FEASIBILITY STUDY

PL GOLD PROJECT, MANITOBA, CANADA

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LIST OF ABBREVIATIONS

The metric system has been used throughout this report. Tonnes are metric of 1,000 kg, or 2,204.6 lb. Grid coordinates are given in the UTM NAD, latitude/longitude system or local mine grid.

All currency is in Canadian dollars (CAD) unless otherwise stated.

Abbreviation	Description
°	degree
%	percent
amsl	above mean sea level
ARD	acid rock drainage
Au	gold
C$	Canadian Dollar
CIL	carbon-in-leach
COG	cut-off grade
$CuSO_4$	copper sulfate
DAR	Developers Assessment Report
DFO	Department of Fisheries and Oceans, a Federal agency
EPCM	Engineering, Procurement and Construction Management
FOS	factor of safety
FS	Feasibility Study
ft	feet
G&A	General and Administrative
GAC	granulated activated carbon
Ga	billion years before present
g	gram
g/h	grams per hour
g/L	grams per liter
g/t	grams per tonne
gpm	gallons per minute
h	hour
Ha	hectare (10,000 m^2)
Hz	hertz
ID^2	inverse distance weighting
I/O	input and output
k	thousand
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
koz	thousand ounces
kPa	kilopascal
kt	thousand tonnes
kV	kilovolt
kVa	kilovolt ampere
kW	kilowatt
kWh	kilowatt hour
kWh/t	kilowatt hours per tonne

LIST OF ABBREVIATIONS
(continued)

L	liter
LAN	local area network
m	metre
m^2	square metre
m^3	cubic metre
mm	millimetre
Mm^3	million cubic metres
Ma	million years before present
MCC	motor control center
min	minute
Mt	million tonnes
MMER	Metal Mining Effluent Regulations
MVA	megavolt ampere
MW	megawatt
$Na_2S_2O_5$	sodium metabisulfite
NaOH	sodium hydroxide
NI 43-101	Canadian National Instrument 43-101
NN	nearest neighbor
oz	troy ounce
P&ID	process and instrument diagrams
PAG	potential acid generating
PFD	process flowsheet diagram
PLC	programmable logic controller
PoE	point-of-entry
psf	pounds per square foot
QA/QC	Quality Assurance/Quality Control
ROM	run-of-mine
RPM	revolutions per minute
SO_2	sulfur dioxide
st	short ton (2,000 pounds)
stpd	short tons per day
TMF	tailings management facility
t	metric tonne (dry)
tpd	tonnes per day
tph	tonnes per hour
t/m^3	tonnes per cubic metre
tpy	tonnes per year
µm	micron
US$ or USD	United States Dollar
V	volt
VFD	variable frequency drive
VOIP	voice-over internet protocol
WRF	waste rock facility

LIST OF ABBREVIATIONS
(continued)

wt%	weight percent
y	year

Conversion factors utilised in this report include:

- 1 troy ounce/ton = 34.2857 grams/tonne
- 1 gram/tonne = 0.0292 troy ounces/ton
- 1 troy ounce = 31.1035 grams
- 1 gram = 0.0322 troy ounces
- 1 pound = 0.4536 kilograms
- 1 foot = 0.3048 metres
- 1 mile = 1.609 kilometres
- 1 acre = 0.4047 hectares
- 1 square mile = 2.590 square kilometres

Unless otherwise noted, all UTM coordinates in this Report are in Zone 14U and the datum of NAD83.

1 SUMMARY

This Feasibility Study was performed by AMPL Professionals Ltd. ("AMPL") and CSA Global Pty Ltd. ("CSA"). CSA managed and is responsible for the geology and mineral resource estimates components while AMPL is responsible for all other areas except the environmental aspects of the Project, which were prepared for Minnova Corp. by its environmental consultants. The Report is specific to the standards dictated by National Instrument 43-101 ("NI 43-101"), companion policy NI 43-101CP and Form 43-101F1 (Standards of Disclosure for Mineral Projects – June 24, 2011) and to CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014).

The PL Mine Property is comprised of mineral resources adjacent to and along strike of the formerly producing Puffy Lake Mine. The Project is contained within the Maverick Gold Project that also includes the Nokomis Gold Deposit. Only the PL Deposit is assessed in this study.

The Project base of operations is located at latitude 55° 02' North, longitude 100° 58' West approximately 63 km northeast of the City of Flin Flon and 12 km southeast of the hamlet of Sherridon, in Manitoba. The Project can be accessed from Provincial Highway 10 south of Flin Flon by travelling 66 km on a year round gravel road that extends northwards toward the town of Sherridon and then east 9 km along the mine access road. In addition, charter floatplane and helicopter services are available at the Flin Flon airport.

The Project area is located at the site of the former gold producer, Puffy Lake Gold Mine, which operated between 1987-1989. The PL Property consists of seventeen (17) staked claims totalling 2961 hectares ("ha") and one mining lease (ML065), covering an area of 378 ha and requiring an annual lease payment of $3,969.

Existing and new mine development, processing plant and surface infrastructure facilities and equipment will be recommissioned where appropriate.

This Report presents:

1. An updated 2017 Mineral Resource Estimate for the PL Deposit on the PL Property (prepared by CSA Global);
2. The 2014 Mineral Resource Estimate for the Nokomis Deposit at the Nokomis Property in accordance with new NI 43-101 standards. (prepared by ACA Howe International, now part of CSA Global); and
3. A Feasibility Study of the PL Deposit conducted by AMPL in accordance to NI 43-101 standards.

The effective date of this Report is October 31, 2017. AMPL and CSA Global understand that the Company will use the Report internally for decision-making purposes and publicly in support of reporting obligations and possible corporate financing activities related to the Project.

1.1 PROPERTY LOCATION ACCESS AND DESCRIPTION

Minnova's 100% owned Maverick Project, consisting of the non-contiguous PL and Nokomis Properties, is located in northwestern Manitoba centred at 55° 02' North latitude, 100° 59' West longitude, approximately 63 km northeast of Flin Flon, Manitoba and approximately 12 km southeast of the community of Sherridon. The PL Deposit is situated on Mining Lease ML065, near 55° 02' 05" North latitude and 100° 58' 52" West longitude. The Nokomis Deposit is located in the vicinity 55° 04' 50" North latitude and 100° 52' 00" West longitude.

The PL Property comprises twenty-eight (28) staked mining claims totaling 5,114 ha and one mining lease totaling 378 ha. The Nokomis Property comprises twenty-seven (27) staked mining claims totaling 566 ha. In total, the Properties encompass an area of 6,058 ha or approximately 61 km^2.

The PL Property is subject to a net smelter royalty (NSR). The NSR varies with the price of gold and is 3.0% for a gold price greater than US$1,000/troy ounce, 2.5% for a gold price between US$750 and US$1,000, and 2.0% for a gold price under US$750. An underlying 2% NSR exists on the Nokomis Property.

The Project can be accessed via an all-weather gravel road that extends from Provincial Highway 10 south of Flin Flon northwards 78 km to the community of Sherridon. At approximately the 66 km marker, the PL mine-access road is taken east 9 km to the mine site. The mine-access road was constructed in 1986 and crosses the Keewatin Railway Company line at "Mile 33" ("Kilometre 52.8") from Sherritt Junction. The rail line passes approximately 6 km west of the PL Property. In addition, charter floatplane and helicopter services are available at the Flin Flon airport, which is serviced by commercial airlines from Winnipeg.

The Project is situated in terrain typical of the Canadian Shield with limited relief. Higher ridges expose extensive outcrop and are interspersed with lakes and low-lying swampy areas. The elevations on the property range from approximately 350 m above sea level at the mill site to 340 m at Puffy Lake.

1.2 PREVIOUS MINING AND PROCESSING OPERATION AT THE PL PROPERTY

Mining and processing of mineralized material had been previously carried out at the PL deposit by Pioneer Metals Corporation between December 1987 and March 1989. The Puffy Lake Mine operated as a ramp access, shallow angle stoping mine. The ramp was developed to a depth of approximately 130 m. Mine production problems led to placing the milling operation on care and maintenance in April 1989. The concentrator equipment was prepared for extended care and maintenance. Since that time, the power transmission line to the concentrator has been damaged by forest fires and parts of the concentrator have been vandalized with the removal of wire. Much of the major equipment inside the mill is in a condition suitable for refurbishing.

No mining or processing of mineralized material has been carried out at the Nokomis Property.

1.3 GEOLOGY AND MINERALIZATION

The PL deposit is located at the boundary of the Kisseynew and the Flin Flon domains of the Paleoproterozoic (Precambrian) Trans-Hudson Orogen. Litho-tectonic units in the area have been metamorphosed to middle and upper amphibolite facies.

The Flin Flon Domain forms a generally east trending 230 km by 80 km belt of complexly folded and metamorphosed volcanic, sedimentary, and intrusive rocks. The belt is bound to the north by the Kisseynew Domain, to the south by Paleozoic rocks, to the east by Archean rocks of the Superior Province, and to the west by the Tabbernor Fault and granitic rocks of the Glennie Domain.

The area around the PL and Nokomis Deposits is underlain by a sequence consisting of supracrustal rocks of the Amisk (Flin Flon arc assemblage), Burntwood and Missi groups, and granitoid gneisses of the Sherridon-Hutchinson Lake Complex.

Gold mineralization at the PL deposit is controlled by four main shear zones named the Sherridon, Upper, Main, and Lower zones. The shear zones and contained mineralization subcrop under a northwest-southeast-trending swamp and have been outlined by surface diamond drilling to a vertical depth of 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.

The Nokomis deposit is a shear related, intrusive hosted, lode gold system. Gold mineralization occurs within a differentiated gabbroic sill emplaced near the contact between metasediments of the Burntwood Suite - Nokomis Group and mafic volcanics of the Amisk Lake group. The Nokomis Deposit is made up of the mineralized Upper and Lower Host zones that have been separated by a steeply dipping normal fault that is intruded by pegmatite. The Upper and Lower Host zones range in thickness from <1 m to 4 m and have been intersected by diamond drilling over a surface strike length of approximately 450 m and a down dip extent of more than 350 m, to a vertical depth of 200 m below surface.

1.4 MINERAL RESOURCE ESTIMATES

This Report presents an updated independent Mineral Resource Estimate for the PL Gold Deposit and the 2014 mineral resource estimate for the Nokomis Deposit, both located on its 100% owned Maverick Gold Project.

The updated mineral resource estimate for the PL Deposit is Mineral resources are inclusive of Mining Reserves and is presented in Table 1.1.

Table 1.1: PL Deposit Mineral Resource Estimate as of October 31, 2017				
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 PL Deposit:

1. *The volume of the historical mined areas was depleted from the resource estimate.*
2. *Capping values range from 30 to 45 g/t Au and affected 16 samples*
3. *Bulk densities of 2.81 t/m^3 were used for tonnage calculations at the PL Deposit.*
4. *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%.*
5. *Tonnes and ounces have been rounded to reflect the relative accuracy of the mineral resource estimate; therefore, numbers may not sum precisely.*
6. *1 troy ounce equals 31.10348 grams.*
7. *Mineral Resource tonnes quoted are not diluted.*
8. *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.*
9. *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.*

The 2017 MRE was prepared by Mr. L. McGarry, CSA Senior Resource Geologist and a Qualified Person ("QP") for the reporting of Mineral Resources, as defined by NI 43-101 and was prepared in accordance with CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (CIM Council, 2003), CIM "Definition Standards for Mineral Resources and Mineral Reserves" and reported in accordance to NI 43-101.

The MRE study relied on historical exploration data collected by previous operators, including information for forty-eight (48) drill holes completed by Minnova, one hundred six (106) drill holes completed by Auriga, and four hundred sixteen (416) historical diamond drill holes. Drill data was provided by Minnova in the form of Microsoft Access™, a digital data export. Digital models of underground workings and topography were provided in AutoCAD™ files.

