-	0.30
Road Upgrades	0.25	0.25	-	0.50

Item	2026	2027	2028	Total
Water Handling & Reticulation	-	2.00	-	2.00
Camp Construction (Contractors and Owners Team)	0.30	1.70	2.00	4.00
Fencing and Security	0.20	-	-	0.20
On-site laboratory (refurbishment)	-	1.00	-	1.00
OP contractor site establishment (2,000 t/d)	-	0.28	-	0.28
OP pre-strip (capitalized)	-	9.42	-	9.42
2 MW genset	-	1.75	-	1.75
Bulk diesel fuel farm (7 x 75,000 L double-walled, 525 kL)	-	0.97	-	0.97
Pre-production costs (Expense class)				
Exploration & Technical Studies	0.50	-	-	0.50
Owner's costs and EPCM (pre-production)	0.50	2.00	-	2.50
Tailings Design	0.25	-	-	0.25
Metallurgical Testing	0.25	-	-	0.25
Metallurgical testing: ore sorting	0.25	0.25	-	0.50
Approvals (NOA, Water, Regulatory, Mining)	0.25	1.25	-	1.50
Total initial (non-sustaining) capital	**4.05**	**49.33**	**2.00**	**55.38**

Source: ABGM cash flow model (2026). Contingency is 15% of all 2026 and 2027 capital, including Expense-class items. Tailings construction is split into US$2.5M initial (2027) and US$4.0M sustaining (2028 to 2031). Ore-sorting test work (US$0.5M) is retained as a study cost only; the base case assumes no ore sorting and 90% recovery.

21.1.5 Sustaining capital

Named sustaining items total US$47.9M and are concentrated in 2028 (US$29.9M: 5 MW and 9 MW gensets for site and mill power until grid connection, mill upgrade with tertiary crushing and gravity, site electrical expansion, UG establishment, dewatering and refurbishment) and 2030 (US$15.0M for ventilation shafts and fans). Tailings lifts of US$1.0M a year are scheduled in 2028 to 2031. The model also carries a general sustaining allowance of 3% a year of the depreciable initial capital base (US$49.9M × 3% = US$1.50M a year) in each production year, US$15.0M over the LOM, for equipment replacement and minor projects not yet engineered.

Table 21-3: Sustaining capital by line item (US$M)

Item	2028	2029	2030	2031	2032 to 2037	Total
Tailings Construction (sustaining)	1.0	1.0	1.0	1.0	-	4.0
Site Plus Mill Electrical Upgrades Expansion	4.0	-	-	-	-	4.0
Mill Upgrade (Tertiary Crusher, Mills, Gravity etc.)	10.0	-	-	-	-	10.0
Vent Shafts and Fans	-	-	15.0	-	-	15.0
UG contractor site establishment	0.8	-	-	-	-	0.8
UG dewatering (pumps and electrical)	1.0	-	-	-	-	1.0
UG water treatment and discharge	0.5	-	-	-	-	0.5
UG refurbishment	1.0	-	-	-	-	1.0
5 MW genset	4.2	-	-	-	-	4.2
9 MW genset	7.4	-	-	-	-	7.4
Subtotal named sustaining items	**29.9**	**1.0**	**16.0**	**1.0**	**-**	**47.9**
Sustaining allowance (3% of depreciable initial base)	1.5	1.5	1.5	1.5	9.0	15.0
Total sustaining capital	**31.4**	**2.5**	**17.5**	**2.5**	**9.0**	**62.9**

Source: ABGM cash flow model (2026). No contingency is applied to sustaining capital.

21.1.6 Closure and reclamation

Closure is modelled as US$2.49M in 2038 (5% of the depreciable initial capital base) plus US$0.09M for OP contractor disestablishment in 2030, a total of US$2.59M. For all-in sustaining cost (AISC) the closure amount is accrued over ounces sold. This provision is low for a combined open-pit, underground and tailings operation and is not yet supported by a closure plan; the model also contains unlinked inputs for a US$1.0M upfront closure provision and a US$7.0M closure accrual. The closure estimate should be replaced by a costed closure plan consistent with the Manitoba mine closure requirements and the required financial assurance.

21.1.7 Contingency

A contingency of 15% is applied to all pre-production (2026 and 2027) capital, US$6.96M, and is included in the depreciable initial base. Sustaining items and closure carry no contingency. A 5% contingency on mining and electrical costs is carried separately in operating costs. For a PEA a contingency of 20% to 30% on direct capital is more typical, given the limited engineering behind the refurbishment allowances. The sensitivity analysis shows the effect of a ±10% and ±20% change in all capital. Some vendor and consultant estimates already include their own contingency.

The QP for Recovery Methods considers that the 15% contingency should not be applied to the US$20.0M refurbishment to 1,000 t/d, because that estimate already includes a US$5.0M contingency. This is shown as a cost difference in Table 21-4; the base case is not changed.

21.1.8 Exclusions

The capital estimate excludes the following:

- Escalation and inflation beyond the cost base date (the model is in real terms).

- Financing costs, interest during construction and hedging (the analysis is unlevered, 100% equity).
- Working capital (no receivables, inventories or payables are modelled) and any salvage value at closure.
- Sunk costs incurred before 2026, and corporate overheads outside the site.
- Owner mining fleet: mining equipment is provided by the contractors and recovered in the contractor rates.
- UG capital development as a separate capital item: UG waste development is costed through the UG contractor waste rate in operating cost.
- Grid connection capital (the grid tariff is applied from Month 37, 2031, with no connection cost in the capital schedule).
- Mine-air heating
- Backfill (to be assessed in PFS mining method trade-off studies)
- Diesel particulate filters, recommended by the VENTSIM study for the forced downcast system.
- Lake and pond movement works (input set to zero pending scope), goods and services taxes, and currency movements.

21.1.9 Process Plant and Tailings Costs: QP Comment

The Recovery Methods estimates by G. Cunningham for the process plant and tailings facilities differ from the allowances carried in the model. G. Cunningham commented on the differences on October 29, 2026. His position is compared with the model in Table 21-4. The model numbers, and therefore the published base case, have not been changed: G. Cunningham agreed on October 29, 2026 that the base case remains as published and that his estimates are tested as a sensitivity.

Table 21-4: Recovery Methods cost estimates compared with the model (US$M unless stated)

Item	G. Cunningham position (October 29, 2026)	Cash flow model (base case)	Difference and comment
Refurbishment to 1,000 t/d, excluding tailings	US$20.00M including a US$5.00M contingency; the 15% model contingency on this item to be removed to avoid double counting	US$20.00M (electrical US$10.00M and mechanical US$10.00M) plus the 15% contingency on pre-production capital (US$3.00M on these items)	About US$3.0M lower if the 15% contingency on the refurbishment is removed. Not applied.
New sulphide tailings facility (1,000 t/d)	US$2.75M (design US$0.25M, construction US$2.50M)	US$2.50M construction and US$0.25M design	Agrees.
Expansion to 2,000 t/d, excluding tailings	US$16.8M plus US$3.2M contingency (US$20.0M)	US$14.00M (mill upgrade US$10.00M and site and mill electrical expansion US$4.00M), no contingency	About US$6.0M higher, including contingency. Not applied in the base case; applied in sensitivities C2 and C3.
Tailings facility upgrade (Ragged Lake weir phases 1 and 2, lining)	US$9.00M preferred	US$4.00M sustaining (US$1.0M a year, 2028 to 2031)	US$5.0M higher. Not applied.
Plant operating cost	US$22.18/t excluding power; with power at US$0.06/kWh, US$38/t at 1,000 t/d and US$27/t at 2,000 t/d	US$22.16/t excluding power plus mill power; LOM average US$28.13/t milled	Consistent (QP comment).
Plant power	50 to 55 kWh/t for processing only; US$0.06/kWh grid cost current and acceptable	75 kWh/t for the mill, including HVAC and offices; grid US$0.060/kWh from Month 37	Consistent (QP comment).

Source: G. Cunningham, Recovery Methods draft and comments of October 29, 2026; ABGM cash flow model (2026).

The QP's figures were run through the model as sensitivity C2 (Item 22): the 2,000 t/d plant at US$20.0M including a US$3.155M contingency, the tailings facility at US$9.0M, and the 15% model contingency removed from the refurbishment and expansion. On that basis after-tax NPV at 5% is US$176.1M, US$4.5M below the base case, and the after-tax IRR is 90.1%. Opex and power are unchanged.

21.2 Operating Cost Estimate

21.2.1 Basis of estimate

Operating costs are built month by month in the model from the production schedule and unit rates, and summed to years. OP and UG mining are by contractors at unit rates per tonne. The owner operates the refurbished PL mill and supplies power and UG diesel. The LOM operating cost is US$1,280.8M, equal to US$197.08/t milled and US$2,061/oz recovered. Table 21-5 lists the principal inputs.

Table 21-5: Principal operating cost inputs

Input	Value
Mining (contractor)	
OP waste (pit optimization rate, including diesel)	US$5.48/t
OP mineralized material including high-grade and low-grade stockpile material (including diesel)	US$6.70/t

Input	Value
UG waste development (excluding diesel)	US$72.44/t
UG mineralized material (excluding diesel)	US$82.05/t
Processing and general and administration (G&A)	
Processing, all-in including power	US$28.00/t milled
Processing excluding power (applied)	US$22.16/t milled
Mill power, 75 kWh/t at blended US$0.078/kWh	US$5.84/t milled
Site G&A (includes owner management)	US$10.60/t milled
Owner senior management; junior management and staff	US$1.5M + US$1.5M a year
Lodge (catering, housekeeping, utilities, administration)	US$90 per person-day, 100 people
Environmental monitoring; safety	US$0.50/t; US$0.50/t
Energy	
Diesel price, UG fleet (delivered)	US$2.10/L
Power, liquefied natural gas (LNG) period (Months 1 to 36, 2028 to 2030)	LNG fuel US$0.120/kWh plus take-or-pay and facility charges
Power, grid (from Month 37, 2031)	US$0.060/kWh
Power consumption factors (kWh per tonne of run-of-mine (ROM) mineralized material)	Mill 75; OP 10; UG 60; other site 25; lodge 15
Other	
Contingency on mining and electrical costs	5%
Community agreements and support	Planning allowance in operating cost (US$12.8M LOM)
Royalty (deducted from revenue, not operating cost)	3% of gross revenue, as modelled (3.0% NSR rate applies at US$1,000/oz or more; slightly conservative)
Refining, transport and selling (deducted from revenue)	US$31.68/oz

Source: ABGM cash flow model (2026). The OP power factor used in the model is 10 kWh/t.

21.2.2 Operating cost by area

Table 21-6 splits the LOM operating cost into direct OP and UG costs (mining contractor, diesel and power for each area, and the 5% contingency on them), processing (including mill power) and shared site costs. Shared costs are not allocated to the mining areas in the headline split; a pro rata memo allocation is given in Table 21-7. UG mining dominates the cost structure: the UG area accounts for 57% of LOM operating cost and the mining contractor rates alone for 66%.

Table 21-6: LOM operating cost by area

Cost area	US$M	% of total	US$/t milled	US$/oz
Open pit (direct)				
Mining contractor, including diesel (net of capitalized pre-strip)	204.6	16.0%	31.48	329
Power	2.1	0.2%	0.33	3
Contingency (5%)	10.3	0.8%	1.59	17
Total open pit	**217.1**	**16.9%**	**33.40**	**349**
Underground (direct)				
Mining contractor	636.5	49.7%	97.94	1,024
Diesel, UG fleet	40.2	3.1%	6.19	65
Power	17.9	1.4%	2.75	29

Cost area	US$M	% of total	US$/t milled	US$/oz
Contingency (5%)	31.9	2.5%	4.91	51
Total underground	**726.5**	**56.7%**	**111.79**	**1,169**
Processing and shared site				
Processing (rate excluding power, plus mill power)	182.8	14.3%	28.13	294
Site G&A and owner management	70.3	5.5%	10.81	113
Site power (other site, lodge), LNG fixed charges, contingency	26.5	2.1%	4.08	43
Lodge / camp	38.4	3.0%	5.91	62
Environmental monitoring and safety	6.5	0.5%	1.00	10
Community agreements and support	12.8	1.0%	1.96	21
Total processing and shared	**337.3**	**26.3%**	**51.90**	**543**
Total operating cost	**1,280.8**	**100.0%**	**197.08**	**2,061**

Source: ABGM cash flow model (2026). Per-tonne and per-ounce values use LOM plant feed of 6.499 Mt and 621,513 oz recovered. OP diesel is nil because it is included in the contractor rate.