Using Micromine™ Version 2014, assay intervals were manually assigned vein set domains. Intervals were selected to exceed a notional cut-off grade of 1 g/t gold over at least 1.5 m true thickness. Where this grade and thickness could not be achieved, a 1 m interval of less than 1 g/t Au was assigned to domain to honour interpreted vein continuity. For each vein set, hanging wall and footwall contact depths were contoured using functions in Leapfrog Geo Version 4, to generate closed wireframe solids that were snapped to drill hole intervals. Fourteen (14) domains were modelled with variable extents, volumes, and drill densities. Of these, six (6) were considered major domains having between 232 and 636 samples assigned to each. Eight (8) minor domains are also modelled with between 15 and 98 samples assigned to each.

To ensure equal sample support, domained assays were regularized to the dominant assay interval length of 1 m using length weighted averages of gold grades. A nominal grade of 0.001 g/t Au was used to populate un-sampled assay intervals. Grade capping analysis was performed on composited gold assay data to assess the influence of extreme grade outliers on sample population statistics. Capping values ranged between 30 and 45 g/t Au and affected a total of 16 samples. Directional variograms were generated for gold mineralization in the six (6) major domains. Anisotropy ellipses aligned with the overall orientation of each domain; longest axis dimensions ranged from 80 m to 150 m.

Ordinary block kriging and Inverse Distance Weighting were used to estimate gold grades into a block model constrained by domain wireframe models. An average bulk density of 2.81 g/cm^3 was used for tonnage estimates.

Resource classification parameters are based on the validity and robustness of input data and the QP's judgment with respect to the proximity of resource blocks to sample locations. Inferred and Indicated and Measured Mineral Resources were identified. To define reasonable prospects of economic extraction, an operating cost of CAD$125 dollars per tonne and a gold price of US$1,250 were used to derive a cut-off value of 2.5 g/t Au. A minimum mining thickness of 1.5 m was selected for the reporting of Mineral Resources.

This Report presents a summary of the 2014 Nokomis MRE study, which has an effective date of April 15, 2014 and was carried out by Mr. McGarry, P.Geo., a Qualified Person under NI 43-101, using information and data supplied by Minnova. The Nokomis MRE is not considered in this Feasibility Study and currently represents a small proportion of the mineral resource attributable to the Project.

CSA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that could potentially affect these Mineral Resource estimates. Mineral Resources may be affected by future engineering feasibility study's assessment of mining, processing, environmental, permitting, taxation, socio-economic, and other factors.

1.5 MINERAL RESERVES

The underground mine will be accessed via a mine portal and main ramp that will connect mine levels on the footwall side of the orebody. The sub-levels will be spaced at every 35 m vertical elevation. The deposit will be mined using an up dip stoping method utilising alimak and slusher units for the mining of the stopes due to the approximate 30 degree dip of the ore zone with a mining rate of approximately 590 tpd.

The underground cut-off grade for stopes is based on operating costs for mining, processing, tailings, surface, and general and administrative (“G&A”) costs, which total $184.85 per tonne, as derived in the sections that follow. The underground cut-off grade is 4.0 grams Au/tonne.

Based on the proposed mining method, a minimum mining height (ore thickness) of 1.5 m has been determined, which allows for breaking of the stope rock and slushing of the broken ore out of the stope. This minimum mining height has been applied to the resource calculations and included in the reported Measured and Indicated Resources for reserves determination.

Additional mining dilution was added based on the mining height of the stope. Different dilution factors were applied for stopes of the minimum 1.5 m height, stopes with a mining height between 1.5-2.4 m and stopes with heights greater than 2.4 m. The 2.4 m height increment is based on this being the height of the in stope raise that will be developed from the Alimak. The raise may contain all waste or ore or a combination of ore and waste, which will cause different overall stope dilution to occur. The different stope dilution factors at zero grade are:

- Minimum Mining Height – 1.5 m 9%
- Stopes of Height – 1.5-2.4 m 15%
- Stopes of Height– Greater than 2.4 m 8%

The overall weighted average dilution of reserves is approximately 13%.

Based on the underground mining cut-off and mine plan the underground reserves are shown in Table 1.2.

Table 1.2: Underground Mining Reserves		
Category	**Tonnes**	**Grade (g Au/t)**
Proven	367,458	7.77
Probable	585,269	6.52
Total Reserves	**952,727**	**7.00**

The open pit cut-off grade is based on surface mine operating costs, processing, tailings, surface and G&A costs, which total $73.17 per tonne. The open pit cut-off grade is 1.6 grams Au/tonne.

Dilution of 20% at zero grade has been included in the open pit reserves.

Based on the open pit cut-off grade, optimization, and mine plan the open pit reserves are presented in Table 1.3.

Table 1.3: Open Pit Mining Reserves		
Category	**Tonnes**	**Grade (g Au/t)**
Proven	87,145	3.99
Probable	226,566	4.49
Total Reserves	**313,711**	**4.35**

The total underground and open pit reserves are 1,266,438 at a grade of 6.34 grams Au/tonne.

1.5.1 Existing Infrastructure

Between May 1986 and April 1989, considerable mining and processing activity took place at the PL Property. Much of the past facilities are still in place though they have been vandalized over the years. The existing site comprises:

- Access road and powerline requiring repair and upgrades;
- Underground portal, ramp, drifts, raises, and stopes requiring dewatering and rehabilitation;
- Processing plant requiring upgrades, rehabilitation and reinstallation of all main power cabling;
- Surface maintenance shop, warehouse and administration offices requiring refurbishment;
- Small exploration camp;
- New core shed; and
- Previous tailings management facility.

This infrastructure remains intact on a care and maintenance basis. The decline ramp and underground workings are currently flooded with water.

As can best be determined (Pioneer monthly and other reports), some 350,000 tonnes of ore was mined from the various stopes during the previous mining operations.

1.5.2 Mining

1.5.2.1 Underground Mine

The known mineralization at the PL Deposit consists of five parallel gold-bearing veins that strike N30°W and dip at approximately 30 degrees to the northeast. The zones are designated from top to bottom, as the Sherridon, Upper, Main, Lower, and Lower 2 zones. The mineralized veins are tens of centimetres thick and would be mined in units. Mining heights would be from the minimum mining height of 1.5 m to upwards of approximately 2-3 m.

The old portal, existing ramp, and the new ramp extension continuing from the existing ramp will provide the main access to the mine. As the ramp progresses downwards vertically, mining levels would be established at required elevation intervals. All men, equipment, and materials will be transported into and from the mine via this main ramp. All ore and waste (as required only) will be transported in underground diesel powered haul trucks operating in the underground drifts and the main ramp.

On each level, the mining areas would be accessed from the main ramp by a 4.0 m high by 5.5 m wide access drift driven in the ore zone and parallel to the strike of the ore zones. The wider access drift would allow for material slushed from the stopes to sit on the hanging wall side of the drift while still allowing equipment access along the footwall side of the drift. The proposed mining method is up dip panel stoping utilizing Alimaks. Stoping will take place in panels, which are nominally 20 m wide (along strike) and extend lengthwise up dip over vertical intervals of approximately 35 m (producing stopes of 60-70 m length).

A pilot raise for drilling in the middle of each stope will also facilitate sufficient amounts of hanging wall ground support to prevent caving of the hanging wall and resulting ore losses. Each pilot raise will be driven 10 m beyond the up dip access drive to allow for easier Alimak set-ups for the next lift of mining. Stopes are mined the full height of the ore between levels, before backfilling. This results in optimum productivities, lower mining costs, and the minimum number of operating stopes to meet daily production targets. Mined out areas will be backfilled with primarily cemented and uncemented hydraulic backfill.

Underground development, including excavation of ramps, accesses, and haulage drifts, will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos, load-haul-dump (LHD) units, 30 and 40 tonne haul trucks, and scissor lifts with work platforms. Mining will utilize diesel powered, track and rubber tired mobile equipment including a single boom extension rods drill, small ANFO loading units, LHDs, and haul trucks.

Open pit mining by a contractor will utilize relatively smaller diesel operated blast hole drills, hydraulic excavators, haul trucks, and support services equipment.

Underground mining of the mineralized zones will be at a proposed rate of approximately 590 tpd or 216,000 tpy. After the first year of production, processing plant feed will be supplemented by open pit production of approximately 400 tpd with yearly production of from approximately 68,000-111,000 tpy, over 4 years. After open pit mining is completed, the mine feed will be supplied solely from the underground mine at 590 tpd for the remainder of the mine life.

Underground infrastructure will include:

- Breakdown maintenance shop;
- Fuel stations;
- Explosives and detonator magazines;
- Refuge stations;
- Main dewatering sumps;
- Main storage areas;

- Latrines;
- Electrical substations; and
- Mine wide wireless communication and control system.

Mine surface support facilities located in the area of the portal will include a surface ventilation fan set-up, backfill plant, maintenance shop, explosives magazines, mine rescue station, power substation, compressor station, small warehousing facility, laydown yard, and a water storage pond.

1.5.3 OPEN PIT

The underground mine design will be impacted by the potential portion of the PL Deposit mineable by open pit(s). Open pit mining will not commence until Year 2 to allow time for permitting of on-site, open pit mining. The PL pits were limited to a bottom elevation of 30-40 m depth (300 m true elevation). Underground stopes located below the open pits would be mined and backfilled prior to mining of the open pits and a small pillar left in the floor of the final pit above the backfilled open stopes.

The open pit optimization and design work resulted in five (5) small open pits to be developed to approximately 40 m depth. The open pits would be mined by a contractor.

The underground mine development schedule includes rehabilitation of existing ramp and lateral development and new waste and ore development. The development schedule ensures development is in place approximately one year before ore zone stope development and mining is required.

The mine production schedule is based on mining 600 tpd of reserves, for 360 days per year. Table 1.4 presents the development schedule for life-of-mine.

Table 1.4: Life-of-Mine Underground Development Schedule

Heading	Quantity	Units	Year -1	1	2	3	4	5	6	7	8	9	10	Total
Existing Development Rehabilitation														
Ramp	1,000	metres	200		800									**1,000**
Levels	400	metres			200	200								**400**
Lateral Development - Waste														
Spiral Ramp Surface to 145 Level	1,160	metres	580	580										**1,160**
Spiral Ramp 145 to 530	1,680	metres		250	300	600	530							**1,680**
40 Level	940	metres	100	300	460	80								**940**
75 Level	730	metres	100	300	230	100								**730**
110 Level	730	metres			200	300	230							**730**
145 Level	1,110	metres		200	910									**1,110**
180 Level	1,130	metres				1,000	130							**1,130**
215 Level	900	metres				100	800							**900**
250 Level	800	metres				260	540							**800**
285 Level	200	metres					200							**200**
320 Level	200	metres					200							**200**
355 Level	200	metres					200							**200**
Intake Ventilation Connection Drifts	300	metres		60	60	90	90							**300**
Exhaust Ventilation Connection Drifts	300	metres		60	60	90	90							**300**
Pre-Production Development - Ore														
75 Level	200	metres	200											**200**
40 Level	100	metres	100											**100**
Raise Development														
Intake Raise 1 - 145 to Surface	150	metres	70	80										**150**
Intake Raise 1 - 495 to 145	210	metres			74	74	62							**210**
Exhaust Raise 1 - 145 to Surface	150	metres		36	36	78								**150**
Exhaust Raise 1 - 495 to 145	210	metres			74	74	62							**210**
Boreholes														
Backfill Boreholes	360	metres		90	90	90	90							**360**
Drainholes	360	metres	70	20	90	90	90							**360**
Total Existing Development Rehab	**1,400**	**metres**	**200**	**0**	**1,000**	**200**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,400**
Total Lateral Development	**10,680**	**metres**	**1,080**	**1,750**	**2,220**	**2,620**	**3,010**	**0**	**0**	**0**	**0**	**0**	**0**	**10,680**
Total Raise Development	**720**	**metres**	**70**	**116**	**184**	**226**	**124**	**0**	**0**	**0**	**0**	**0**	**0**	**720**
Total Boreholes	**720**	**metres**	**70**	**110**	**180**	**180**	**180**	**0**	**0**	**0**	**0**	**0**	**0**	**720**

The production schedule is derived from scheduling all the stopes, which meet the 4 grams Au per tonne cut-off grade. Year 1 initial stoping started on the 75 and 110 Levels working up to the 40 Level, as this minimizes pre-production development, while accessing more than one year of production at grades significantly above the mine average grade. In subsequent years, stoping blocks between 2 or 3 levels were scheduled generally moving from the top down. Pillars in between the blocks would be removed at the end of each block life. Table 1.5 presents the summary production schedule.