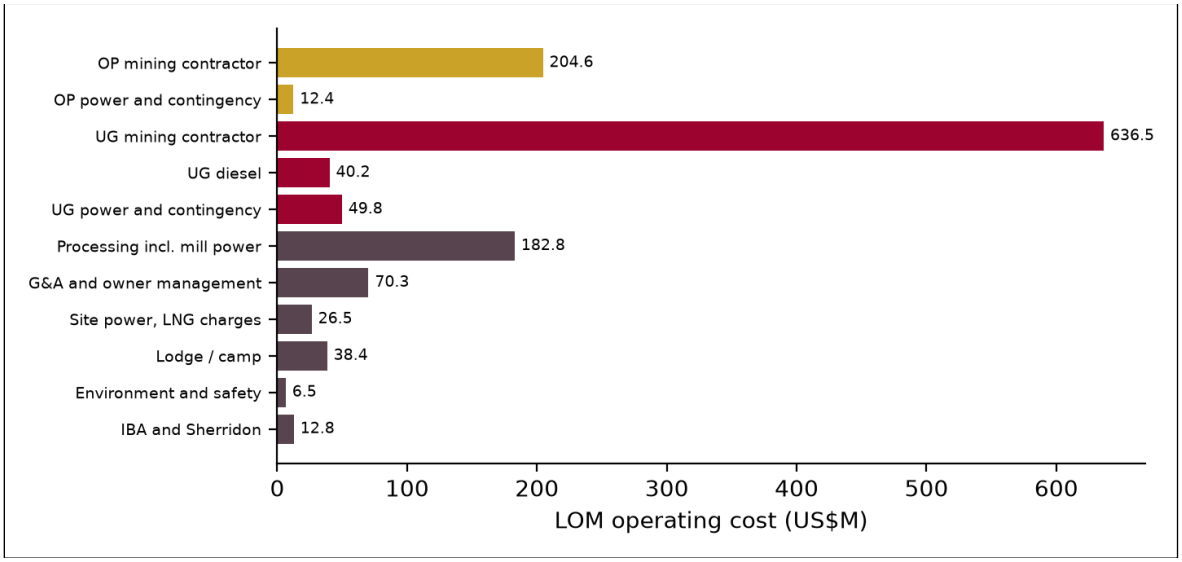


Figure 21-2: LOM operating cost composition (US$M)

Source: ABGM cash flow model (2026).

21.2.3 Unit operating costs

Table 21-7: LOM unit operating costs

Measure	Unit	Value
Open pit		
OP direct opex per tonne mined (mineralized material + waste, 38.7 Mt)	US$/t	5.61
OP direct opex per tonne OP mineralized material (1.76 Mt)	US$/t	123.58
OP direct opex per OP ounce recovered (212 koz)	US$/oz	1,024
Underground		
UG direct opex per tonne UG mineralized material (4.74 Mt)	US$/t	153.18
UG direct opex per tonne mined (mineralized material + waste, 8.16 Mt)	US$/t	89.06
UG direct opex per UG ounce recovered (409 koz)	US$/oz	1,774
Processing, shared and total		
Processing per tonne milled	US$/t	28.13

Measure	Unit	Value
Shared site, G&A and other per tonne milled	US$/t	23.76
Total operating cost per tonne milled	US$/t	197.08
Total operating cost per ounce recovered	US$/oz	2,061
Memo: fully allocated (processing and shared costs pro rata to mineralized material tonnes)		
OP: per tonne OP mineralized material / per OP ounce	US$/t; US$/oz	175.48 / 1,454
UG: per tonne UG mineralized material / per UG ounce	US$/t; US$/oz	205.08 / 2,376

Source: ABGM cash flow model (2026).

21.2.4 Mining costs

Open pit. OP mining by contractor moves 38.7 Mt (mineralized material 1.76 Mt, waste 36.9 Mt, strip ratio 21.0:1) from the North-East (NE), South-East (SE) and North-West (NW) pits over 2028 to 2030. Mining costs are taken from the pit optimization and entered directly in US$: US$5.48/t for waste and US$6.70/t for mineralized material (including high-grade and low-grade stockpile material). These rates include diesel, so no separate OP diesel cost is carried and no exchange-rate conversion is applied. LOM OP contractor cost is US$214.0M (waste US$202.3M, mineralized material US$11.8M), of which US$9.42M for waste mined in the first four months of 2028 is capitalized as pre-strip; the balance, US$204.6M, is operating cost. The same rates were used for the OP cut-off grade in the 2026 Mineral Resource Estimate (MRE).

Underground. UG development starts in Month 13 (January 2029), with first mineralized material in Month 15 (March 2029), and produces 4.74 Mt of mineralized material and 3.42 Mt of development waste at US$82.05/t mineralized material and US$72.44/t waste (US$389.1M and US$247.4M respectively). The UG contractor rates are quoted in US$/t and exclude diesel, which is supplied by the owner.

21.2.5 Diesel

Diesel is an owner cost for the UG fleet only, supplied at US$2.10/L delivered. Consumption is calculated monthly in the model as fleet count × effective operating hours × litres per hour. LOM UG diesel is 19.15 million litres (ML), US$40.2M. OP diesel is included in the pit-optimization contractor rates and is not costed separately, so it is not double counted.

Table 21-8: UG diesel consumption by machine type (LOM)

Machine	L/h	Peak fleet	LOM ML	US$M
Development drill rig	23	4	1.67	3.5
Production drill rig	23	4	1.67	3.5
Rock bolter	23	2	0.96	2.0
Load-haul-dump (LHD) loader, 9 t	29	4	3.51	7.4
Articulated dump truck (ADT), 40 t class	30	7	7.23	15.2
Utility vehicle	23	3	1.92	4.0
Explosives charger	23	8	1.28	2.7
Grader	11	3	0.92	1.9
Total UG diesel			**19.15**	**40.2**

Source: ABGM cash flow model (2026).

21.2.6 Power: LNG, grid and processing power

Site power for the first 36 months (2028 to 2030) is generated by LNG-fuelled gensets assuming the following supply terms (AECO gas price plus fees C$2.60/GJ, liquefaction C$5.50/GJ, transport C$10.00/GJ, 36% engine efficiency, giving a fuel cost of US$0.120/kWh), plus take-or-pay on nominated volumes and a monthly facility charge. From Month 37 (2031) the site is assumed to be on grid power at US$0.060/kWh. Mill power is costed in the power lines and removed from the US$28.00/t processing rate to avoid double counting (US$5.84/t at the LOM blended rate of US$0.078/kWh). OP power uses 10 kWh per tonne of OP mineralized material (US$2.1M LOM) and UG power 60 kWh per tonne of UG mineralized material (US$17.9M LOM). The UG factor includes about 20 kWh/t for the main ventilation fans (about 1.7 MW), which is consistent with the 1,735 kW of main fan power in the ventilation simulation (VENTSIM) report; it does not include auxiliary fans or mine-air heating.

21.2.7 Processing, G&A and other site costs

Processing is costed at US$22.16/t milled excluding power (US$28.00/t all-in). Including mill power, processing costs US$182.8M over 6.50 Mt. Site G&A of US$10.60/t milled includes owner management; the owner senior and junior management costs (US$1.5M each a year) are carried separately and deducted from G&A so they are counted once; the combined cost is US$70.3M. Lodge costs are US$38.4M, environmental monitoring and safety US$6.5M, and community agreements and support US$12.8M. The 5% contingency on OP and UG costs totals US$42.2M; the contingency on site electrical costs is included in the power cost and is not reported separately.

The QP for Recovery Methods estimates the plant operating cost at US$22.18/t excluding power and, with power at US$0.06/kWh, at US$38/t at 1,000 t/d and US$27/t at 2,000 t/d. The LOM average is close to the model processing cost of US$28.13/t milled, and the QP considers the model consistent with his estimate. The processing power estimate of 50 to 55 kWh/t covers processing only, while the model's 75 kWh/t includes HVAC and offices.

21.2.8 Annual operating cost schedule

Table 21-9: Annual operating cost by area (US$M unless stated)

Item	LOM	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Plant feed (Mt)	6.499	0.492	0.720	0.720	0.720	0.720	0.720	0.720	0.720	0.720	0.247
Open pit (direct)											
Mining contractor including diesel	204.6	58.4	93.8	52.4	-	-	-	-	-	-	-
Power	2.1	0.6	0.9	0.6	-	-	-	-	-	-	-
Contingency (5%)	10.3	3.0	4.7	2.6	-	-	-	-	-	-	-
Total open pit	**217.1**	**62.0**	**99.5**	**55.6**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Underground (direct)											
Mining contractor	636.5	-	5.7	37.7	108.1	95.6	93.7	93.2	93.1	91.4	17.9
Diesel	40.2	1.0	2.0	5.5	6.2	5.2	5.2	5.2	5.2	3.8	1.0
Power	17.9	-	0.2	1.4	2.5	2.6	2.6	2.6	2.6	2.6	0.6
Contingency (5%)	31.9	-	0.3	2.0	5.4	4.8	4.7	4.7	4.7	4.6	0.9
Total underground	**726.5**	**1.0**	**8.2**	**46.6**	**122.3**	**108.2**	**106.1**	**105.7**	**105.6**	**102.5**	**20.4**
Processing and shared											
Processing including mill power	182.8	15.6	22.8	22.8	19.2	19.2	19.2	19.2	19.2	19.2	6.6
Site G&A and owner management	70.3	5.8	8.1	7.4	7.5	7.7	7.8	7.7	7.7	7.7	2.9
Site power, LNG fixed charges, contingency	26.5	4.5	5.7	5.4	1.7	1.7	1.8	1.7	1.7	1.7	0.4
Lodge / camp	38.4	3.6	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.9	3.3
Environmental and safety	6.5	0.5	0.8	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.2
Community agreements and support	12.8	1.5	1.7	1.5	1.2	1.3	1.4	1.1	1.1	1.2	0.7
Total processing and shared	**337.3**	**31.5**	**43.0**	**41.7**	**34.3**	**34.6**	**34.8**	**34.5**	**34.4**	**34.6**	**14.0**
Total operating cost	**1,280.8**	**94.5**	**150.7**	**143.9**	**156.5**	**142.8**	**140.9**	**140.2**	**140.0**	**137.0**	**34.4**
Operating cost per tonne milled (US$/t)	197.08	192.03	209.27	199.80	217.41	198.28	195.71	194.66	194.46	190.30	139.44

Source: ABGM cash flow model (2026). 2026, 2027 and 2038 have no operating cost. Per-tonne values are calculated as cost / plant feed.

22 ECONOMIC ANALYSIS

Cautionary statement (NI 43-101, section 3.4(e)). The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. No Mineral Reserves have been estimated. Inferred Mineral Resources make up approximately 32.1% of the PEA plant feed tonnes and 24.6% of the contained gold. Most of the Inferred material is in the underground mine plan; Inferred makes up about 6.9% of the open pit plant feed tonnes.

The PEA plant feed of 6.50 Mt at 3.30 g/t gold (Au), containing 691 koz, includes mining dilution and losses. It compares with the 2026 Mineral Resource Estimate (Measured and Indicated plus Inferred) of 5.42 Mt at 4.61 g/t Au, containing 804 koz.

The economic analysis is a discounted cash flow (DCF) analysis of annual project cash flows on a 100% equity, unlevered basis. Pre-tax and after-tax results are presented. The headline measure is after-tax net present value (NPV) at a 5% real discount rate; the internal rate of return (IRR) and payback are also shown.

22.1 Principal Assumptions

Table 22-1: Principal economic assumptions

Assumption	Value
Market and revenue	
Gold price, base case	US$3,000/oz, flat, real
Gold price, alternative cases	US$3,500/oz and US$4,000/oz (long-term), flat, real
Payable gold	100% of recovered gold; no by-product credits
Refining, transport and selling	US$31.68/oz
Royalty	3% of gross revenue, as modelled (3.0% NSR rate applies at US$1,000/oz or more; slightly conservative)
Community agreements and support	Planning allowance (in operating cost)
Physicals	
Plant feed; head grade	6.50 Mt; 3.30 g/t Au (OP 1.76 Mt, UG 4.74 Mt)
Metallurgical recovery	90% overall (design); no ore sorting
Recovered gold	621,513 oz (OP 212,038 oz; UG 409,426 oz)
Pre-production; production life	2026 to 2027; 10 years (2028 to 2037); closure 2038
Financial	
Currency and terms	Real (unescalated) US$
Exchange rate	CAD:USD 0.74 (US$0.74 per C$1.00)
Discount rate	5% real (headline); 8%, 10% and 12% also reported
Discounting convention	End of year; valuation at the start of 2026 (first pre-production year)
Financing	100% equity; no debt, interest or hedging
Working capital; salvage	Not modelled; nil
Taxes	Manitoba mining tax; federal (15%) and Manitoba (12%) corporate income tax

Source: ABGM cash flow model (2026).

The base-case gold price of US$3,000/oz is applied flat in real terms over the life of mine (LOM) and is the price used for the 2026 MRE. The US$3,500/oz and US$4,000/oz cases use the same physicals and costs; only revenue, revenue-linked items (royalty, refining and the community allowance) and taxes change.

The refining terms used in the financial model, covering payable gold, treatment and refining charges, insurance and transport, are typical of doré sales and will be confirmed by refinery quotation at the PFS stage. No silver credit is assumed in the PEA.

22.2 Cash Flow Forecast

The project produces 621.5 koz of recovered gold over ten years (2028 to 2037). Production is front-loaded: the open pits supply high-grade mineralized material in 2028 to 2030 (peak 91.6 koz in 2029), 2030 is a transition year and the UG mine supplies all plant feed from 2031. At US$3,000/oz, gross revenue is US$1,864.5M and net revenue after royalty and refining is US$1,788.9M. After operating costs of US$1,280.8M, earnings before interest, taxes, depreciation and amortization (EBITDA) are US$508.1M. After capital of US$120.9M and taxes of US$139.6M, the undiscounted after-tax free cash flow is US$247.6M. Cumulative after-tax cash flow turns positive in 2028, the first production year.

Table 22-2: Annual cash flow forecast, base case US$3,000/oz

Item	LOM	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Mining and processing														
OP mineralized material mined (Mt)	1.756	-	-	0.527	0.735	0.494	-	-	-	-	-	-	-	-
OP waste mined (Mt)	36.91	-	-	11.74	16.22	8.95	-	-	-	-	-	-	-	-
UG mineralized material mined (Mt)	4.743	-	-	-	0.027	0.195	0.707	0.728	0.731	0.729	0.727	0.729	0.169	-
UG waste mined (Mt)	3.415	-	-	-	0.049	0.299	0.692	0.495	0.465	0.462	0.462	0.437	0.056	-
Plant feed (Mt)	6.499	-	-	0.492	0.720	0.720	0.720	0.720	0.720	0.720	0.720	0.720	0.247	-
Head grade (g/t Au)	3.30	-	-	5.71	4.39	3.71	2.80	3.05	3.28	2.46	2.51	2.80	2.73	-
Recovered gold (koz)	621.5	-	-	73.9	91.6	79.4	58.8	61.4	70.2	52.2	51.9	57.9	24.3	-
Revenue (US$M)														
Gross revenue	1,864.5	-	-	221.6	274.7	238.2	176.5	184.1	210.6	156.6	155.7	173.6	72.8	-
Less royalty (3%)	(55.9)	-	-	(6.6)	(8.2)	(7.1)	(5.3)	(5.5)	(6.3)	(4.7)	(4.7)	(5.2)	(2.2)	-
Less refining and selling	(19.7)	-	-	(2.3)	(2.9)	(2.5)	(1.9)	(1.9)	(2.2)	(1.7)	(1.6)	(1.8)	(0.8)	-
Net revenue	**1,788.9**	**-**	**-**	**212.6**	**263.6**	**228.5**	**169.3**	**176.7**	**202.1**	**150.3**	**149.4**	**166.6**	**69.9**	**-**
Operating cost (US$M)														
Open pit (direct)	217.1	-	-	62.0	99.5	55.6	-	-	-	-	-	-	-	-
Underground (direct)	726.5	-	-	1.0	8.2	46.6	122.3	108.2	106.1	105.7	105.6	102.5	20.4	-
Processing	182.8	-	-	15.6	22.8	22.8	19.2	19.2	19.2	19.2	19.2	19.2	6.6	-
Shared site, G&A and other	154.4	-	-	15.9	20.2	18.9	15.1	15.4	15.6	15.3	15.3	15.4	7.4	-
Total operating cost	**1,280.8**	**-**	**-**	**94.5**	**150.7**	**143.9**	**156.5**	**142.8**	**140.9**	**140.2**	**140.0**	**137.0**	**34.4**	**-**
EBITDA	**508.1**	**-**	**-**	**118.1**	**112.9**	**84.7**	**12.8**	**33.9**	**61.1**	**10.1**	**9.4**	**29.5**	**35.4**	**-**
Capital (US$M)														
Initial / non-sustaining	55.4	4.0	49.3	2.0	-	-	-	-	-	-	-	-	-	-
Sustaining (including 3% allowance)	62.9	-	-	31.4	2.5	17.5	2.5	1.5	1.5	1.5	1.5	1.5	1.5	-
Closure	2.6	-	-	-	-	0.1	-	-	-	-	-	-	-	2.5
Total capital	**120.9**	**4.0**	**49.3**	**33.4**	**2.5**	**17.6**	**2.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**1.5**	**2.5**
Pre-tax free cash flow	**387.2**	**(4.0)**	**(49.3)**	**84.7**	**110.4**	**67.1**	**10.3**	**32.4**	**59.7**	**8.6**	**7.9**	**28.0**	**33.9**	**(2.5)**
Taxes (US$M)														
Manitoba mining tax	53.0	-	-	3.1	18.8	10.1	1.0	3.2	8.9	0.9	0.8	2.8	3.4	-
Federal and Manitoba income tax	86.6	-	-	3.3	24.7	15.4	2.5	7.9	13.7	2.1	1.9	6.8	8.2	-
Total taxes	**139.6**	**-**	**-**	**6.5**	**43.5**	**25.4**	**3.5**	**11.1**	**22.6**	**3.0**	**2.7**	**9.6**	**11.6**	**-**
After-tax free cash flow	**247.6**	**(4.0)**	**(49.3)**	**78.2**	**66.9**	**41.6**	**6.8**	**21.3**	**37.0**	**5.7**	**5.2**	**18.4**	**22.3**	**(2.5)**
Cumulative after-tax free cash flow		(4.0)	(53.4)	24.8	91.8	133.4	140.2	161.5	198.5	204.2	209.4	227.8	250.1	247.6

Source: ABGM cash flow model (2026). Pre-tax free cash flow is EBITDA less total capital. Negative values in brackets.

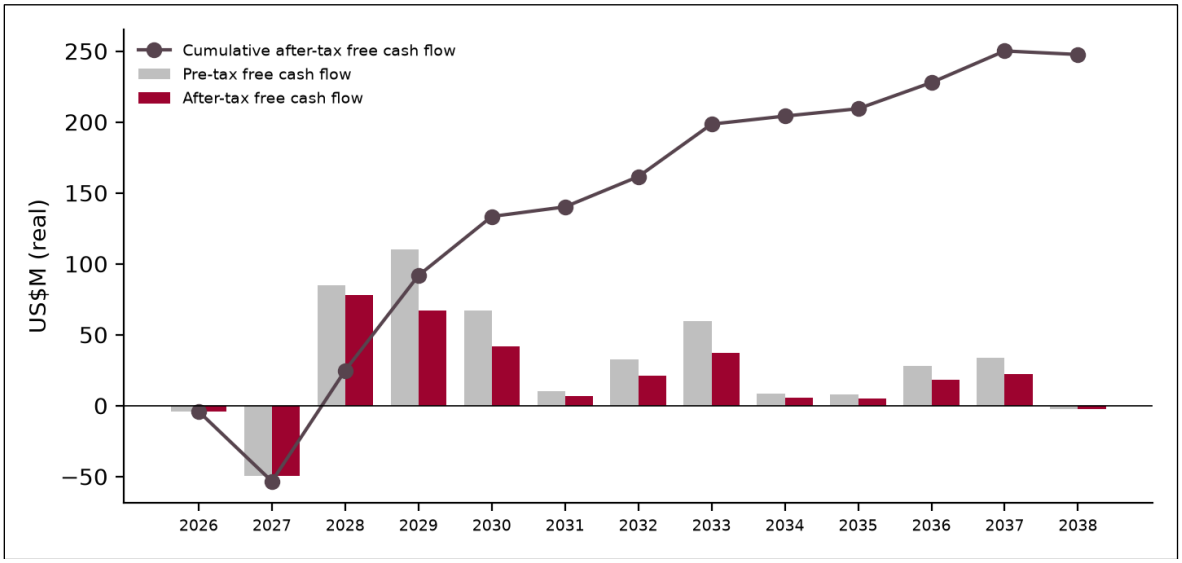


Figure 22-1: Annual pre-tax and after-tax free cash flow and cumulative after-tax free cash flow, base case (US$M)

Source: ABGM cash flow model (2026).

22.3 Taxes, Royalties and Other Government Levies

Royalty. The model deducts a royalty of 3% of gross revenue. The PL Property is subject to a sliding-scale net smelter return (NSR) royalty held by Franco-Nevada Corporation, at 3.0% for gold prices at or above US$1,000/oz, so the 3.0% rate applies in all PEA price cases. The modelled basis of 3% of gross revenue is retained, consistent with the filed MRE cut-off calculation. Applying the rate to gross revenue rather than to NSR (after refining, transport and selling) is slightly conservative, by about US$0.6M undiscounted over the LOM (3% of the refining and selling charges). Refining, transport and selling charges of US$31.68/oz (US$19.7M) are also deducted from gross revenue. Community agreements and support (US$12.8M) are included in operating costs. These are not taxes but reduce the profit on which taxes are calculated.

Taxes are calculated in the model on the annual EBITDA from the cash flow model.

Manitoba mining tax (Mining Tax Act) is a profit-based tax on mining profit. The model applies graduated rates of 10% on profit up to C$50M, 15% between C$55M and C$100M and 17% above C$105M, with notch rates of 65% and 57% on the excess over C$50M and C$100M in the C$50M to C$55M and C$100M to C$105M bands; thresholds are converted to US$ at 0.74. A new-mine holiday is modelled: mining profit is not taxed until the unrecovered capital pool (all capital spent in 2026 to 2028, US$86.8M by 2028) has been recovered, which happens during 2028; after that, capital spent in each year is deducted in that year. Taxable mining profit over the LOM is US$389.7M and Manitoba mining tax is US$53.0M. Mining tax is deductible in calculating corporate income tax.

Federal (15%) and Manitoba (12%) corporate income tax is applied to EBITDA less: (i) Canadian exploration expense (CEE) and capital cost allowance (CCA) pool deductions, being the opening pool of C$16.39M (US$12.1M) from the audited March 2026 financial statements plus capital spent, claimed up to profit after mining tax (US$130.5M LOM, of which US$98.9M in 2028); (ii) Manitoba mining tax; (iii) share-issue costs

(C$0.28M) at 20% a year; and (iv) the opening non-capital loss pool of C$4.91M (US$3.6M), used in 2028. Net taxable corporate income is US$320.7M, giving federal tax of US$48.1M and Manitoba corporate tax of US$38.5M.

At US$3,000/oz total taxes are US$139.6M, 27.5% of LOM EBITDA (for comparison, the model's flat 25% of earnings before interest and taxes (EBIT) method gives US$100.6M). The tax calculation is a PEA-level model and must be reviewed by a qualified tax adviser before the PEA is finalized. Open points include the holiday pool definition (pre-production depreciable assets, Mining Tax Act Formula 5) and the post-holiday depreciation allowance, mining-tax loss carry-forward, the processing allowance, and CCA by class.

Table 22-3: Tax parameters used in the model

Parameter	Value
Manitoba mining tax	
Thresholds 1 / 2 / 3 / 4	C$50M / 55M / 100M / 105M (US$37.0 / 40.7 / 74.0 / 77.7M)
Rates by band	10% to T1; 65% notch T1 to T2; 15% T2 to T3; 57% notch T3 to T4; 17% above T4
New-mine holiday	No mining tax until unrecovered capital is recovered
Corporate income tax	
Federal / Manitoba rates	15% / 12%
Opening CEE/CCA pool (audited financial statements, March 2026)	C$16.39M (US$12.13M)
Opening non-capital loss pool	C$4.91M (US$3.63M)
Share-issue cost pool; deduction rate	C$0.28M (US$0.21M); 20% a year

Source: ABGM cash flow model (2026).

Table 22-4: Annual tax calculation, base case US$3,000/oz (US$M)

Item	LOM	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
EBITDA	508.1	-	-	118.1	112.9	84.7	12.8	33.9	61.1	10.1	9.4	29.5	35.4	-
Total capital (pool additions)	120.9	4.0	49.3	33.4	2.5	17.6	2.5	1.5	1.5	1.5	1.5	1.5	1.5	2.5
Manitoba taxable mining profit	389.7	-	-	31.3	110.4	67.1	10.3	32.4	59.7	8.6	7.9	28.0	33.9	-
Manitoba mining tax	53.0	-	-	3.1	18.8	10.1	1.0	3.2	8.9	0.9	0.8	2.8	3.4	-
CEE/CCA deduction claimed	130.5	-	-	98.9	2.5	17.6	2.5	1.5	1.5	1.5	1.5	1.5	1.5	-
Net taxable corporate income	320.7	-	-	12.4	91.6	57.0	9.2	29.1	50.7	7.8	7.1	25.2	30.5	-
Federal corporate tax (15%)	48.1	-	-	1.9	13.7	8.5	1.4	4.4	7.6	1.2	1.1	3.8	4.6	-
Manitoba corporate tax (12%)	38.5	-	-	1.5	11.0	6.8	1.1	3.5	6.1	0.9	0.9	3.0	3.7	-
Total taxes	**139.6**	**-**	**-**	**6.5**	**43.5**	**25.4**	**3.5**	**11.1**	**22.6**	**3.0**	**2.7**	**9.6**	**11.6**	**-**

Source: ABGM cash flow model (2026).