Table 1.5: Summary Mine Production Schedule

Mining	Year 1		Year 2		Year 3		Year 4		Year 5		Total	
	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade
U/G Level												
40	97,740	7.54	105,967	6.85	15,962	4.73					219,669	7.00
75	118,260	8.36	20,964	5.80	29,022	4.87			8,846	4.25	177,092	7.28
110			20,936	10.63	29,643	9.26	13,968	5.05	11,650	4.48	76,196	8.13
145			68,133	8.19	33,125	6.30	36,946	5.40	22,581	4.16	160,784	6.60
180					67,015	6.70	89,367	5.67	28,195	4.44	184,577	5.86
215					29,592	10.17	17,394	5.82	8,052	4.49	55,038	7.97
250					11,641	13.06	13,783	7.53	1,914	4.05	27,339	9.64
285							11,354	10.67	873	4.46	12,228	10.22
320							5,516	4.72	5,635	4.24	11,151	4.47
355							27,672	7.25	981	4.03	28,653	7.14
Total U/G Mining	**216,000**	**7.99**	**216,000**	**7.54**	**216,000**	**7.42**	**216,000**	**6.16**	**88,727**	**4.34**	**952,727**	**7.00**
Open Pit												
1 - Ore									38,441	3.44	38,441	3.44
1 - Waste			0		0		0		228,338		228,338	
2 - Ore			59,516	4.32	20,000	4.96	20,000	4.96	18,167	4.96	117,683	4.64
2 - Waste			404,710		136,000		136,000		123,533		800,243	
3 - Ore			4,046	3.36	20,000	4.24	26,176	4.24	45,205	4.24	95,428	4.20
3 - Waste			43,903		217,000		284,010		490,476		1,035,389	
4 - Ore									6,384	9.28	6,384	9.28
4 - Waste			0		0		0		44,148		44,148	
5 - Ore			4,046	4.08	27,609	4.08	21,433	4.08	2,688	3.60	55,776	4.06
5 - Waste			37,830		258,144		200,399		25,133		521,506	
Total O/P Ore Mining			**67,609**	**4.25**	**67,609**	**4.39**	**67,609**	**4.40**	**110,885**	**4.36**	**313,711**	**4.35**
Total O/P Waste Mining			**486,444**		**611,144**		**620,408**		**911,628**		**2,629,624**	
Total Mine Production	**216,000**	**7.99**	**283,608**	**6.75**	**283,609**	**6.69**	**283,609**	**5.74**	**199,612**	**4.35**	**1,266,438**	**6.34**

Surface facilities will generally be centred near the portal or processing plant.

Surface support facilities will include explosives magazines, mine supervision, geology, engineering and administration offices and mine change house, power substation, compressor station, warehouse and laydown yard, and water collection ponds.

Mine manpower, included in the operating costs, totals forty-five (45) people during pre-production to an average steady state of approximately one hundred (100) personnel at full production. Staff comprise sixteen (16) people of the total production period mine manpower complement and hourly employees make up the remainder.

1.5.3.1 Processing

The processing plant, crushing complex, fine-ore bin, grinding complex, and concentrator/recovery plant site is compact and well-planned. The forest fire of 1989 did not affect the concentrator building and associated structures. The processing plant building remains in good structural condition with minor repairs to cladding and the roof required. The steel structure and concrete foundations remain in very good condition. A majority of the equipment is in good condition and will be recommissioned for the mill process circuit. Some equipment has been cannibalized or destroyed by vandals and requires replacement. Some equipment has been removed, primarily in the refinery area, and will be replaced with equivalent or better technology. The equipment's main power feed electrical wiring has been removed from the processing plant building. The motor control centre has been vandalized and will require upgrading and replacement.

The past producing mine included a conventional processing plant comprising crushing, grinding, gravity concentration (jig) flotation, Merrill Crowe, leaching, and refining for gold recovery to doré bars. The original process flowsheet will be essentially replicated with some additions of up to date technology and equipment. Analysis of the existing grinding mills in the plant indicate that the actual processing plant capacity is approximately 750-800 tpd. To achieve the rate of 1,000 tpd when open pit mining is anticipated would require the installation of an additional ball mill, which could potentially be accommodated relatively simply into the existing plant.

Ore from the underground mine will be delivered to the crusher dump hopper. A primary Jaw crusher will reduce the size of the run-of-mine ore to a nominal 120 mm. A cone crusher operating in closed circuit with a double deck vibrating screen will further reduce the size of the ore to 13 mm. A fine ore bin with a 1,500-tonnes live capacity will be used for ore storage prior to rod milling.

Ore from the fine ore bin will feed the comminution circuit, which consist of a primary rod mill and a secondary ball mill. The ore will be ground to a final product size of P80 of 100 microns. A Knelson concentrator will be incorporated in the grinding circuit to recover coarse free gold, which will be treated using an in-line-leach reactor. The leachate from the ILR circuit will be directed to the Merrill-Crowe circuit while the tailings will be sent back to the grinding circuit.

The rod and ball mills will operate in closed circuit with cyclones. The overflow from which will feed the flotation circuit that will consist of roughers, scavengers, and cleaner cells. The final flotation concentrate will feed cyclones operating in a closed circuit with a re-grind mill. Cyclones overflow will feed a thickener. Thickened pulp will constitute the feed to the cyanide circuit. Scavenger flotation tailings will be used either to produce backfill or be pumped directly to the tailing impoundment area. The thickened flotation concentrate will be subjected to cyanidation utilizing four leach tanks operating in series. The residual leached tails will be sent to two drum filters also operating in series in order to wash the pulp and remove gold in solution. The final re-pulped filter cake will be pumped along with the flotation tailings to the tailings impoundment area.

The Merrill-Crowe process will be used to recover gold from the pregnant solution generated from the leaching process. Precipitate from the Merrill-Crowe process will be recovered using press filters and sent to the refinery for smelting using an induction furnace in order to produce doré bars. A portion of the barren solution will be treated with hydrogen peroxide in order to destroy residual cyanide in solution and sent to the final tailings pump box for disposal to the TMF.

Based on the metallurgical testwork and process presented in the previous sections, the expected gold recovery is 90% with a plant operating availability exceeding 90%.

The past employed Ragged Lake Tailings Management Facility, which is presently in the process of being added to Schedule 2 of the Metal Mining Effluent Regulations ("MMER"), designating the area as a tailings disposal area, will be recommissioned. This tailings disposal area requires addition, before use, to the MMER Schedule 2, as the regulations were not in effect when the original mine was in production.

The tailings area encompasses a total area of approximately 0.5 km^2 and can hold 1 million m^3 of tailings and water at its present elevation of 346.0 masl. This figure will increase to approximately 2.3 million m^3 once the required control structures and dams to raise the operating level of the tailings basin to 348.0 masl are built. There are three additional structures to be built to control water flows from the south end and the west side of the lake.

The processing plant will be operated on two (2) shifts by eight (8) people, as shown in Table 17.3. The processing plant staff will total nine (9) people.

1.5.3.2 Support Facilities

The PL Mine, as a past producing mine, has significant infrastructure in place which will require refurbishing and upgrades as well as additions and expansions to the existing facilities.

Infrastructure required for the PL Mine will include:

- Upgraded all season 9 km gravel access road;
- Site road upgrades;
- Completion of power supply and distribution lines refurbishment;
- Recommission of the main power substation and MCC room and installation of a new crusher building substation and MCC room;
- Refurbishment of the mechanical workshop, office, dry complex, assay laboratory facilities, and other service areas and equipping of these facilities;
- Expand the exploration camp to thirty-two (32) person capacity (first 6 months of pre-production period) and complete expansion of the camp to accommodate 150 people for operations period;
- Installation of a potable and service water supply and treatment facilities; and
- Installation of sewage treatment facilities for the main camp and the mill/shop/office complex.

Most employees will live in camp and be rotated in and out using charter air services. Employees will be expected to arrive in Winnipeg, where the charter flights originate and will be flown to Flin Flon. From the airport everyone will be bussed to the mine. Charter flights will be flown once per week.

The general and administration manpower requirements are estimated to be fifteen (15) staff and five (5) surface department personnel.

1.5.4 ENVIRONMENTAL AND SOCIAL CONSIDERATIONS

Proposed initial production from underground operations only, at a rate of less than 600 tpd does not require any additional permits and is not considered a designated project under item 16(c) of the Schedule to the Regulations. As a result, the Company is not required to submit a Project Description under the Canadian Environmental Assessment Act (CEAA).

An application has been made to Fisheries and Oceans Canada ("DFO") to have Ragged Lake TMF added to Schedule 2 of the MMER. The draft application was submitted to DFO in May 2017 and returned in June 2017 for revisions. A revised application is in process and expected to be filed in January 2018. A decision is expected within 12 months of acceptance by DFO of the revised application.

Minnova has continued its dialogue about the status of the PLGP with First Nations and local communities and businesses.

AECOM (2014) reports that the Mathias Colomb Cree Nation (MCCN), located approximately 73 km north of Sherridon at the community of Pukatawagan, is the closest First Nation community to the Project area. Pukatawagan, which had a population of 1,826 in 2011 (1,478 in 2006), is accessible year-round by air and by rail, and by winter road for a period of approximately three (3) months of the year, depending upon conditions.

AECOM (2014) also reports that the Metis communities of Sherridon/Cold Lake had a population of one hundred thirteen (113) people in 2001 according to census information. Sherridon is located about 156 km north of The Pas adjacent to the First Nations-owned Keewatin Railway Company (KRC) rail line that extends from Sherritt Junction to Pukatawagan. The rail line crosses the mine access road about 6 km from the mill. Sherridon is an incorporated community under the Province of Manitoba's Northern Affairs Act and is administered by a mayor and council. The Cold Lake settlement is located approximately 1.5 km west of Sherridon.

Other First Nations, located within a similar distance to the site, include the Opaskwayak Cree Nation at Opaskwayak (86 km); the Nisichawayasihk Cree Nation at Nelson House (150 km); the Mosakahiken Cree Nation at Moose Lake (150 km); the Cross Lake First Nation at Cross Lake (195 km); and the Norway House Cree Nation at Norway House (240 km) (AECOM, 2014).

Additional stakeholder consultation will be conducted, as more details are developed regarding mine development.

1.5.5 Capital Expenditures

The estimated total Project pre-production capital expenditure, inclusive of contingencies and excluding working capital, is approximately $34.3 million. The total expenditures include EPCM, contractor overheads, and a 10% contingency on all estimated expenditures. A summary of Project pre-production capital expenditures is presented in Table 1.6. A working capital allowance of $1.1 million is estimated to be required.