Table 22-5: LOM taxes by gold price case (US$M)

Item	US$3,000/oz (base)	US$3,500/oz	US$4,000/oz
EBITDA	508.1	808.0	1,107.8
Manitoba mining tax	53.0	106.2	163.4
Federal and Manitoba corporate income tax	86.6	153.2	218.7
Total taxes	**139.6**	**259.4**	**382.1**
Total taxes as % of EBITDA	27.5%	32.1%	34.5%
Pre-tax NPV at 5% less after-tax NPV at 5%	102.8	189.5	278.1

Source: ABGM cash flow model (2026), recalculated at each gold price.

22.4 Economic Results: NPV, IRR and Payback

At the base-case gold price of US$3,000/oz the project has an after-tax NPV at 5% of US$180.6M (US$151.4M at 8%), an after-tax IRR of 111.0% and an after-tax payback of 0.68 years from the start of production in 2028. The pre-tax NPV at 5% is US$283.4M with a pre-tax IRR of 143.1%. At US$3,500/oz the after-tax NPV at 5% rises to US$310.8M and at US$4,000/oz to US$438.9M (Table 22-7). The all-in sustaining cost (AISC) is US$2,288/oz.

Note on IRR. The after-tax IRR exceeds 100% in all three price cases. This reflects the low pre-production capital (US$53.4M in 2026 and 2027, because the existing PL mill and site infrastructure are refurbished rather than rebuilt) relative to the high-margin OP production of 2028 and 2029, when AISC is about US$1,798 to US$1,830/oz. With such a small, short initial investment the IRR is mathematically very large and highly sensitive to the timing of the pre-production spend; it is a less informative measure for this project than NPV and payback.

Table 22-6: Economic results, base case US$3,000/oz (US$M unless stated)

Measure	Pre-tax	After-tax
Undiscounted free cash flow	387.2	247.6
NPV at 5% (headline)	**283.4**	**180.6**
NPV at 8%	238.2	151.4
NPV at 10%	213.1	135.1
NPV at 12%	191.4	121.1
IRR (%)	143.1	111.0
Payback from start of production (years)	Not reported	0.68
Pre-production capital (2026 and 2027)	53.4	53.4
After-tax NPV at 5% / pre-production capital	-	3.4x

Source: ABGM cash flow model (2026). Pre-tax values are EBITDA less total capital, discounted on the same basis as the after-tax values. Pre-tax payback is not shown because the model does not report it.

Table 22-7: Economic results by gold price case

Measure	US$3,000/oz (base)	US$3,500/oz	US$4,000/oz
After-tax NPV at 5% (US$M)	**180.6**	**310.8**	**438.9**
After-tax NPV at 8% (US$M)	151.4	260.1	367.2
After-tax NPV at 10% (US$M)	135.1	232.1	327.7
After-tax IRR (%)	111.0	151.6	187.5
After-tax payback from start of production (years)	0.68	0.54	0.45
LOM after-tax free cash flow (US$M)	247.6	427.7	604.9
Total taxes (US$M)	139.6	259.4	382.1
Pre-tax NPV at 5% (US$M)	283.4	500.2	717.0
Pre-tax IRR (%)	143.1	201.8	254.6
LOM pre-tax free cash flow (US$M)	387.2	687.1	987.0
EBITDA (US$M)	508.1	808.0	1,107.8
AISC (US$/oz)	2,288	2,305	2,323

Source: ABGM cash flow model (2026), recalculated at each gold price. Costs and physicals are the same in all cases.

Table 22-8: NPV at a range of real discount rates, base case (US$M)

Discount rate	Pre-tax NPV	After-tax NPV
0%	387.2	247.6
3%	320.0	204.2
5%	**283.4**	**180.6**
8%	238.2	151.4
10%	213.1	135.1
12%	191.4	121.1
15%	163.9	103.2
20%	128.2	80.1

Source: ABGM cash flow model (2026), annual cash flows. 0% is the undiscounted cash flow.

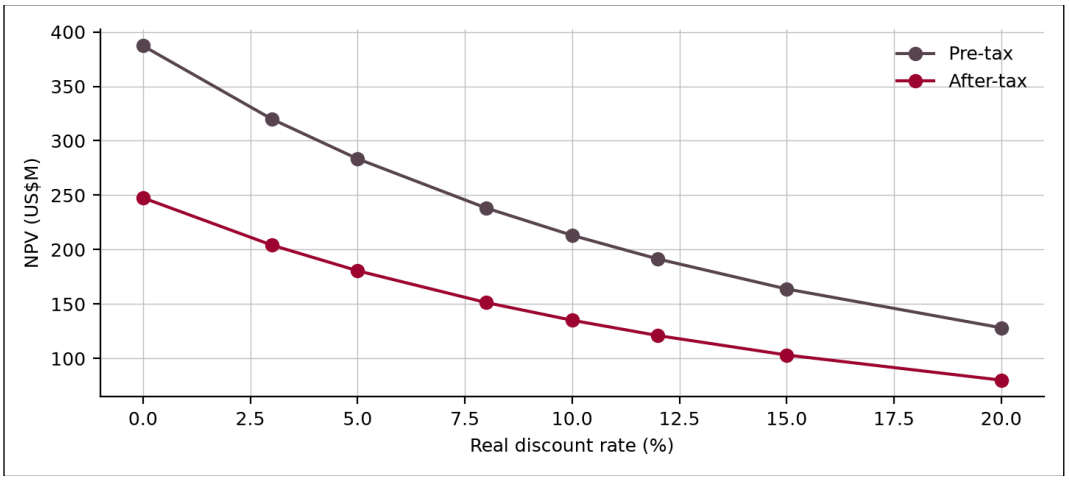


Figure 22-2: NPV versus discount rate, base case (US$M)

Source: Table 22-8.

22.5 Sensitivity Analysis

22.5.1 Standard sensitivities

The sensitivities in this section were recomputed for this draft from the cash flow model. The model itself was not changed. Two methods were used:

- Full-model recalculation. For gold price, metallurgical recovery and the CAD:USD exchange rate, the input was changed in a copy of the model and the whole workbook was fully recalculated. This re-runs the production schedule, revenue, royalty, community allowance, operating cost, capital, the full tax calculation and the cash flow. An unchanged copy recalculated the same way reproduces the published base case exactly (after-tax NPV at 5% US$180.602M, IRR 111.0%, AISC US$2,288/oz).
- Model sensitivity engine. Operating cost and capital cost were flexed using the model's built-in sensitivity calculation, recalculated in the same copy. It scales total operating cost (excluding the revenue-linked community allowance) or total capital (including sustaining and closure) by the stated factor and re-runs the complete tax logic. The engine was checked against the full-model runs: its gold price and recovered-ounce results at ±10% and ±20% match the full-model results for US$2,400, US$2,700, US$3,300 and US$3,600/oz and for 72%, 81% and 99% recovery to four decimal places.

Table 22-9: After-tax NPV at 5% sensitivity (US$M), change in input

Variable	-20%	-10%	Base	+10%	+20%
Gold price	5.0	96.7	180.6	257.2	336.3
Grade / recovery (recovered ounces)	7.3	97.6	180.6	256.3	334.8
Operating cost	288.0	234.0	180.6	122.1	60.5
Capital cost	192.6	186.3	180.6	173.8	167.0
CAD:USD exchange rate (US$ per C$)	-	181.6	180.6	178.9	-

Source: ABGM cash flow model (2026). Gold price, grade/recovery and exchange rate (±10% on 0.74) use full-model recalculation; operating and capital cost use the model's sensitivity calculation. For cost variables "+" is a cost increase (adverse). For the exchange rate "+" is a stronger Canadian dollar.

Table 22-10: After-tax IRR sensitivity (%), change in input

Variable	-20%	-10%	Base	+10%	+20%
Gold price	27.9	79.2	111.0	135.5	159.3
Grade / recovery (recovered ounces)	30.4	79.6	111.0	135.2	158.8
Operating cost	136.2	123.5	111.0	95.0	74.8
Capital cost	141.5	124.4	111.0	98.7	87.8
CAD:USD exchange rate	-	112.8	111.0	108.9	-

Source: ABGM cash flow model (2026), sensitivity calculation; exchange rate from full-model recalculation.

Table 22-11: After-tax payback sensitivity (years from start of production), change in input

Variable	-20%	-10%	Base	+10%	+20%
Gold price	1.26	0.86	0.68	0.59	0.52
Grade / recovery (recovered ounces)	1.25	0.85	0.68	0.59	0.52
Operating cost	0.60	0.64	0.68	0.74	0.82
Capital cost	0.55	0.62	0.68	0.75	0.84
CAD:USD exchange rate	-	0.67	0.68	0.69	-

Source: ABGM cash flow model (2026), sensitivity calculation; exchange rate from full-model recalculation.

The project is most sensitive to gold price and recovered ounces (head grade or recovery), which have almost identical effects because costs are driven by tonnes rather than ounces, followed by operating cost. It is relatively insensitive to capital cost and to the exchange rate (most costs are in US$). A 10% change in gold price changes after-tax NPV at 5% by about US$80M, a 10% change in operating cost by about US$56M, a 10% change in capital by about US$6M and a 10% change in the exchange rate by about US$1.4M. At 20% below the base gold price (US$2,400/oz) the after-tax NPV at 5% falls to US$5.0M, close to break-even. At low gold prices the value of the project depends almost entirely on the OP years, as UG AISC exceeds US$2,800/oz in 2031, 2034 and 2035.

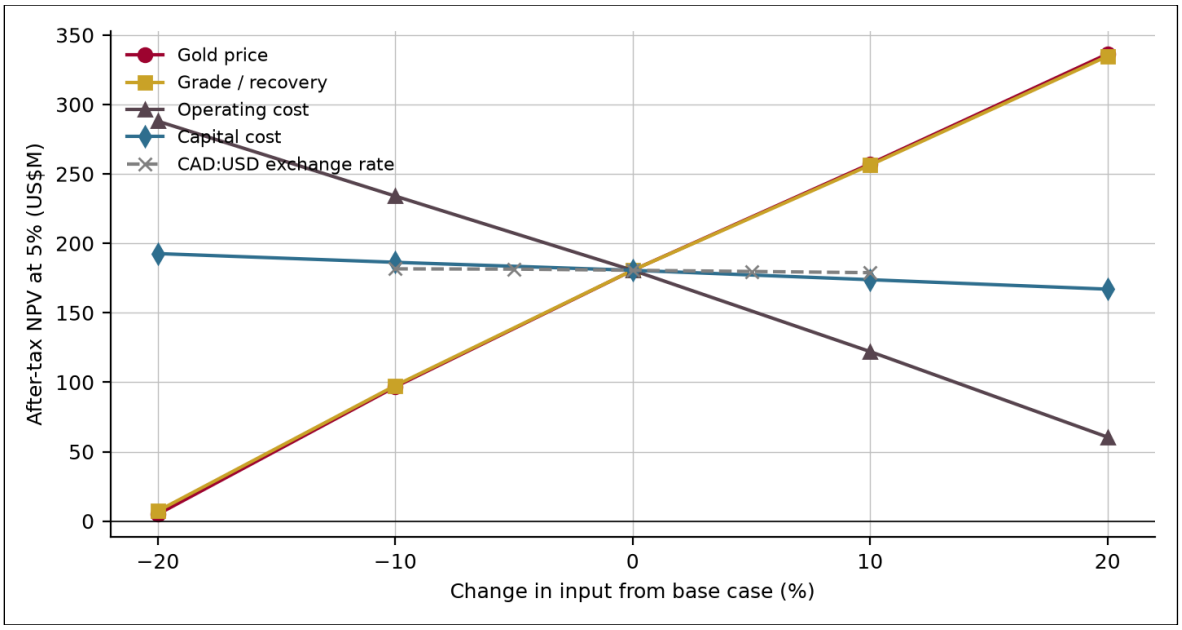


Figure 22-3: After-tax NPV at 5% sensitivity (spider)

Source: Tables 22-9 and 22-14.

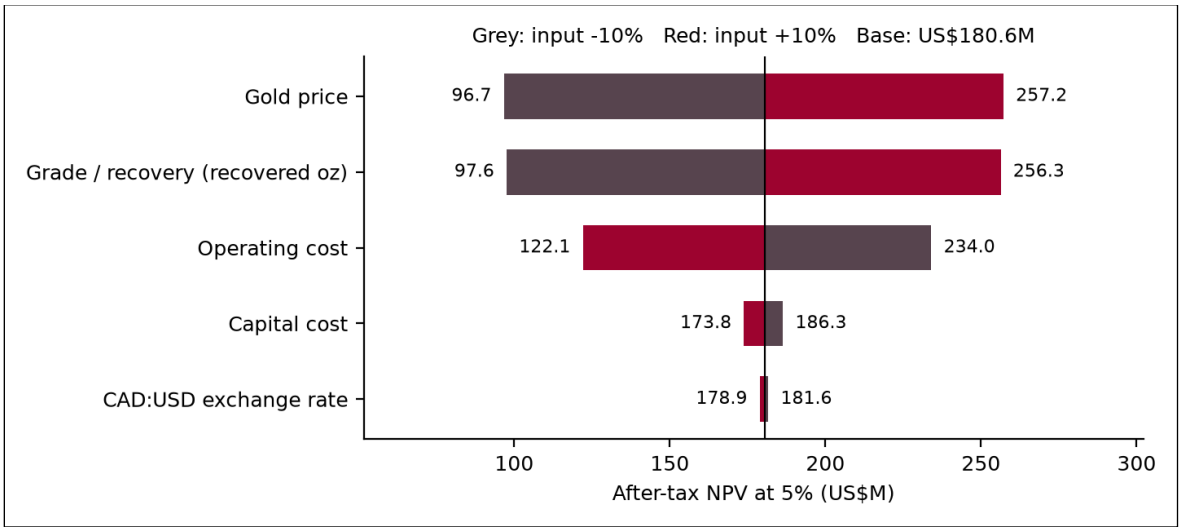


Figure 22-4: Tornado chart: after-tax NPV at 5% for ±10% changes in input

Source: Table 22-9. For cost variables the red bar (+10%) is a cost increase.

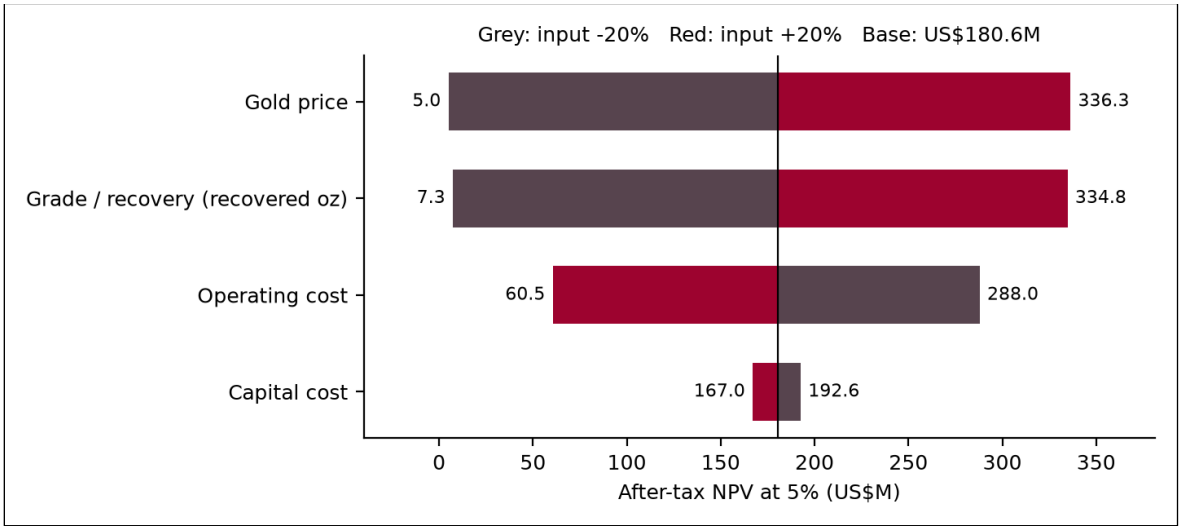


Figure 22-5: Tornado chart: after-tax NPV at 5% for ±20% changes in input

Source: Table 22-9. The exchange rate was run at ±5% and ±10% only.

Table 22-12: Gold price sensitivity, US$2,500 to US$4,500/oz (US$M unless stated)

US$/oz	2,500	2,750	3,000 (base)	3,250	3,500	3,750	4,000	4,250	4,500
After-tax NPV at 5%	**38.9**	**110.8**	**180.6**	**244.1**	**310.8**	**373.6**	**438.9**	**504.1**	**568.5**
After-tax NPV at 8%	32.9	93.0	151.4	204.4	260.1	312.6	367.2	421.7	475.6
Pre-tax NPV at 5%	66.7	175.1	283.4	391.8	500.2	608.6	717.0	825.4	933.8
After-tax IRR (%)	52.0*	85.3*	111.0	131.3	151.6	169.4	187.5	204.8	221.7
Pre-tax IRR (%)	69.9	109.9	143.1	173.4	201.8	228.7	254.6	279.5	303.6
After-tax payback (years)	1.09	0.81	0.68	0.61	0.54	0.49	0.45	0.41	0.38
LOM after-tax free cash flow	51.6	151.0	247.6	335.4	427.7	514.6	604.9	695.2	784.0
Total taxes	35.7	86.3	139.6	201.7	259.4	322.4	382.1	441.7	502.8
AISC (US$/oz)	2,270	2,279	2,288	2,297	2,305	2,314	2,323	2,332	2,340

*Source: ABGM cash flow model (2026), recalculated at each gold price. * Below US$3,000/oz after-tax cash flow is negative in some production years, so the cash flow changes sign more than once and the IRR may not be unique; rely on NPV.*

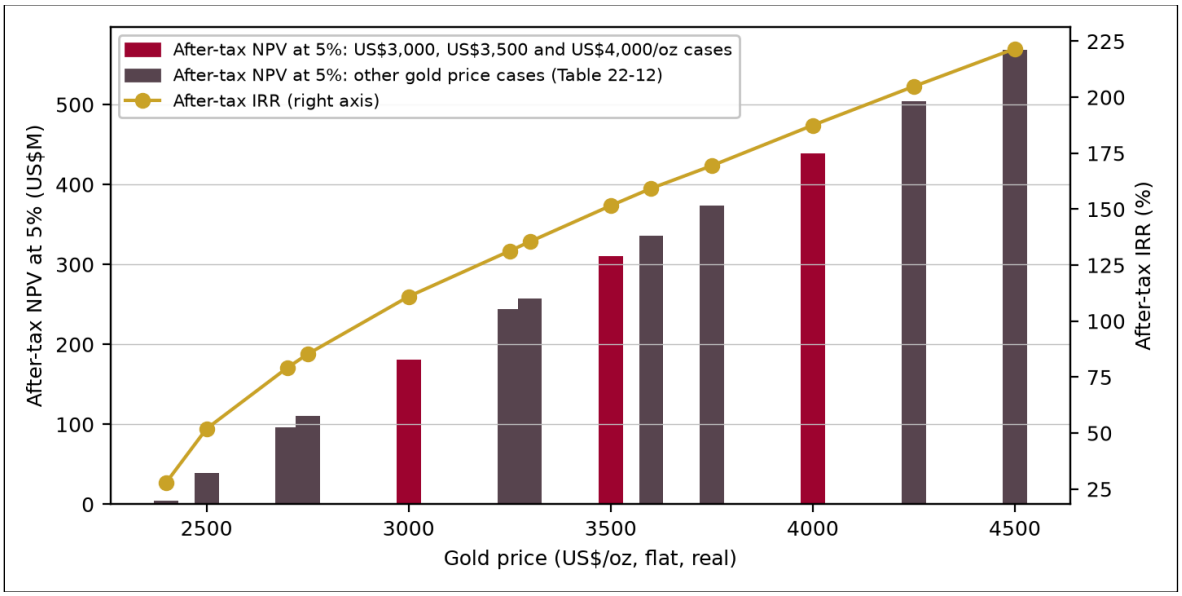


Figure 22-6: After-tax NPV at 5% (bars) and IRR (line, right axis) versus gold price

Source: Full-model recalculation, Table 22-12.

Table 22-13: Metallurgical recovery sensitivity (no ore sorting in any case)

Overall recovery	72%	81%	85%	90% (base)	95%	99%
Recovered gold (koz)	497.2	559.4	587.0	621.5	656.0	683.7
After-tax NPV at 5% (US$M)	**7.3**	**97.6**	**134.9**	**180.6**	**222.7**	**256.3**
After-tax IRR (%)	30.4	79.6	94.9	111.0	124.1	135.2
After-tax payback (years)	1.25	0.85	0.76	0.68	0.63	0.59
AISC (US$/oz)	2,826	2,527	2,414	2,288	2,175	2,092

Source: ABGM cash flow model (2026), recalculated with the recovery changed. 72% and 81% are the -20% and -10% cases and 99% the +10% case on the base 90%; a +20% case (108%) is not physically possible, so the +20% "grade / recovery" values in Table 22-9 are a head-grade proxy only.

Table 22-14: Exchange rate sensitivity (US$ per C$1.00)

CAD:USD	0.666	0.703	0.740 (base)	0.777	0.814
After-tax NPV at 5% (US$M)	181.6	181.5	180.6	179.7	178.9
After-tax IRR (%)	112.8	112.1	111.0	110.0	108.9
AISC (US$/oz)	2,283	2,285	2,288	2,291	2,293

Source: ABGM cash flow model (2026), recalculated with the exchange rate changed (-10%, -5%, +5%, +10%). The exchange rate affects only C$-denominated inputs (gensets, fuel farm, OP establishment, LNG charges and tax thresholds); mining contractor and processing rates are in US$.

22.5.2 Plant ramp-up and capital sensitivity

The base case assumes first plant feed in Month 3 (March 2028), reaching 1,000 tonnes per day (t/d) in Month 4 and 2,000 t/d in Month 8, at a flat 90% recovery. Sensitivities are run on the project ramp-up profile and on the process plant and tailings capital estimated by the QP for Recovery Methods. They are sensitivities only, and the base case is unchanged. Each case is a full recalculation of a copy of the cash flow model. An unchanged copy recalculated the same way reproduces the base case exactly (after-tax NPV at 5% of US$180.6M).

Case C1 changes only the ramp-up:

- First feed is in January 2029 (Month 13, where Month 1 is January 2028).
- Project 1 processes 22,500 t, 25,000 t and 27,500 t in its first three months and 30,000 t/month (1,000 t/d) after that.
- Project 2 is in place before throughput passes 1,000 t/d. Its ramp starts in December 2029 (Month 24), adding 25,000 t, 27,500 t and then 30,000 t/month.
- Recovery on each project's tonnes is 80% in its first month, 85% in its second and 90% after that.
- Initial and sustaining capital, grid power (Month 49) and site fixed costs all move 12 months later.
- Mining starts are deferred to keep the running ROM stockpile within the 157 thousand tonne (kt) pad. The SE, NE, NW and UG starts move from Months 4, 1, 10 and 13 to Months 16, 11, 25 and 20.
- The mining sequence, tonnes, grades and unit costs are unchanged. The peak stockpile is 145.5 kt, and all mineralized material is processed by Month 131.
- Operating costs and power are unchanged.
- The valuation date does not move, so cash flows are still discounted from 2026.

Case C2 changes only the capital costs, on the base schedule:

- The 2,000 t/d plant expansion is US$20.0M (US$16.8M direct plus US$3.2M contingency, compared with US$14.0M in the base case), spread over 2026, 2027 and 2028 (US$2.9M, US$14.3M and US$2.9M).
- The tailings storage facility expansion is US$9.0M (US$4.0M in the base case).
- The plant estimates already include contingency, so the 15% contingency no longer applies to the plant refurbishment or the plant expansion. This removes US$3.0M.

Case C3 combines C1 and C2 and is the combined ramp-up and capital downside case. Table 22-15 shows the results of all three cases.

Results for C3:

- After-tax NPV at 5% falls by US$20.8M to US$159.8M, and after-tax IRR falls from 111.0% to 70.4%. Most of the loss comes from the one-year delay to first gold and the slower ramp (C1, US$16.6M lower), and the capital changes account for US$4.5M (C2).
- LOM gold falls by 1.2 thousand ounces (koz) because recovery is lower during both ramps.
- Payback, in years from the start of 2028, lengthens from 0.68 to 2.22 years. That is 1.22 years after first feed in January 2029.
- AISC is US$2,306/oz, compared with US$2,288/oz in the base case. In C2, AISC falls to US$2,276/oz because most of the plant expansion moves into pre-production and is no longer counted as sustaining capital.