Table 1.6: Project Pre-Production Capital Expenditures

CAPEX Cost Centre	Total Cost ($)	Year -2	Year -1	Total ($)
Mine	$11,492,000		$11,492,000	$11,492,000
Processing Plant	$7,845,000		$7,845,000	$7,845,000
Infrastructure	$5,301,000		$5,301,000	$5,301,000
Tailings Management Facility	$1,000,000		$1,000,000	$1,000,000
Owners Costs	$6,170,000		$6,170,000	$6,170,000
Contingency	$2,515,000		$2,515,000	$2,515,000
TOTAL CAPEX	**$34,323,000**		**$34,323,000**	**$34,323,000**

A working capital allowance, in addition to capital expenditures of $1.1 million, has been included in the cash flow model. This represents approximately 1 month of operating costs, which will be incurred before the first revenue is realized. The working capital requirement is less than will normally be expected, as payment for mine product will be immediately after the concentrate has been shipped.

Sustaining capital expenditures are estimated to be $53.7million mainly related to ongoing mine development and tailings management and water treatment facilities improvements.

Closure costs have been estimated at $1.5 million at the end of the Project life and is included as a separate line item in the cash flow model.

1.5.6 Operating Costs

The estimated total average operating cost (excluding smelting and refining) for the PL Mine is approximately $194.08 per tonne ore and $163.44 for per tonne ore for underground and open pit mining, respectively. Table 1.7 presents a summary table of life-of-mine average operating costs for each department on a cost per tonne of ore basis.

Table 1.7: Project Operating Costs Summary

Department	Underground Cost ($/t Ore Mined)	Open Pit Cost ($/t Ore Mined)
Mining	111.68	81.04
Processing and Tailings	24.37	24.37
Surface Department, Environmental, and G&A	49.12	49.12
Royalty (2%)	8.91	8.91
Total	**$194.08**	**$163.44**

1.5.7 Financial Analysis

All gold produced will be sold to the Royal Canadian Mint or other refiner (such as Johnson Mathey) as is the common practice in Canada. The gold value will be paid upon receipt of refiner and referee assay results for doré bars. The price paid is based on LME's daily set gold prices for spot market sales.

For the purpose of this study, value added taxes and other taxes, along with import duty costs have not been included. Exploration costs and all costs associated with areas beyond the property limits have also not been included.

The expected cash flow estimates are calculated using the forecast mine plan, operating costs, and capital expenditures incorporating expected long-term metal prices based on the past 36-months of moving average prices for each metal, as follows:

Gold (US$/oz.)	$1,250

The discounted cash flow analysis uses 2017 Constant Canadian Dollar values.

A summary of the expected parameters used for the financial analysis is presented in Table 1.8.

Table 1.8: Expected Project Parameters	
Reserves – Underground Reserves – Open Pit	952,727 tonnes at a grade of 7.00 g Au/t 313,711 tonnes at a grade of 4.35 g Au/t
Estimated Mining Dilution: Underground Open Pit	 12% at 0% grade 20% at 0% grade
Projected Mining Recovery	91%
Payable Gold Produced	47,000 to 55,000
Pre-Production Capital Expenditures	$34.3 million
Working Capital	$1.1 million
Total Sustaining Capital Expenditures	$53.7 million
Closure Cost	$1.5 million
Estimated Operating Costs ($/tonne): Mining – Underground Mining – Open Pit Processing G&A Royalty (2%)	 $111.68 $81.04 $24.37 $49.12 $8.91
Life-of-Mine	4.4 years

Processing plant recoveries and concentrate grades are varied by year within the model based on the precious metals feed grades to the processing plant.

An NSR royalty of 3% payable on net revenue is included.

Costs for metal sales and shipping are included in the deductions that the refiner makes.

U.S. dollar denominated components are converted using C$:US$ exchange rate of 1:1.25.

The cash flow analysis excludes:

- Any element or impact of financing arrangements; and
- All exploration and acquisition costs incurred prior to the production decision.

In calculating the after tax returns, the Project is subject to Federal and Manitoba Corporate Income Tax and Manitoba Mining Tax.

The overall level of accuracy of this study is approximately ±10%.

The Project expected investment and returns, based on the expected cash flow parameters, are shown in Table 1.9.

Table 1.9: Expected Project Returns		
	Pre-Tax	**After-Tax**
Undiscounted Net Revenue	$376 million	$376 million
Undiscounted Total Cash Flow	$71 million	$47 million
NPV (5%)	$55.9 million	$36.7million
NPV (10%)	$44.3 million	$28.8million
IRR	65%	53%
Payback Period	1.2 years	1.2 years

Results indicate that at the expected parameters and metals prices, the Project is viable.

Several sensitivities on the Base Case scenario have been investigated to determine the effect on the key financial statistics, if increases and decreases of 5, 10, 15, and 20% occur to the following parameters:

- Mined Grade;
- Gold Price;
- Operating Cost; and
- Capital Expenditures.

The results of the sensitivity analysis are presented in Table 1.10 and Table 1.11.

Table 1.10: Sensitivity Analysis for After-Tax NPV									
Parameter	**After-Tax NPV$_{5\%}$ ($M)**								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade	-$24.8	-$9.2	$6.3	$21.7	$36.7	$51.6	$66.4	$81.3	$96.1
Gold Price	-$25.0	-$9.4	$6.2	$21.6	$36.7	$51.6	$66.5	$81.4	$96.3
Operating Costs	$77.3	$67.2	$57.0	$46.8	$36.7	$26.5	$16.2	$5.8	-$4.7
Capital Costs	$50.2	$46.8	$43.4	$40.0	$36.7	$33.4	$30.1	$26.8	$23.5

Table 1.11: Sensitivity Analysis for After-Tax IRR									
Parameter	**After-Tax IRR (%)**								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**

Mined Grade		2	22	39	53	68	81	94	107
Gold Price			16	37	53	65	76	87	98
Operating Costs	108	97	84	70	53	34	6		
Capital Costs	84	75	67	60	53	48	42	37	32

Figure 1.1 and Figure 1.2 show the sensitivity analysis in graphical form.

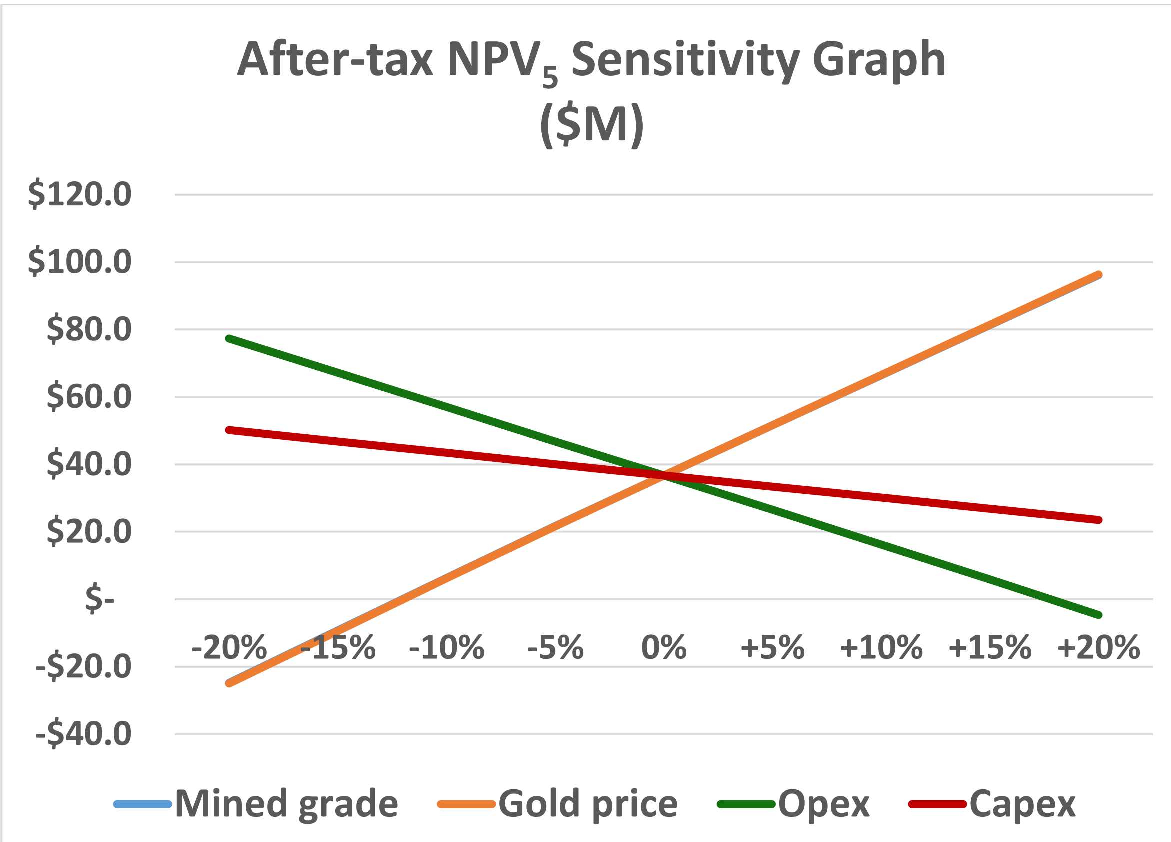


Figure 1.1 Graph of NPV sensitivity analysis

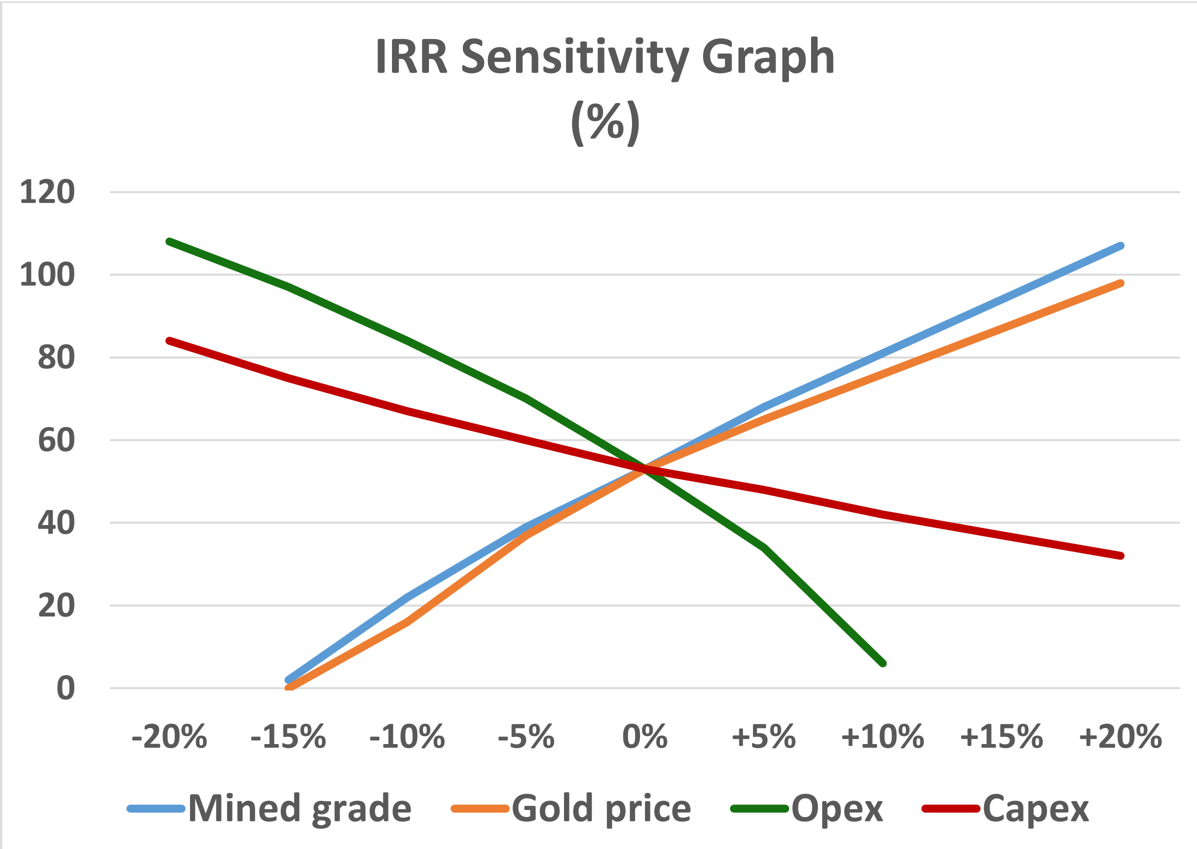


Figure 1.2 Graph of IRR sensitivity analysis

The sensitivity analysis shows that the Project economics are most sensitive to mined grade and gold price. The Project is least sensitive to capital expenditures.

1.6 INTERPRETATIONS AND CONCLUSIONS

The QP has reviewed the Maverick Project data provided by the Company, including drilling database; has visited the site; and has reviewed sampling procedures and security. The QP is of the opinion that the data presented by the Company are generally an accurate and reasonable representation of mineralization styles encountered at the Project.

It is the QP's opinion that Minnova's current sampling programs are conducted to industry standards and that the QA/QC programs undertaken by Minnova are sufficient to provide confidence in the analyses undertaken at the assay laboratories employed. Based on a review of available historical QA/QC data, and a positive comparison of the distribution of gold assay grades obtained by previous operators, the QP concludes that databases for the PL Deposit are of sufficient quality to provide the basis for the conclusions and recommendations reached in this Report.

Minnova's 2016 to 2017 in-fill drilling programs at the PL Deposit have provided further detail on the nature of the mineralized zones and permitted the completion of a NI 43-101 compliant updated Mineral Resource Estimate.

Minnova's 2017 summer exploration program has successfully discovered new gold showings in the PL Property package. The discovery of high-grade gold samples and their spatial distribution suggests exploration potential for additional gold mineralized zones in the PL Property and PL Deposit area.

The present resource model assumes that all of the previously mined stope and development openings have been properly located in three-dimensional space with respect to the resource model. There may be some error in the spatial locations still unknown but any required adjustments will have an equal chance of either positively or negatively affecting the final resource to reserve conversion ratio. Therefore, the probability that the overall quantity of presently identified tonnages and grades in the Mineral Reserves at PL will change is well within an acceptable margin of error to enable the Project economics to be estimated to the required degree of accuracy for a Feasibility Study.

The Mineral Reserves estimate for the PL Mine property amount to 1.3 million tonnes of ore at a gold grade of 6.34 g Au/t.

The mine plan as presented will support operation of the onsite processing plant for approximately 4.4 years at a nominal production rate of 590 mtpd or an average annual production rate of 216,000 tonnes.

The mine will require one year of pre-production development and construction prior to commencing production. The pre-production period will involve rehabilitating the portal, dewatering and rehabilitating approximately 200 metres of the existing ramp and new ramp, level and stope development in the new eastern zones of the mine. Infrastructure and supporting services will also be constructed and installed.

The proposed mining method is up dip stoping using an Alimak to develop and drill long holes for stopes. The stopes will utilize slushers to move the ore form the stopes to locations where LHDs and trucks will be used to move ore to surface.

Stopes will be backfilled with hydraulic tailings.

Open pit mining will take place from 5 small open pits to be developed to approximately 40 m depth. The open pits would be mined by a contractor.

Mine manpower, included in the operating costs, totals 45 people during pre-production to an average steady state of approximately 100 personnel at full production. Staff comprise 16 people of the total production period mine manpower complement and hourly employees make up the remainder.

The processing plant will include a conventional circuit comprising crushing, grinding, gravity concentration using a Knelson concentrator, flotation, Merrill Crowe, leaching, and refining for gold recovery to doré bars.

Based on the metallurgical testwork and process presented in the previous sections, the expected gold recovery is 90% with a plant operating availability exceeding 90%.

The past employed Ragged Lake Tailings Management Facility, which is presently in the process of being added to Schedule 2 of the Metal Mining Effluent Regulations (MMER), designating the area as a tailings disposal area, will be recommissioned.
The processing plant will be operated on 2 shifts 7 days a week requiring a total of sixteen (16) personnel. The processing plant staff will total nine (9) people.

The estimated total Project pre-production capital expenditure, inclusive of contingencies and excluding working capital, is approximately $34.3 million. The total expenditures include EPCM, contractor overheads, and a 10% contingency on all estimated expenditures. A working capital allowance of $1.1 million is estimated to be required. Sustaining capital expenditures are estimated to be $53.7million mainly related to ongoing mine development and tailings management and water treatment facilities improvements. Closure costs have been estimated at $1.5 million at the end of the Project life and is included as a separate line item in the cash flow model.

The estimated total average operating cost (excluding smelting and refining) for the PL Mine is approximately $194.08 per tonne ore and $163.44 for per tonne ore for underground and open pit mining, respectively.

The overall economic results indicate that the Project will have positive economic returns and generate approximately $47 million undiscounted after-tax cash flow ($71 million pre-tax) over the Project's 4.4 year mine life.

At the base case metal prices, the Project's after-tax net present value is estimated to be approximately $28.8 million at a discount rate of 10%. The post-tax IRR is estimated to be 53% and payback has been calculated at 1.2 years from start of production.

The sensitivity analysis shows that the Project economics are most sensitive to mined grade and gold price. The Project is least sensitive to capital expenditures.

1.7 PROJECT RISKS ASSESSMENT

1.7.1 GEOLOGY AND RESOURCE ESTIMATE

Technical factors which may affect the Mineral Resource estimates include:

- Gold price and valuation assumptions;
- Changes to the technical inputs used to estimate gold content (*e.g.*, bulk density estimation, and grade model methodology); and
- Geological interpretation (revision of vein models and the modeling of internal waste domains, *e.g.*, dikes and structural offsets such as faults and shear zones).

1.7.2 MINE

Existing mine rehabilitation can be unpredictable with respect to ground conditions and remediation work required. Conditions are expected to be good requiring predictable ground support but to minimize the impact of rehabilitation on mine production only a short potion of the ramp near the portal will be utilized in the early years of the mine. A new ramp and level development will access new mining areas and the old working areas will be accessed and rehabilitated as required later in the Project life, if the costs allow.

Qualified mining personnel capable of performing the different tasks are not readily available in the area. Highly qualified people will be hired in other parts of Canada and will be flown in on rotation and live in camp.

The groundwater table and inflows that reflect average conditions in the Feasibility study could be worse than expected leading to added pumping and sump requirements but the pumps and sumps have excess capacity and extra sumps can be added with minimal effect on Project economics.

1.7.3 PROCESSING AND TAILINGS

No recent testwork has been conducted and the reliance on historical data creates the risk that that data may not satisfactorily apply to the current resource. For example, operating records indicate that solution fouling was periodically experienced. If significant fouling is encountered, mill operation and gold recovery could be adversely affected.

1.7.4 PROJECT FINANCING AND ECONOMICS

As with all resource development projects, there is the inherent risk that the Project will not raise the necessary capital to fund any new construction.

This Project is exposed to gold price and shows its greatest sensitivity to gold pricing. Tight control on capital and operating spending will alleviate much of this sensitivity, but a prolonged low gold price environment could render the mine uneconomic.

1.7.5 ENVIRONMENT AND PERMITTING

The mine has all permits to operate an underground mine at the planned rate of 595 tonnes per day except for reactivation of the tailing management facility.

The inclusion Ragged TMF requires a listing on Schedule 2 of the Metal Mining Effluent Regulations (MMER).” The Company is in the process of submitting an Assessment of Alternatives report to determine if the Ragged TMA is in fact the best option for deposition of new tailings. The assessment will consider all possible alternatives for safe, long term tailings storage from environmental, socio-economic, and technical perspectives. Should the approval of the Schedule 2 listing be delayed the mine can deposit tailings in the old workings for a number of years.

1.7.5.1 Public Consultation

Consultation with local communities and First Nations groups has been ongoing and will continue as the project progresses. There is at present no significant opposition to the project and therefore the risk of social concerns impacting mine development and operation is very low.

1.8 RECOMMENDATIONS

Based on this Feasibility Study, the PL Mine Project should proceed to the pre-production phase (subject to financing) with recommendations as follows:

1.8.1 MINING

- Mobile mining equipment sourcing will be critical to determining Project construction start up and should be an initial high priority.
- Prior to stope development an updated study for rock mechanics and in stope support should be completed.
- During early development further underground diamond drilling in stoping areas should be performed as included in the Feasibility Study plan.

1.8.2 METALLURGY AND PROCESSING

- During the first 2 months of the preproduction period, further metallurgical testwork should be performed including variability and potential fouling materials testing.

1.8.3 INFRASTRUCTURE

- Completion of the access road and powerline upgrades are critical to allow year round access and minimize power costs.
- The initial expansion to the camp (to house people in the first 6 months) is also the other critical component to initial Project execution.

2 INTRODUCTION

This Feasibility Study technical report (the “Report”) has been jointly prepared by AMPL Professionals Ltd. (“AMPL”) and CSA Global Canada Geosciences Ltd. (“CSA Global”) at the request of Mr. Gorden Glenn, Chairman, President, and CEO of Minnova Corp. (“Minnova” or the “Company” or the “Issuer”) and focuses on the Issuer’s Maverick Project (the “Project”) in Manitoba, particularly the PL Property and PL Gold Deposit.

Minnova’s registered office is at 365 Bay Street, Suite 400, Toronto, Ontario M5H 2V1. It is a public company listed on the TSX-V (MCI-TSXV) and is currently focused on exploration and development of the PL Gold Deposit on the PL Property, Maverick Project, Sherridon area, Manitoba.

CSA Global prepared the resource estimates components of the study. A-Z Mining Professionals Ltd. prepared all other aspects of the study based on technical and economic input by a team of consultants under the management of AMPL Professionals Limited (AMPL), including EHA Engineering Ltd. (collectively referred to as the Consultants). The environmental components of the study were prepared by BluMetric Environmental and Parks Environmental working directly under the supervision of Minnova Corporation.

Minnova and its Consultants have contemplated the re-opening of the former producing Puffy Lake gold mine (now called PL Project/Mine) utilizing available infrastructure, which has largely remained intact since its closure in 1989. Mineral exploration programs have been undertaken by Minnova and previous owners, including a 2017 core drilling program completed in June. A revised mineral inventory and mineral resource estimate has been completed and incorporated into this study.

2.1 TERMS OF REFERENCE

A-Z Mining Professionals Ltd. and CSA Global were commissioned by the Issuer to prepare a technical report on its 100% owned Maverick Project in Manitoba. The Report is specific to the standards dictated by National Instrument 43-101 (“NI 43-101”), companion policy NI43-101CP and Form 43-101F1 (Standards of Disclosure for Mineral Projects – June 24, 2011) and to CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014). The Report focuses on an update of the PL Deposit mineral resource estimate and a Feasibility Study of the PL Deposit and Mine. The Report is also intended to enable the Company and potential partners to reach informed decisions with respect to the Project.

Although copies of the licences, permits, and work contracts were reviewed, AMPL Professionals has not verified the legality of any underlying agreement(s) that may exist concerning the licences or other agreement(s) between third parties. AMPL reserves the right, but will not be obligated to revise this Report and conclusions, if additional information becomes known to AMPL subsequent to the date of this Report.