Alternative readings of the schedule and of the Project 1 recovery ramp are also run:

- If capital, grid power and site fixed costs stay on the base calendar and only the plant start is delayed, after-tax NPV at 5% is US$156.7M in C1 and US$152.2M in C3.
- If grid power still arrives in Month 37, after-tax NPV at 5% is US$166.6M in C1 and US$162.4M in C3.
- If Project 1 recovery follows the ramp-up workbook (75%, 77.5%, 80%, 84% and 87%, reaching 90% in the sixth month), after-tax NPV at 5% is US$162.2M in C1 and US$158.0M in C3.

Table 22-15: Plant ramp-up and capital cost sensitivity (US$3,000/oz)

Measure	Base case	C1: ramp only	C2: capital only	C3: ramp and capital
First plant feed	Mar 2028	Jan 2029	Mar 2028	Jan 2029
Mining start months SE / NE / NW / UG	4 / 1 / 10 / 13	16 / 11 / 25 / 20	4 / 1 / 10 / 13	16 / 11 / 25 / 20
After-tax NPV at 5% (US$M)	180.6	164.0	176.1	159.8
Change in after-tax NPV at 5% (US$M)	-	(16.6)	(4.5)	(20.8)
Pre-tax NPV at 5% (US$M)	283.4	260.3	276.8	254.0
After-tax IRR (%)	111.0	84.4	90.1	70.4
Pre-tax IRR (%)	143.1	110.5	116.1	92.2
After-tax payback (years from start of 2028)	0.68	2.18	0.75	2.22
Pre-tax payback (years from start of 2028)	0.63	2.12	0.71	2.15
AISC (US$/oz)	2,288	2,295	2,276	2,306
LOM gold produced (koz)	621.5	620.4	621.5	620.4
Gold sold 2028 / 2029 / 2030 (koz)	73.9 / 91.6 / 79.4	0.0 / 51.3 / 98.2	73.9 / 91.6 / 79.4	0.0 / 51.3 / 98.2
Peak ROM stockpile (kt)	156.8	145.5	156.8	145.5
Change in peak ROM stockpile (kt)	-	(11.3)	-	(11.3)

Source: ABGM cash flow model (2026), full recalculation. Base case values are from the unchanged model. Pre-tax cash flow is EBITDA less capital. Payback after first feed is 0.52 years in the base case, 1.18 years in C1 and 1.22 years in C3. Figures in brackets are reductions.

First UG development in Month 13 (January 2029) and first UG mineralized material in Month 15 (March 2029).

22.6 Cash Costs and All-in Sustaining Costs

Cash costs and AISC are calculated per ounce sold following the World Gold Council (WGC) guidance note on non-GAAP (generally accepted accounting principles) metrics (June 2013, updated November 2018). Total cash cost comprises on-site mining, processing, G&A and community costs (including the community allowance and the operating cost contingency), refining and selling charges and the 3% royalty. AISC adds sustaining capital (named sustaining items and the 3% allowance) and closure accrued over ounces sold. No corporate G&A is allocated and there are no by-product credits. Income and mining taxes are excluded. All-in cost (AIC) adds non-sustaining (initial) capital. AISC is US$2,288/oz at US$3,000/oz, US$2,305/oz at US$3,500/oz and US$2,323/oz at US$4,000/oz, the difference being the revenue-linked royalty.

Table 22-16: LOM cash cost, AISC and all-in cost (WGC basis), base case

Component	US$M	US$/oz sold
On-site operating costs	1,280.8	2,061
Refining, transport and selling	19.7	32
Cash cost before royalty	**1,300.5**	**2,093**
Royalty (3%)	55.9	90
Total cash cost	**1,356.5**	**2,183**
Sustaining capital: named items	47.9	77
Sustaining capital: 3% allowance	15.0	24
Closure and reclamation (accrued)	2.6	4
Corporate G&A; by-product credits	-	-
AISC	**1,422.0**	**2,288**
Non-sustaining (initial) capital	55.4	89
All-in cost	**1,477.3**	**2,377**

Source: ABGM cash flow model (2026); 621,513 oz sold.

Table 22-17: Annual cash cost, AISC and all-in cost per ounce sold, base case

Measure	LOM	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Gold sold (koz)	621.5	73.9	91.6	79.4	58.8	61.4	70.2	52.2	51.9	57.9	24.3
Total cash cost (US$/oz)	2,183	1,401	1,767	1,934	2,782	2,447	2,129	2,806	2,819	2,489	1,541
AISC (US$/oz)	**2,288**	**1,830**	**1,798**	**2,158**	**2,829**	**2,476**	**2,154**	**2,839**	**2,852**	**2,520**	**1,606**
All-in cost (US$/oz)	2,377	1,858	1,798	2,158	2,829	2,476	2,154	2,839	2,852	2,520	1,606

Source: ABGM cash flow model (2026). Annual all-in cost excludes pre-production capital; the LOM all-in cost includes it.

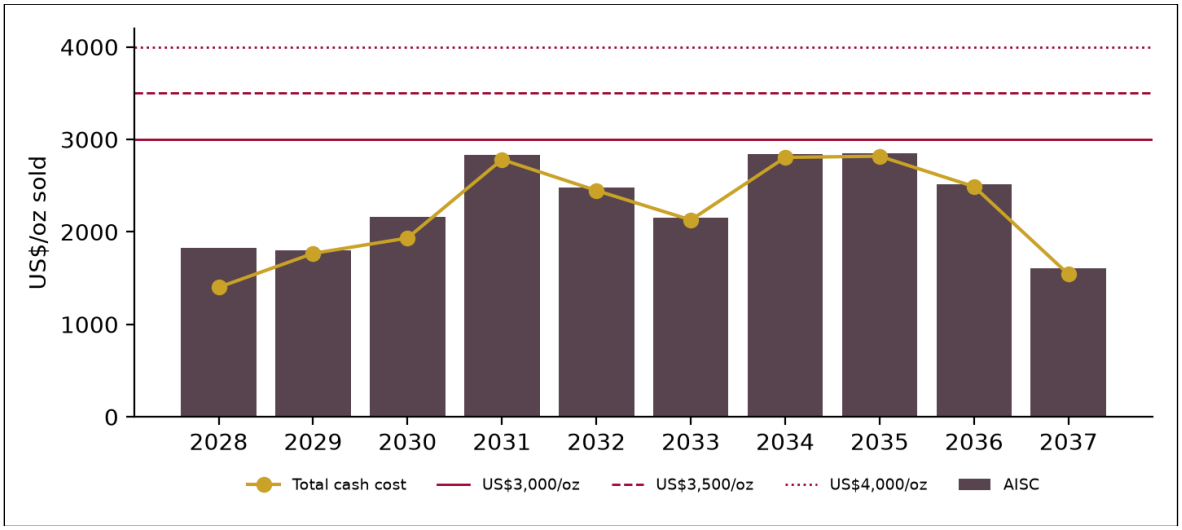


Figure 22-7: Annual total cash cost and AISC per ounce sold compared with the gold price cases

Source: ABGM cash flow model (2026).

23 ADJACENT PROPERTIES

This Item describes nearby properties for context only. The information comes from public disclosure by their owners and has not been verified by the QP. Mineralization on these properties is not necessarily indicative of mineralization on the PL Property, and none of it is included in the 2026 MRE.

23.1 Nokomis Property (Minnova)

Minnova's contiguous Nokomis Property lies less than 8 km northeast of the PL mill. It comprised 27 claims (566 ha) in 2017 and is subject to an underlying 2% NSR royalty. Gold occurs in shear-related, intrusion-hosted Upper and Lower Host zones within a differentiated gabbroic sill. In 2014 ACA Howe estimated 371,000 t at 3.41 g/t Au (41,000 oz) Indicated and 247,000 t at 2.41 g/t Au (19,000 oz) Inferred at a 0.6 g/t Au cut-off (2018 FS, Table 14.14; April 17, 2014 news release). This estimate is historical in the context of this Technical Report. The QP has not verified it, it is not relied upon, and Nokomis is excluded from the 2026 MRE.

23.2 Sherridon Copper-Zinc-Gold-Silver Project (T2 Metals Corp.)

T2 Metals Corp. (TSXV: TWO) holds the Sherridon volcanogenic massive sulphide (VMS) project around the community of Sherridon, northwest of the PL Property. The project includes the former Sherritt Gordon East and West mines. T2 Metals reported a Mineral Resource Estimate with an effective date of April 22, 2026 for the Bob, Cold Lake, Lost Lake and Jungle deposits. It comprises 10.04 Mt at 0.39% Cu, 0.83% Zn, 0.30 g/t Au and 4.75 g/t Ag Indicated, and 18.15 Mt at 1.08% Cu, 1.00% Zn, 0.27 g/t Au and 7.05 g/t Ag Inferred (T2 Metals, May 7, 2026). The supporting technical report was filed in June 2026 (T2 Metals, June 26, 2026). The project covers an area of about 5,866 ha and is approximately 12 km north of the PL Property boundary. The Sherridon VMS mineralization is of a different deposit type from the shear-hosted gold at PL.

23.3 Snow Lake Operations (Hudbay Minerals Inc.)

Hudbay's Snow Lake operations, about 200 km by road east of Flin Flon (via PTH 10, PTH 39 and PR 392) in the same Flin Flon-Snow Lake greenstone belt, include the Lalor gold-copper-zinc mine, the New Britannia gold mill (nameplate 1,500 t/d, permitted 2,500 t/d, operating above 2,000 t/d), the Stall concentrator and several satellite deposits. In September 2026 Hudbay reported Proven and Probable Mineral Reserves of 27.0 Mt containing about 2.0 Moz of gold, a reserve mine life to 2043 and 2026 to 2030 average gold production of 185,000 oz per year (Hudbay, 2026). New Britannia is a refurbished past-producing gold mill, a regional precedent for restarting historical gold processing infrastructure.

23.4 McIlvenna Bay (Eldorado Gold Corp., formerly Foran Mining)

The McIlvenna Bay underground copper-zinc-gold-silver VMS mine in east-central Saskatchewan, in the western Flin Flon greenstone belt, produced its first copper concentrate in June 2026 and its first zinc concentrate in July 2026. It was ramping up to commercial production at a design rate of 4,900 t/d, with a study under way on expansion to about 7,000 t/d (Eldorado Gold, 2026). It shows continued investment in the region's mining workforce, services and logistics.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Schedule

The base case assumes pre-production work in 2026 and 2027, first plant feed in March 2028 (Month 3, where Month 1 is January 2028), the plant reaching 1,000 t/d in Month 4 and 2,000 t/d in Month 8, and first underground mineralized material in 2029. The QP for Recovery Methods describes a sequence of refurbishment, about 21 months of operation at 1,000 t/d to confirm the process, then construction and commissioning of the 2,000 t/d plant. The project team expects first feed in about January 2029, rather than July 2028, to allow more time for plant refurbishment. The effect of the QP's sequence and a January 2029 first feed is shown in Item 22 (sensitivities C1 and C3), and the execution schedule is to be developed at the next study stage.

24.2 Plant Ramp-Up Considerations

The 2,000 t/d plant must be in place once throughput exceeds 1,000 t/d. The project ramp-up profile (tonnage and recovery ramp-up for the 1,000 t/d and 2,000 t/d plants), agreed by the project team, is the basis for the ramp-up sensitivity. On this basis, with first feed in January 2029 and the QP's plant and tailings capital, the after-tax NPV at 5% is US$159.8M (sensitivity C3 in Item 22), compared with US$180.6M in the base case.

24.3 Gold Inventory and Doré Shipment

About two weeks of plant gold production will be locked up in the circuit at start-up. At 2,000 t/d doré shipments are expected to be about 2,000 oz to 2,800 oz of doré a week, transported from site by helicopter. These are operating considerations and are not costed separately. The model average at the 2029 peak (91.6 koz) is about 1,760 oz of gold a week. At a doré grade of about 70% to 80% gold, this is about 2,200 oz to 2,500 oz of doré a week, within the expected range.

24.4 Alternative Cases

Alternative cases assessed as cost differences or sensitivities, and not included in the base case, are the QP's process plant and tailings capital estimates, mine-air heating and backfill costs, and the QP's ramp-up basis. Adopting any of these would change the published base case.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Data

The PL deposit is a structurally controlled, multi-lens orogenic gold system. Its geometry is well understood from more than 40 years of drilling and from the 1986 to 1989 underground development. The 2026 database (1,005 collars, approximately 146.7 km of drilling) has been structurally validated. Historical holes with limited QA/QC documentation are assessed against nearby modern drilling and geological continuity. In Mr. King's

opinion, the data are adequate to support the classification of Measured, Indicated and Inferred Mineral Resources as reported.