The Issuer reviewed draft copies of this Report for factual errors. Any changes made as a result of these reviews did not include alterations to the conclusions made. Therefore, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

Minnova has warranted that full disclosure of all material information in its possession or control at the time of writing has been made to AMPL and CSA Global, and that it is complete, accurate, true, and not misleading. The Company has also provided AMPL and CSA Global with an indemnity in relation to the information provided by it, since the Authors have relied on Minnova's information while preparing this Report. The Company has agreed that neither it nor its associates or affiliates will make any claim against AMPL or CSA Global to recover any loss or damage suffered as a result of AMPL's or CSA Global's reliance upon that information in the preparation of this Report. Minnova has also indemnified AMPL and CSA Global against any claim arising out of the assignment to prepare this Report, except where the claim arises out of any proven willful misconduct or negligence on the part of AMPL or CSA Global. This indemnity is also applied to any consequential extension of work through queries, questions, public hearings, or additional work required arising out of the engagement.

2.2 QUALIFIED PERSONS

This Report was prepared by the following Qualified Persons (QPs) (Table 2.1):

Table 2.1: Qualified Persons Report Responsibilities

Qualified Person	**Report Section Responsibility**
Ian D. Trinder, M.Sc., P.Geo. (ON, MB) Principal Geologist, CSA Global	Sections 4.1-4.3, 4.6, 5.1-5.3.4, 5.4, 6 - 10, 11.1 to 11.4, and in part Sections 1, 25 and 27
Leon McGarry, Ph.D., P.Geo. (BC) Senior Resource Geologist, CSA Global	Sections 11.5, 12, 14, and in part Sections 1, 25 and 26 and Property Inspection
AMPL Professionals	Sections 2, 3, 4.4, 4.5, 5.3.5, 13, 15 - 24, and in part Sections 1, 5, 25 – 27.

Mr. Ian Trinder, M.Sc., P.Geo., CSA Global Principal Geologist and Qualified Person (QP), and Mr. Leon McGarry B.Sc., P.Geo., CSA Global Senior Resource Geologist and Qualified Person (QP) are responsible for the preparation of certain sections of this Report. Mr. Trinder has a Master of Science degree in geology and is a registered Professional Geoscientist (P.Geo.) in good standing registered in the Provinces of Ontario and Manitoba Canada (APGO no. 0452, APEGM no. 22924). Mr. Trinder has over 30 years of experience in the mining industry with a background in international precious and base metals mineral exploration including Project evaluation and management. Mr. McGarry has a Bachelor of Science degree in geology and is a registered Professional Geoscientist (P.Geo.) in good standing registered in the Provinces of Ontario and Saskatchewan, Canada (APGO no. 2348, APEGS no. 34929). Mr. McGarry has over 10 years of experience in the mining industry including a background in international mineral exploration, mineral resource modelling and estimation, and Project management for gold and base metal deposits.

2.3 SITE INSPECTIONS

Mr. McGarry completed a two-day field visit at the PL Mine Project from May 29-30, 2017. Mr. Chris Buchanan, M. Sc., P. Geo., Minnova Senior Geologist, accompanied and guided the Author during the field visit, providing valuable insight into the history and current status of the PL Mine Project.

Mr. McGarry conducted a review of available data at Minnova's field office at the PL Mine site and an inspection of diamond drill hole collars, surface outcrops and workings at several areas of the Project area. Selected drill core from diamond drill holes was examined at Minnova's secure core logging and storage facility at the PL Mine site. Mr. McGarry and Mr. Trinder reviewed the Company's most recent work, geological compilation reports, and data as well as historical geological information.

Mr. Leon McGarry visited the Nokomis Property between January 20-21, 2014 to review the geology, collect verification samples, and confirm the location of drill collars.

The Authors consider Mr. McGarry's site visit current under Section 6.2 of NI 43-101.

Mr. B. LeBlanc, Mr. M. Buck, and Mr. C. Clarke, along with a number of other specialist experts who contributed to the study under the supervision of AMPL, visited the PL Mine Project site from March 26 to March 31, 2017.

2.4 OTHER SOURCES OF INFORMATION

This report has been prepared by AMPL and CSA Global based on review of publicly available geological reports, maps, assessment files, mining claim information and technical papers, and company letters and memoranda made available by the Issuer, as listed in Section 27 (References) of this Report. CSA Global has taken reasonable steps to verify the information provided where possible.

CSA Global also had discussions with the management and consultants of the Issuer, including Mr. Gorden Glenn, Minnova's Chairman, President and CEO and Mr. Chris Buchanan, M. Sc., P. Geo., a consultant of the Company.

2.5 UNITS AND CURRENCY

Units of measurement used in this report conform to the SI (metric) system. Gold and silver assay values are reported in grams per tonne ("g/t") unless ounces per ton ("oz/T") are specifically stated. Base metal assay values are stated in percentages ("%") or parts-per-million ("ppm"). All currency in this report is in Canadian dollars (C$) unless otherwise noted.

All currency in this report is in Canadian Dollars ($CAD) unless otherwise noted.

3 RELIANCE ON OTHER EXPERTS

The Consultants have relied upon the following reports in conducting this study:

- "Condition Assessment Report Maverick Project Puffy Lake, Manitoba," Aecom Canada Ltd., October 2011.

- Manitoba Hydro powerline rehabilitation cost estimate.

The Authors have relied upon the Issuer, its management, and legal counsel for information related to the property location and description, underlying contracts and agreements pertaining to the acquisition of the property, and their status (Section 4).

Particularly, the Authors have relied, and believe that they have a reasonable basis to rely upon Aikens, MacAulay & Thorvaldson LLP, with office noted below, for legal tenure documents with respect to mining lease ML065, as outlined in a letter dated October 29, 2010 and supplied by Minnova (Section 4).

Aikens, MacAulay & Thorvaldson LLP
30th Floor Commodity Exchange Tower
360 Main Street
Winnipeg MB R3C 4G1

The Authors have also relied upon the Manitoba Department of Growth, Enterprise and Trade, Mineral Resources Division, Unit 360, 1395 Ellice Avenue, Winnipeg, Manitoba R3G 3P2 or on their website https://web33.gov.mb.ca/mapgallery/mgm-md.html for publicly available tenure data effective as of March (Section 4).

The Authors have not independently verified ownership or mineral title beyond information that is publicly available or been provided by the Issuer. The Project description presented in this report is not intended to represent a legal, or any other opinion as to title.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

Minnova's 100% owned Maverick Project comprising the Puffy Lake ("PL") and Nokomis Properties is located in northwestern Manitoba at about 55° 02' North latitude, 100° 59' West longitude, approximately 63 km northeast of Flin Flon, Manitoba and approximately 12 km southeast of the community of Sherridon (Figure 4.1).

Figure 4.1 Maverick project location

4.2 PROJECT DESCRIPTION AND TITLE

As shown in Figure 4.2, the PL and Nokomis properties are non-contiguous. The PL Property comprises twenty-eight (28) staked mining claims totaling 5,114 ha and one mining lease totaling 378 ha. The

Nokomis Property comprises twenty-seven (27) staked mining claims totaling 566 ha. In total, the Properties encompass an area of 6,058 ha or approximately 61 km^2.

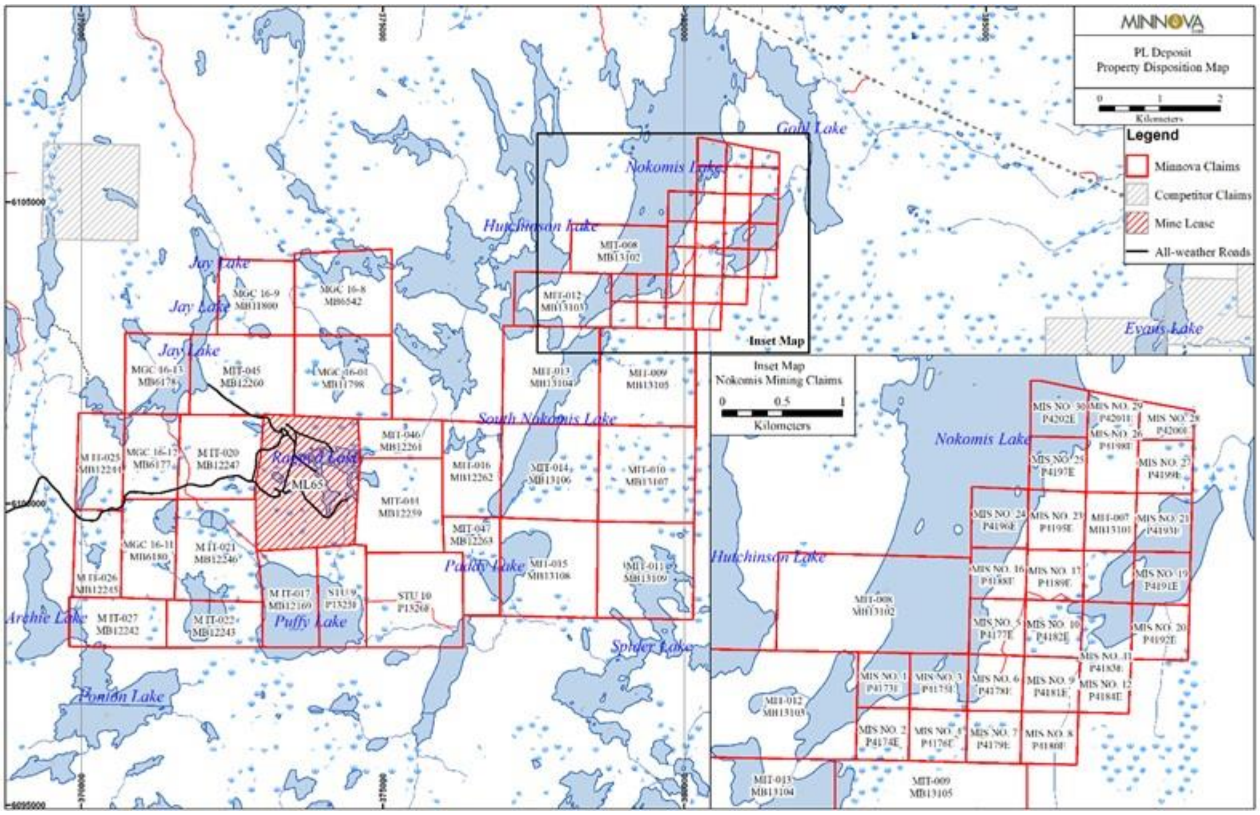


Figure 4.2 Property disposition map of the Maverick Property and PL Gold Project (Source: Minnova (2017)

On October 8, 2010, Minnova (then Auriga Gold) completed the acquisition of the past-producing Puffy Gold Mine ("PL Mine") and the PL Property and interests in the adjacent Nokomis Property from Pioneer Metals ULC ("Pioneer"). The purchased properties and other adjacent staked properties were renamed the Maverick Gold Project ("MGP"). Pioneer is a wholly-owned subsidiary of Barrick Gold Corporation ("Barrick"). Under the agreement, Minnova acquired 100% of Pioneer's interest in the PL Mine subject to a 3% net smelter royalty ("NSR") that reduces to 2.5% and 2%, if gold is below US$1,000/oz. and US$750/oz., respectively. The agreement also provided for the acquisition of Pioneer's 54% interest in the adjacent Nokomis Property. In consideration of the acquisitions, Minnova made total payments of $2.5 million; and issued stock to Pioneer valued at $1.0 million.

On November 22, 2011, Minnova (then Auriga Gold) completed the acquisition, the remaining 46% minority interest in the Nokomis Property from Claude Resources Inc. ("Claude"). Under the terms of the Agreement, Auriga agreed to issue to Claude 3,428,571 common shares of Auriga at an issue price of $0.35 per share. Transfer of 100% of the Nokomis Property to Minnova was then complete. The Nokomis Property is located less than 8 km northeast of the existing mine and mill infrastructure on the PL Property.

Minnova staked six (6) new PL mining claims in 2016 (MGC 16 series, Table 4.1) and an additional twenty-one (21) new PL mining claims in 2017 (MIT series, Table 4.1) in 2017. Transfer of title of the 2016 staked claims from the staker (Donnell Nice) to Minnova is pending. Minnova relinquished six (6) original property Puffy Lake mining claims in 2017 (STU 1, 3, 8, and 13 and UTS 1 and 2), four (4) of which were subsequently restaked (MIT-20, 21, 44 and 45). Minnova relinquished nine (9) original property Nokomis claims in 2015 (NOK 1-5, NOK 7-9, and MIS No. 22).

Table 4.1: Summary of PL and Nokomis Properties Claim Status

Name	Number	Registered Owner Name	Recorded Date	Expiry Date	Size (ha)	Claim Group	Assess 2018 ($)	Assess 2019 ($)
PL (Puffy Lake)								
MGC 16-01	MB11798	Donnell Nice[A]	9/6/2016	11/5/2018	224		2,800	2,800
MGC 16-8	MB6542	Donnell Nice[A]	9/6/2016	11/5/2018	224		2,800	2,800
MGC 16-9	MB11800	Donnell Nice[A]	9/6/2016	11/5/2018	188		2,350	2,350
MGC 16-11	MB6180	Donnell Nice[A]	9/6/2016	11/5/2018	160		2,000	2,000
MGC 16-12	MB6177	Donnell Nice[A]	9/6/2016	11/5/2018	140		1,750	1,750
MGC 16-13	MB6178	Donnell Nice[A]	9/6/2016	11/5/2018	130		1,625	1,625
MIT-008	MB13102	Minnova Corp.	11/23/2017	1/22/2020	127			1,588
MIT-009	MB13105	Minnova Corp.	11/23/2017	1/22/2020	248			3,100
MIT-010	MB13107	Minnova Corp.	11/23/2017	1/22/2020	256			3,200
MIT-011	MB13109	Minnova Corp.	11/23/2017	1/22/2020	256			3,200
MIT-012	MB13103	Minnova Corp.	11/23/2017	1/22/2020	135			1,688
MIT-013	MB13104	Minnova Corp.	11/23/2017	1/22/2020	256			3,200
MIT-014	MB13106	Minnova Corp.	11/23/2017	1/22/2020	256			3,200
MIT-015	MB13108	Minnova Corp.	11/23/2017	1/22/2020	256			3,200
MIT-016	MB12262	Minnova Corp.	11/23/2017	1/22/2020	157			1,963
MIT-017	MB12169	Minnova Corp.	5/25/2017	7/24/2019	160			2,000
MIT-020	MB12247	Minnova Corp.	5/25/2017	7/24/2019	190			2,375
MIT-021	MB12246	Minnova Corp.	5/25/2017	7/24/2019	212			2,650
MIT-022	MB12243	Minnova Corp.	5/25/2017	7/24/2019	128			1,600
MIT-025	MB12244	Minnova Corp.	5/25/2017	7/24/2019	128			1,600
MIT-026	MB12245	Minnova Corp.	5/25/2017	7/24/2019	120			1,500
MIT-027	MB12242	Minnova Corp.	5/25/2017	7/24/2019	128			1,600
MIT-044	MB12259	Minnova Corp.	8/23/2017	10/22/2019	217			2,713
MIT-045	MB12260	Minnova Corp.	9/22/2017	11/21/2019	220			2,750
MIT-046	MB12261	Minnova Corp.	11/23/2017	1/22/2020	89			1,113
MIT-047	MB12263	Minnova Corp.	11/23/2017	1/22/2020	117			1,463
STU 10	P1326F	Auriga Gold Corp.[B]	11/4/1992	1/3/2018	256	G12019	6,400	6,400
STU 9	P1325F	Auriga Gold Corp.[B]	11/4/1992	1/3/2018	136	G12019	3,400	3,400
				Sub-total:	**5,114**		**23,125**	**68,828**
Lease	ML65	Auriga Gold Corp.[B]	4/1/1992	4/1/2033	378		4536[C]	4536[C]

Table 4.1: Summary of PL and Nokomis Properties Claim Status								
Name	**Number**	**Registered Owner Name**	**Recorded Date**	**Expiry Date**	**Size (ha)**	**Claim Group**	**Assess 2018 ($)**	**Assess 2019 ($)**
				Total:	**10,606**		**27,661**	**142,192**
Nokomis								
MIS NO. 1	P4173E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 10	P4182E	Auriga Gold Corp.[B]	7/7/1972	9/5/2024	21	G12022	0[D]	0[D]
MIS NO. 11	P4183E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 12	P4184E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 16	P4188E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 17	P4189E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 18	P4190E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 19	P4191E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 2	P4174E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 20	P4192E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 21	P4193E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 23	P4195E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 24	P4196E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 25	P4197E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 26	P4198E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 27	P4199E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 28	P4200E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 29	P4201E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 3	P4175E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 30	P4202E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 4	P4176E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 5	P4177E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 6	P4178E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 7	P4179E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 8	P4180E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIS NO. 9	P4181E	Auriga Gold Corp.[B]	7/7/1972	9/5/2020	21	G12022	0[D]	0[D]
MIT-007	MB13101	Minnova Corp.	11/23/2017	1/22/2020	20			250
				Total:	**566**		**0**	**250**

Notes:
A. Title transfer from claim staker Donnell Nice to Minnova not yet filed with Manitoba Mining Recorder
B. Name change from Auriga to Minnova not yet filed with Manitoba Mining Recorder
C. Annual lease rental, not assessment work requirement
D. No assessment work required until 2020; $525/claim ($25/ha)

In a letter from the Manitoba Mines Branch to the law firm of Aikens, MacAulay & Thorvaldson LLP of Winnipeg, Manitoba dated October 29, 2010, the government confirmed the transfer of mining lease ML065 from Pioneer Metals Unlimited Liability Company ("Pioneer") to Minnova (then Auriga Gold). The transfer was registered in the Winnipeg Recording Office under Document No. 17465 on October 14,

2010. Mining lease ML065 was surveyed by Bastin & Shepherd, Manitoba Land Surveyors of Winnipeg, Manitoba between September 1, 1987 and September 24, 1987. Staked mining claims have not been surveyed and the surface rights remain with the Crown.

4.3 PROPERTY CLAIM STATUS

To the best of the Authors' knowledge, the Project mining claims and mining lease are in good standing until the expiry dates shown in Table 4.1. Minnova warrants that there are no current or pending challenges to ownership of the Project claims of which it is aware.

The Authors are unaware of any other encumbrances on the Project other than assessment work requirements, mining lease rental, and a royalty described in Section 4.3.1 to 4.3.3, below.

4.3.1 MINING CLAIM ASSESSMENT WORK REQUIREMENTS

Once a mining claim is recorded, it is in good standing for two years plus 60 days. If all work and reporting requirements are met, then a claim can be renewed annually for an indefinite period of time. Activities that constitute required work are described in Schedule B of Mineral Disposition and Mineral Lease Regulation, 1992, M.R. 64/92. Minimum expenditures for required work on a claim are:

1) $12.50/ha or part thereof for each of the second to the 10th years; and
2) $25/ha or part thereof for the 11th year and for each year thereafter.

Meeting the work requirements of all dispositions can be done on an annual basis, or through two additional means.

1) An over-accumulation of credits in one year can be applied against the required work of the next year, or for as many years as the amount accrued meets the minimum work required.

2) If the necessary work has not been completed, a cash payment equal to the work commitment of that year may be made to keep the disposition in good standing. If the work is completed and reported within five (5) years, you can apply to have your cash refunded. If the work is not completed within that time period, the cash payment is forfeited.

Grouping claims allows work completed on a claim to be applied to any one, or more, of the claims of the same group as long as all claims are in good standing.

Work requirements for the PL Property mining claims will total $23,125 in 2018 and rise to $68,828 in 2019 ($12.50/ha) when claims staked in 2017 reach their 2-year anniversary date (Table 4.1).

Accumulated work credits have the mining claims on the Nokomis Property in good standing until 2019 and 2020 at which time work requirements will total $ 250 and $13,900 respectively ($25/ha and $12.50/ha; Table 4.1).

4.3.2 Mining Lease Rental

Manitoba mining leases have 21-year renewable terms in the case of producing leases and are renewable under specified circumstances in the case of non-producing leases. Rental for a first term mineral lease or the renewal of a mineral lease in production is $10.50/ha or a fraction thereof per year but not less than $193. Rental for a first term mineral lease not in production is $12/ha or a fraction thereof per year but not less than $257. Rental for a renewal mineral lease when the lease is not in production is $12/ha or a fraction thereof per year but not less than $200. A detailed statement on exploration work carried out on the mineral lease must be filed on the 5th, 10th, and 21st anniversaries of the lease at a Manitoba Growth, Enterprise and Trade office no later than 60 days after the specified anniversary date.

ML65 on the PL Property was renewed for a second term in 2013; its annual rental cost, as a lease not in production, is $12/ha for a total of $4,536/year.

4.3.3 Royalties

The following royalties are listed on title:

- PL Property Net Smelter Royalty:
  - Under the terms of Minnova's acquisition agreement with Pioneer, the PL Property is subject to a net smelter royalty (NSR) in favour of Pioneer (now Barrick). The NSR varies with the price of gold and is 3.0% for a gold price greater than US$1,000/troy ounce, 2.5% for a gold price between US$750 and US$1,000, and 2.0% for a gold price under US$750.

- Nokomis Property Net Smelter Royalty
  - An underlying 2% NSR in favour of Hudson Bay Mining and Placer Dome (now Barrick Gold) exists on the Nokomis Property.

4.4 ENVIRONMENTAL LIABILITIES

The PL Mine is a past producing mine that operated under Manitoba Environment Act License 1207E from late 1987 through April 1989. Development and operations included a portal, ramp, over 7,000 m of underground workings, construction, and operation of a conventional processing plant with a nameplate capacity of 1,000 tpd. Over 350,000 tonnes of tailings were deposited into the Ragged TMF. As such, the PL Mine site area has already been environmentally impacted.

Proposed initial production from the underground operations only, at a rate of less than 600 tpd does not require any additional permits and is not considered a designated project under item 16(c) of the Schedule to the Regulations. As a result, the Company is not required to submit a Project Description under the Canadian Environmental Assessment Act (CEAA).

Minnova has provided a letter of credit in the amount of $75,000 to the Government of Manitoba under the terms of the Closure plan on 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 $5,000,000 for the performance of the Closure plan and the rehabilitation program.

The Company's provision for closure and reclamation costs are based on Minnova's estimates of costs to abandon and reclaim mineral properties and facilities as well as an estimate of the future timing of the costs to be incurred. The Company has estimated its total provision for closure and reclamation to be $2,900,324 at September 30, 2017, based on a total future liability of approximately $3,073,160, an inflation rate of 1.5% and a discount rate of 2.10%. Reclamation is expected to occur in approximately 10 years.

4.5 REQUIRED PERMITS

4.5.1 CROWN LAND PERMITS

Crown land permits including GP0004134 (Tailings Dam), GP0002799 (Road to Puffy), GP0003758 (Water Pipe ROW) and GP0004038 (Access Road) are renewed on an annual basis. To date, Minnova has maintained its annual renewals and the land permits remain in good standing.

4.5.2 WATER RIGHTS ACT LICENCE

A new Water Rights Act Licence was granted to Minnova in September 2017 for future operations. This allows Minnova to take water from local lakes and wells for future operations.

4.5.3 OTHER OPERATIONAL PERMITS

Numerous minor and routine permits will be obtained as required as the Project progresses to construction and operation (*e.g.*, Sand & Gravel; Septic Effluent, etc.).