25.2 Mineral Resources

The 2026 MRE contains 3,757,026 t at 4.98 g/t Au (601,158 oz) Measured and Indicated and 1,667,132 t at 3.78 g/t Au (202,810 oz) Inferred, reported at a gold price of US$3,000/oz and constrained by an open-pit shell and underground MSO shapes. The open-pit Measured and Indicated portion is 1,578,110 t at 5.28 g/t Au (268,091 oz). A like-for-like comparison with the historical 2017 estimate is not appropriate because of the lower cut-off grades and the updated reporting constraints. The increase in Measured and Indicated Mineral Resources mainly reflects the increased drill density and the resulting improvement in geological confidence.

25.3 Risks and Uncertainties

- Historical workings: The 1986 to 1989 underground workings lie partly within and beneath the conceptual open-pit shell. Any unmapped or inaccurately surveyed voids could reduce the Mineral Resource tonnes near the historical workings and affect pit wall and crown pillar stability. The QP has reduced this risk by depleting all material within the mined-out perimeters. The void model should nonetheless be verified against the historical production records before any mine design is done.
- Metallurgical recovery: The cut-off grades use 90% recovery, based on historical testwork and the 2018 FS. The historical plant achieved about 83% in doré, peaking at 89%. At 82% recovery, the full-cost open-pit cut-off would be 0.65 g/t Au and the underground break-even grade 2.11 g/t Au. No new metallurgical testwork has been completed for the 2026 MRE.
- Underground mining method: Underhand drift mining without backfill has not been used at PL, where historical stoping was up-dip slusher mining. The stability of unfilled drift voids, the sill pillar spacing and the 25 m crown pillar beneath the pit are not yet supported by detailed geotechnical analysis. The underground costs are based on contractor prices applied to a generic design and on benchmarking of similar operations.
- Cost and price assumptions: Costs are in US$. The open-pit costs are taken from the September 2026 optimization input workbook (Table 14-11) and, like the underground costs, are based on 2026 contractor rates and benchmarking against similar operations. The selling cost and haulage to mill are included in the open-pit cut-off but not in the underground cut-off. Including them would raise the underground cut-offs to about 1.98 g/t Au (break-even) and 1.48 g/t Au (marginal). The US$3,000/oz gold price is a long-term planning assumption below the spot price at the time of assessment
- Permitting and tenure: The existing Environment Act Licence 1207E relates to the historical underground operation and the existing plant. An open pit would require an amendment to reflect

the addition of open pit mining methods and expansion to 2,000 t/d. The RPEEE open pit shell lies almost entirely within Mineral Lease ML065. A sliver of approximately 0.1 ha on the western rim extends up to about 25 m beyond the lease boundary as mapped in iMaQs, onto the adjoining claim MB12247, which is also held by Minnova Corp. The iMaQs boundary is not a legal survey, and this overlap is within plausible digitizing error; it should be checked against the ML065 survey plan. The pit shell should also respect the required setbacks from water bodies, the tailings management area and the plant.

- In Mr. King's opinion, the 2026 MRE is a reasonable basis for the PEA mine plan.

25.4 Opportunities

- Down-dip extensions below the approximately 200 m depth limit of the 2025 to 2026 program.
- The PL North target, which is open along strike to the northwest and down dip.
- The Nokomis satellite deposit and the untested surface showings along strike.
- Possible grade upside from MSFA re-assaying of coarse-gold intervals.

25.5 Mining

In Mr. Oosthuizen's opinion, the combined open-pit (OP) and underground (UG) mine plan, the contractor mining costs and the production schedule are appropriate for a PEA. The PEA cash flow model carries no mine-air heating or backfill cost. The underground mining method, including options that require no backfill or only partial backfill, will be assessed in trade-off studies at the PFS stage. The underground mining study proposes a trial mine of about six months through the existing portal and decline before South-East pit mining reaches the portal; this trial mine is not included in the PEA mine schedule or cash flow model, and the QPs recommend that its timing, scope and cost, and its sequencing with the South-East pit, be assessed in the PFS.

25.6 Metallurgy and Processing

In Mr. Cunningham's opinion, the historical testwork and plant performance support a recovery of 90% provided that the operating problems of 1987 to 1989 are eliminated. The 21-month construction sequence and the ramp-up profile he has defined are slower than the base case, and the 2,000 t/d plant must be in place once throughput exceeds 1,000 t/d.

25.7 Infrastructure and Environment

The PEA general site layout is shown in Figure 18-1. Flotation tailings are stored in the Ragged Lake TMA behind a NAG waste rock embankment on perimeter B (crest 352.4 m, pond 351.4 m, enclosing 124.8 ha, storing 6.2 Mt). Sulphide tailings go to a lined cell outside the embankment. Waste rock is placed in a south-west WRD on ML065 (about 23.1 ha, 10.19 Mt capacity) and a north WRD on claim MIT-045 (about 57.4 ha,

29.74 Mt capacity), about 80.5 ha in total. Supporting infrastructure includes the truck shop, genset, temporary 70-person camp, LNG/CNG pad and power plant, 525 kL fuel farm, contractor yard, effluent treatment plant and discharge, security gate, explosives and detonator magazines, and the 25 kV line. The south-west WRD location, the foundation and drainage of both WRDs, any PAG cell liner, Crown tenure for the north WRD, and the fuel tank, magazine and reagent storage designs will be confirmed at the PFS stage (Item 18).

25.8 Economics

Cautionary statement. The preliminary economic assessment (PEA) is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

In Mr. Oosthuizen's opinion, the PEA shows positive economics at the base-case gold price of US$3,000/oz: after-tax NPV at 5% of US$180.6M, after-tax IRR of 111.0% and AISC of US$2,288/oz.

25.9 PEA Risks

Table 25-1: Principal PEA risks

Risk	Description	Mitigation
Plant ramp-up and schedule	Base case reaches 2,000 t/d in Month 8; the QP's sequence and a January 2029 first feed are slower	Ramp-up sensitivity on the QP's basis is run (C3 after-tax NPV at 5% US$159.8M); develop the execution schedule at the next study stage
Capital costs	QP estimates for the expansion, US$16.8M plus US$3.2M contingency (US$20.0M), and tailings (US$9.0M) exceed the model	Decision by the lead QP; update estimates at PFS
Uncosted items	Mine-air heating (about 11.7 MW thermal peak) and backfill are not costed	Cost at the next study stage
Royalty basis	Model deducts 3% of gross revenue; royalty is a 3% NSR sliding scale	Resolved: basis consistent with the filed MRE; the 3.0% rate applies at all PEA price cases and the gross-revenue basis is slightly conservative
Geotechnical and hydrogeological	Crown pillar, unfilled drift voids and inflows	PFS geotechnical and hydrogeological program
Tailings capacity and dams	The PEA stores 6.2 Mt of life-of-mine flotation tailings at a pond level of 351.4 m (crest 352.4 m, enclosing 124.8 ha). The 10 Mt in the Intent to Apply allows for possible future mine life beyond the PEA. Dam heights, foundations and consequence class are based on 30 m terrain data and an assumed High class	Lidar or ground survey, bathymetry, foundation investigation and dam-breach study; TMA staging design at the next study stage
Waste rock storage	The south-west WRD covers about 23.1 ha and holds about 10.19 Mt on ML065, and the north WRD about 57.4 ha holding about 29.74 Mt on claim MIT-045, about 80.5 ha in total, after 0.58 Mt is used in the TMA embankment. The south-west WRD location, the foundation and drainage of both WRDs, any PAG cell liner, and Crown tenure for the north WRD will be confirmed at the PFS stage.	Confirm the south-west WRD location, the foundation and drainage of both WRDs, any PAG cell liner, and Crown tenure for the north WRD at the PFS stage; include both WRDs in the Notice of Alteration.
Permitting	Open pit, expansion and Ragged Lake listing on MDMER Schedule 2	Submit Notice of Alteration and Alternatives Assessment Report
Power supply	Model assumes grid power from Month 37 (2031); Manitoba Hydro advises the Sherridon supply may take about two to three years to restore, at about 3 MW to 3.5 MW, below the project demand	Assess life-of-mine LNG generation and grid options at the next study stage
Inferred Mineral Resources	Part of the mine plan is Inferred	Infill drilling
Closure cost	The closure allowance in the financial model is a PEA-level estimate and may increase once a costed closure plan is prepared, covering about 643 ha of new disturbance, the pits, the WRDs and the expanded TMA.	Prepare a costed closure plan for the PFS and the Manitoba closure plan submission.

Source: ABGM.

25.10 PEA Opportunities

- Gold prices above US$3,000/oz.
- Ore sorting (testwork only at this stage) to reduce the plant expansion or raise mill head grade.
- An earlier grid connection, reducing LNG power costs.
- Nokomis as a future satellite source of plant feed. The contiguous Nokomis deposit lies less than 8 km from the PL mill (Item 23) and is not included in the PEA mine plan.

26 RECOMMENDATIONS

The QPs recommend the work program summarized in Table 26-1 (Mineral Resource and permitting program from the 2026 MRE) and Table 26-2 (PEA engineering program). Phase 2 of the Mineral Resource program depends on the results of Phase 1.

Table 26-1: Recommended Mineral Resource and permitting program

Phase	Activity	Budget (C$)
1	Assay the outstanding 2025 to 2026 samples and complete the resampling program; update the MRE: certify the database against laboratory certificates	100,000
1	Run a systematic MSFA (screen fire) check program on mineralized intervals, with an updated QA/QC review (CRMs, blanks, duplicates, umpire laboratory)	50,000
1	Run a bulk density program by domain and lithology to replace the single 2.81 t/m³ value	50,000
1	Verify the historical void model (survey records and probe drilling near the open-pit area)	50,000
1	Metallurgical testwork: variability composites, gravity-recoverable gold, flotation and cyanidation of the concentrates, arsenic deportment and detoxification, and ore sorting	250,000
1	Submit required permit amendments and updates (Notice of Alteration to Environment Act Licence 1207E) and Alternatives Assessment Report for Ragged Lake TMA listing on Schedule 2 of MDMER	250,000
1	Update the RPEEE inputs: geotechnical pit-slope recommendations, current mining-cost benchmarks, royalty and selling-cost treatment, and documented dilution, mining recovery and stope-dimension parameters; re-run the pit optimization and MSO (Oosthuizen)	50,000
2	Drill the down-dip extensions below 200 m and the PL North target to convert and expand the Mineral Resources	750,000
2	Nokomis: database compilation, infill drill program planning and permitting for 2027 drill program to update current Mineral Resource Estimate	50,000
2	Geotechnical and hydrogeological data collection to support future technical studies	100,000
	Total	**1,700,000**

Source: 2026 MRE technical report.

Table 26-2: Recommended PEA engineering program

Activity	Purpose	Budget
Hydrogeological and dewatering program (Water Hunters PFS scope)	Inflows, dewatering design and water balance	US$153,865 excluding VAT (about US$154,000; R2,437,575 source; US$176,944 including VAT), converted from ZAR at 0.0853 CAD/ZAR (November 5, 2026) and US$0.74 per C$1.00, as quoted
Geotechnical PFS program	Pit slopes, crown pillar and UG spans, backfill design	Included in the US$500,000 technical studies allowance in the financial model
Tailings and waste rock engineering	Lidar or ground survey and lake bathymetry; foundation investigation; dam-breach study and consequence classification; TMA staging, water balance and embankment design (perimeter B); sulphide TMA liner design; south-west and north WRD design, PAG cells, drainage and Crown land tenure	US$250,000 (tailings design allowance in the financial model)
Ventilation and mine-air heating design and costing	Heating, auxiliary fans and diesel particulate filters	Included in the US$500,000 technical studies allowance in the financial model
Process plant ramp-up and execution schedule; plant and tailings estimates	Reconcile the QP's estimates with the model	Included in the US$500,000 technical studies allowance in the financial model
Detailed execution and ramp-up schedule	Plant construction, commissioning and ramp-up schedule for the next study stage; the ramp-up sensitivity on the QP's basis is reported in Item 22	Included in the US$500,000 technical studies allowance in the financial model
Royalty, tax and cost base date confirmation	Close the open items in the cash flow model	Included in the US$500,000 technical studies allowance in the financial model
Pre-feasibility study	Next study stage	Included in the US$500,000 technical studies allowance in the financial model
Total		**About US$904,000 (US$500,000 technical studies, US$250,000 tailings design and US$154,000 hydrogeology)**

Source: Water Hunters (October 19, 2026); ABGM cash flow model (2026).

The QPs recommend that the project proceed to a PFS, with a recommended budget of C$1.7M for the Mineral Resource and permitting program (Table 26-1) and about US$0.9M for engineering studies, consistent with the allowances in the financial model (Table 26-2).

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28 DATE AND SIGNATURE PAGE

This Technical Report, entitled 'NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project', was prepared for Minnova Corp. with an Effective Date of **September 10, 2026** and a report date of **November 16, 2026**. It has been signed by the following Qualified Persons:

(Signed and sealed) _____

Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat.

Principal Geologist, A and B Global Mining (Pty) Ltd

Date signed: **November 16, 2026**

(Signed and sealed) _____

Ettienne Jaques Oosthuizen, B.Tech. (Mining Engineering), MIMMM, QMR

Principal Mining Engineer, A and B Global Mining (Pty) Ltd

Date signed: **November 16, 2026**

(Signed and sealed) _____

Gordon Ian Cunningham, B.Eng. (Chemical Engineering), Pr.Eng ECSA, FSAIMM

Associate Principal Process Engineer, A and B Global Mining (Pty) Ltd

Date signed: **November 16, 2026**

29 CERTIFICATES OF QUALIFIED PERSONS

29.1 Certificate of Qualified Person: Nicholas Ross King

I, Nicholas Ross King, of A and B Global Mining (Pty) Ltd, Building 4, Ground Floor, Princeton Office Park, Olievenhoutbosch Road and Bauhinia Street, Highveld, Centurion, Gauteng, 0157, South Africa, certify that:

1. I am employed as Principal Geologist by A and B Global Mining (Pty) Ltd.
2. This certificate applies to the technical report entitled 'NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project', with an Effective Date of **September 10, 2026** and a report date of **November 16, 2026** (the Technical Report), prepared for Minnova Corp.
3. I graduated with a BSc (Hons) in Geology from Rhodes University in 1996 and a Graduate Diploma in Engineering (Mining Engineering) from the University of the Witwatersrand in 2007.
4. I am registered as a Professional Natural Scientist (Pr.Sci.Nat.) with the South African Council for Natural Scientific Professions (SACNASP), registration no. 400166/07.
5. I have practised my profession for over 30 years. My relevant experience includes exploration and mine geology, geological modelling and Mineral Resource estimation, including as Chief Geologist at Gold Fields' South Deep Gold Mine from 2007 to 2017, where I was responsible for geological modelling and resource estimation for portions of the South Deep Mineral Resources and managed the brownfields surface exploration program; senior geologist roles at the Beatrix and Leeudoorn gold mines; drilling and database management; and the preparation of technical reports in accordance with mineral reporting codes.
6. I have read the definition of 'qualified person' in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). I certify that, by reason of my education, my affiliation with a professional association (as defined in NI 43-101) and my past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. I visited the PL Gold Mine from April 24 to 27, 2026.
8. I am responsible for Items 4 to 12, Item 14 (excluding subsection 14.8, Reasonable Prospects for Eventual Economic Extraction) and Item 23 of the Technical Report, and I have overall responsibility for the Mineral Resource Estimate used in the Technical Report. I share responsibility for Items 1, 2, 3, 24, 25, 26 and 27.
9. I am independent of Minnova Corp., the vendor and the property, applying all of the tests in section 1.5 of NI 43-101. I hold no shares, options or other securities of Minnova Corp., I am not a director, officer or employee of Minnova Corp., and I do not derive a material part of my income from Minnova Corp.

10. Before this Technical Report, my involvement with the property comprised supervising the 2026 MRE disclosed on September 10, 2026, acting as QP for the technical report 'NI 43-101 Technical Report and Mineral Resource Estimate for the PL Gold Mine, Manitoba, Canada' (Effective Date September 10, 2026), and the related QP review of the October 14, 2026 news release. I have had no other prior involvement.

11. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and to any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this **16th** day of **November**, 2026.

(Signed and sealed) _____

Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat.

29.2 Certificate of Qualified Person: Ettienne Jaques Oosthuizen

I, Ettienne Jaques Oosthuizen, of A and B Global Mining (Pty) Ltd, Building 4, Ground Floor, Princeton Office Park, Olievenhoutbosch Road and Bauhinia Street, Highveld, Centurion, Gauteng, 0157, South Africa, certify that:

1. I am employed as Principal Mining Engineer by A and B Global Mining (Pty) Ltd.
2. This certificate applies to the technical report entitled 'NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project', with an Effective Date of **September 10, 2026** and a report date of **November 16, 2026** (the Technical Report), prepared for Minnova Corp.
3. I graduated with a Bachelor of Technology (Mining Engineering) from the University of Johannesburg in 2006.
4. I am a Professional Member of the Institute of Materials, Minerals and Mining (MIMMM), membership number 685324, and hold the Institute's Qualified for Minerals Reporting (QMR) designation. I am also a member of the Southern African Institute of Mining and Metallurgy (SAIMM), membership number 7046076.
5. I have practised my profession for over 25 years. My relevant experience includes deep-level underground gold mining production at AngloGold Ashanti's Mponeng Mine from 1999, and, since joining A and B Global Mining in 2011, mine design, life-of-mine planning and scheduling, open-pit optimization, financial modelling, scoping and pre-feasibility studies, and Mineral Reserve estimation and reporting for open-pit and underground gold operations, including Harmony Gold's Kalgold Mine, Resolute's Bibiani Gold Mine, the Nyanzaga open pit, Chaarat Gold and Polyus Gold.
6. I have read the definition of 'qualified person' in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). I certify that, by reason of my education, my affiliation with a professional association (as defined in NI 43-101) and my past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. I visited the PL Gold Mine from January 25 to 28, 2026.
8. I am responsible for subsection 14.8 of Item 14 (Reasonable Prospects for Eventual Economic Extraction), Item 15, Item 16, Item 18, Item 19, Item 20, Item 21 (excluding the process plant and tailings capital and operating costs) and Item 22 of the Technical Report. I share responsibility for Items 1, 2, 3, 24, 25, 26 and 27.
9. I am independent of Minnova Corp., the vendor and the property, applying all of the tests in section 1.5 of NI 43-101. I hold no shares, options or other securities of Minnova Corp., I am not a director, officer or employee of Minnova Corp., and I do not derive a material part of my income from Minnova Corp.

10. Before this Technical Report, my involvement with the property comprised the pit optimization and MSO work for the 2026 MRE and the RPEEE parameters in the technical report 'NI 43-101 Technical Report and Mineral Resource Estimate for the PL Gold Mine, Manitoba, Canada' (Effective Date September 10, 2026), and acting as QP for the mining methods, mine plan, costs and economic analysis of the PEA announced on October 14, 2026. I have had no other prior involvement.

11. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and to any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this **16th** day of **November**, 2026.

(Signed and sealed) _____

Ettienne Jaques Oosthuizen

29.3 Certificate of Qualified Person: Gordon Ian Cunningham

I, Gordon Ian Cunningham, of Turnberry Projects (Pty) Ltd, Broadacres Estate, 2 District Drive, Merrivale Heights, HOWICK, 3291, SOUTH AFRICA., certify that:

1. I am employed as Associate Principal Process Engineer by A and B Global Mining (Pty) Ltd.
2. This certificate applies to the technical report entitled 'NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project', with an Effective Date of **September 10, 2026** and a report date of **November 16, 2026** (the Technical Report), prepared for Minnova Corp.
3. I graduated with a B.Eng. in Chemical Engineering from the University of Queensland in 1975.
4. I am registered as a Professional Engineer with the Engineering Council of South Africa (Pr.Eng ECSA), Registration No. 920082, and I am a Fellow of the Southern African Institute of Mining and Metallurgy (FSAIMM), Registration No. 19584.
5. I have practised my profession for 51 years. My relevant experience includes 19 years in metallurgical operations and production, 5 years in corporate consulting and 25 years as an independent consulting metallurgist.
6. I have read the definition of 'qualified person' in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). I certify that, by reason of my education, my affiliation with a professional association (as defined in NI 43-101) and my past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. **I visited the PL Gold Mine from January 25 to 28, 2026.**
8. I am responsible for Item 13 (Mineral Processing and Metallurgical Testing) and Item 17 (Recovery Methods) of the Technical Report, and for the process plant and tailings capital costs and the process plant operating costs in Item 21 (Capital and Operating Costs). I share responsibility for Items 1, 2, 3, 18, 20, 24, 25, 26 and 27.
9. I am independent of Minnova Corp., the vendor and the property, applying all of the tests in section 1.5 of NI 43-101. I hold no shares, options or other securities of Minnova Corp., I am not a director, officer or employee of Minnova Corp., and I do not derive a material part of my income from Minnova Corp.
10. Before this Technical Report, my involvement with the property comprised the process plant, ramp-up and recovery methods work for the PEA announced on October 14, 2026. I have had no other prior involvement.
11. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority, and to any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this **16th** day of **November**, 2026.

(Signed and sealed) _____

Gordon Ian Cunningham, B.Eng. (Chemical Engineering), Pr.Eng ECSA, FSAIMM

30 CONSENTS OF QUALIFIED PERSONS

30.1 Consent of Qualified Person: Nicholas Ross King

TO: British Columbia Securities Commission; Alberta Securities Commission; Ontario Securities Commission; TSX Venture Exchange

AND TO: Minnova Corp.

Subject: Consent of Qualified Person, NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project

Dear Sirs/Mesdames

I, Nicholas Ross King, BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat., consent to the public filing of the technical report titled "NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project" with an effective date of **September 10, 2026** and a report date of **November 16, 2026** (the "Technical Report"), prepared for Minnova Corp. ("Minnova").

I am responsible for Items 4 to 12, Item 14 (excluding subsection 14.8) and Item 23 of the Technical Report, and share responsibility for Items 1, 2, 3, 24, 25, 26 and 27.

I also consent to the use of extracts from, or a summary of, the Technical Report in Minnova's news release dated October 14, 2026 announcing the Preliminary Economic Assessment **[and any later news release to be listed]** (the "Disclosure").

I confirm that I have read the Disclosure and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this **16th** day of **November**, 2026.

_____

NICHOLAS ROSS KING

BSc (Hons) Geology, GDE (Mining Engineering), Pr.Sci.Nat. (SACNASP registration no. 400166/07)

PRINCIPAL GEOLOGIST

A and B Global Mining (Pty) Ltd

30.2 Consent of Qualified Person: Ettienne Jaques Oosthuizen

TO: British Columbia Securities Commission; Alberta Securities Commission; Ontario Securities Commission; TSX Venture Exchange

AND TO: Minnova Corp.

Subject: Consent of Qualified Person, NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project

Dear Sirs/Mesdames

I, Ettienne Jaques Oosthuizen, B.Tech. (Mining Engineering), MIMMM, QMR, consent to the public filing of the technical report titled "NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project" with an effective date of **September 10, 2026** and a report date of **November 16, 2026** (the "Technical Report"), prepared for Minnova Corp. ("Minnova").

I am responsible for subsection 14.8 of Item 14, Item 15, Item 16, Item 18, Item 19, Item 20, Item 21 (excluding the process plant and tailings costs) and Item 22 of the Technical Report, and share responsibility for Items 1, 2, 3, 24, 25, 26 and 27.

I also consent to the use of extracts from, or a summary of, the Technical Report in Minnova's news release dated October 14, 2026 announcing the Preliminary Economic Assessment **[and any later news release to be listed]** (the "Disclosure").

I confirm that I have read the Disclosure and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this **16th** day of **November**, 2026.

_____

ETTIENNE JAQUES OOSTHUIZEN

B.Tech. (Mining Engineering), MIMMM, QMR

MIMMM and QMR, IOM3 Membership No. 685324; Member, SAIMM, Membership No. 7046076

PRINCIPAL MINING ENGINEER

A and B Global Mining (Pty) Ltd

30.3 Consent of Qualified Person: Gordon Ian Cunningham

TO: British Columbia Securities Commission; Alberta Securities Commission; Ontario Securities Commission; TSX Venture Exchange

AND TO: Minnova Corp.

Subject: Consent of Qualified Person, NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project

Dear Sirs/Mesdames

I, Gordon Ian Cunningham, B.Eng. (Chemical Engineering), **Pr.Eng ECSA, FSAIMM**, consent to the public filing of the technical report titled "NI 43-101 Technical Report: Preliminary Economic Assessment, Minnova PL Gold Mine Project" with an effective date of **September 10, 2026** and a report date of **November 16, 2026** (the "Technical Report"), prepared for Minnova Corp. ("Minnova").

I am responsible for Item 13 (Mineral Processing and Metallurgical Testing) and Item 17 (Recovery Methods) of the Technical Report, and for the process plant and tailings capital costs and the process plant operating costs in Item 21, and share responsibility for Items 1, 2, 3, 18, 20, 24, 25, 26 and 27.

I also consent to the use of extracts from, or a summary of, the Technical Report in Minnova's news release dated October 14, 2026 announcing the Preliminary Economic Assessment **[and any later news release to be listed]** (the "Disclosure").

I confirm that I have read the Disclosure and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this **16th** day of **November**, 2026.

_____

GORDON IAN CUNNINGHAM

B.Eng. (Chemical Engineering), **Pr.Eng ECSA, FSAIMM**

[Professional registration body and registration number]

ASSOCIATE PRINCIPAL PROCESS ENGINEER

A and B Global Mining (Pty) Ltd