4.5.4 MANITOBA ENVIRONMENT ACT LICENSE 1207E

The PL Mine’s historic Ragged Lake Tailings Management Facility (TMF), depicted in Figure 4.3, was approved as a tailing depositional and mine water treatment area and used as such under Manitoba Environment Act License 1207E from late 1987 through April 1989, before the federal Metal Mining Effluent regulations were developed.

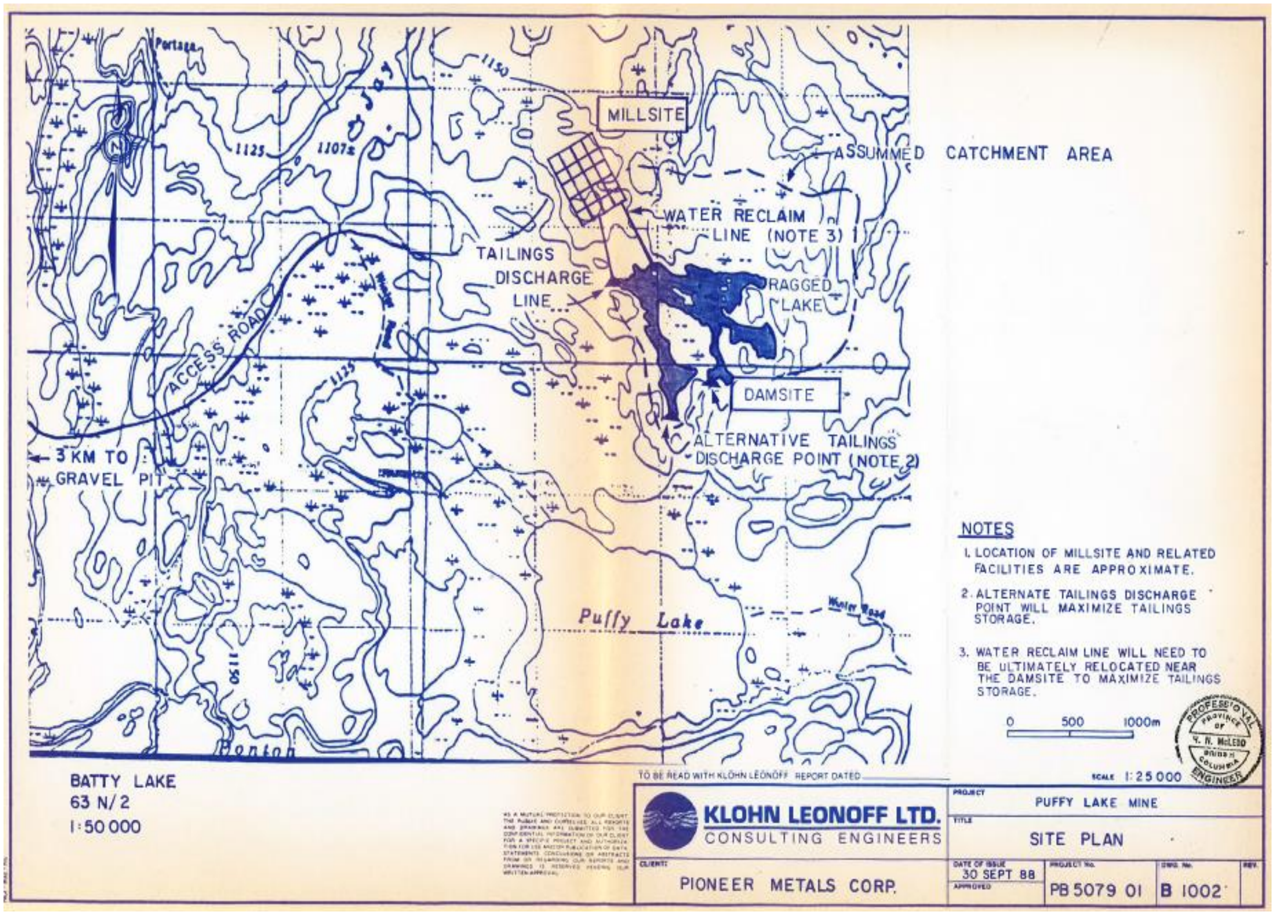


Figure 4.3 Location map of Ragged Lake TMA

On May 12, 2012, the Company notified Manitoba Conservation and Water Stewardship that it had acquired the assets of Pioneer Metals, namely the PL Gold Mine and its associated Environment Act License No. 1207E. The Provincial Ministry was also notified of the Company's intent to re-start operations and comply in all respects with the water quality limits, sampling, and reporting criteria set out in the federal Metal Mines Effluent Regulation ("MMER"). On May 17, 2012, the Ministry confirmed the Company's Environmental License was in good standing to re-start mining operations and noted Minnova's duty to comply with all criteria set out in the MMER.

Proposed initial production from underground operations only, at a rate of less than 600 tpd does not require any additional permits and is not considered a designated project under item 16(c) of the Schedule to the Regulations. As a result, the Company is not required to submit a Project Description under the Canadian Environmental Assessment Act (CEAA).

An application has been made to Fisheries and Oceans Canada ("DFO") to have Ragged Lake TMF added to Schedule 2 of the MMER. The draft application was submitted to DFO in May 2017 and returned in June 2017 for revisions. A revised application is in process and expected to be filed in January 2018. A decision is expected within 12 months of acceptance by DFO of the revised application.

4.5.4.1 Metal Mining Effluent Regulations (MMER)

The MMER came into force on December 6, 2002, under the *Fisheries Act*, and in 2012 applied to 117 mines across Canada. The MMER impose limits on releases of arsenic, copper, cyanide, lead, nickel, zinc, radium-226, and total suspended solids and prohibit the discharge of effluent that is acutely lethal to fish.

The MMER include provisions to allow the use of natural, fish-frequented water bodies for mine waste disposal because at some sites the disposal of mine waste in such water bodies may be the preferred disposal option for pollution prevention and reduction of long-term environmental risk. The use of a water body frequented by fish for mine waste disposal can only be authorized through an amendment to the MMER, which adds that water body to Schedule 2 of the Regulations.

In the case where a fish-frequented water body has been added to Schedule 2, Section 27.1 of the MMER requires the development and implementation of a fish habitat compensation plan that has to be approved by the Minister of the Environment before the deposit of a deleterious substance into a tailings impoundment area. The objective of this requirement is to help ensure that offsets are provided for losses of fish habitat that occur as a result of the use of natural, fish-frequented water bodies for mine waste disposal. The fish habitat compensation plan must be approved before deposit of mine waste into the water body can begin. The MMER also require the mining company to submit an irrevocable letter of credit to ensure that adequate funding is available to implement all elements of the fish habitat compensation plan.

Any effluent discharged from a mine waste disposal area established in a water body listed in Schedule 2 (a tailings impoundment area as set out in the Regulations) must meet the effluent discharge limits specified in Schedule 4 of the MMER to help ensure the protection of downstream ecosystems. In addition, the MMER require that environmental effects monitoring must be conducted downstream from the effluent discharge point to determine if there are any effects on fish, fish habitat, or the use of fisheries resources.

An application has been made to the DFO to have Ragged Lake, the historical tailings impoundment area of the Puffy Lake Mine, added to Schedule 2 of the Metal Mining Effluent Regulations. The application was submitted to DFO in May 2017. A decision is expected within 12 months.

4.6 OTHER SIGNIFICANT FACTORS AND RISKS

Changes in environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues could potentially materially affect access, title, or the right or ability to perform the work recommended in this report on the Project. However, at the time of this report, the Authors are unaware of any such potential issues affecting the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Project can be accessed via an all-weather gravel road that extends from Provincial Highway 10 south of Flin Flon northwards 78 km to the community of Sherridon (Figure 5.1). At approximately the 66 km marker, the PL Mine access road is taken east 9 km to the mine site. The mine-access road was constructed in 1986 and crosses the Keewatin Railway Company line (owned by a consortium of three First Nation communities) at “Mile 33” (“Kilometre 52.8”) from Sherritt Junction. The rail line passes approximately 6 km west of the PL Property. In addition, charter floatplane and helicopter services are available at the Flin Flon airport, which is serviced by commercial airlines from Winnipeg.

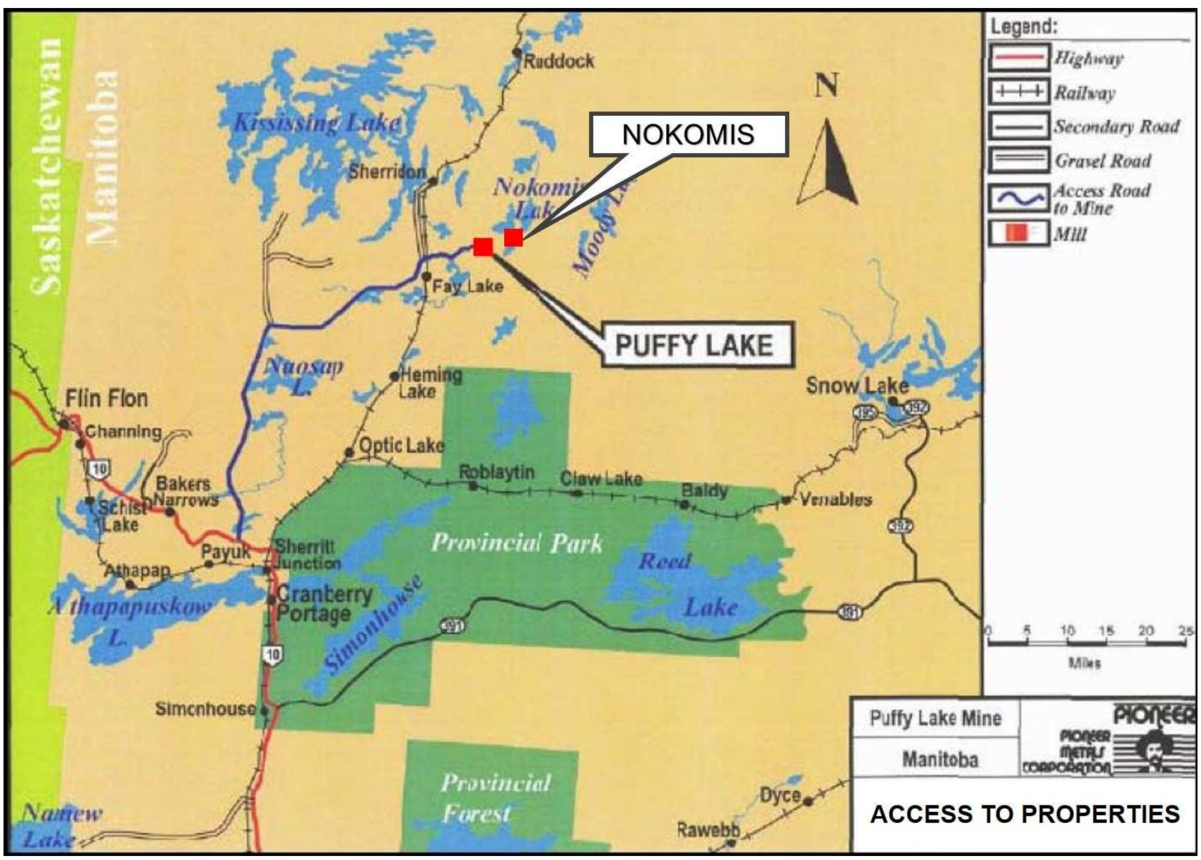


Figure 5.1 Regional map showing site access
(Source: Minnova, 2017)

5.2 CLIMATE

The Maverick Project, located in the mid-north area of Manitoba, has a continental subarctic climate characterized by short, warm summers and long cold winters. The closest Environment Canada weather station to the Project is just outside Flin Flon, 65 km to the southwest. Climate information for the Flin Flon area is presented graphically in Figure 5.2 and is based on a 30-year average of data collected between 1961 and 1990. It is unlikely that the local climate at the Project is significantly different.

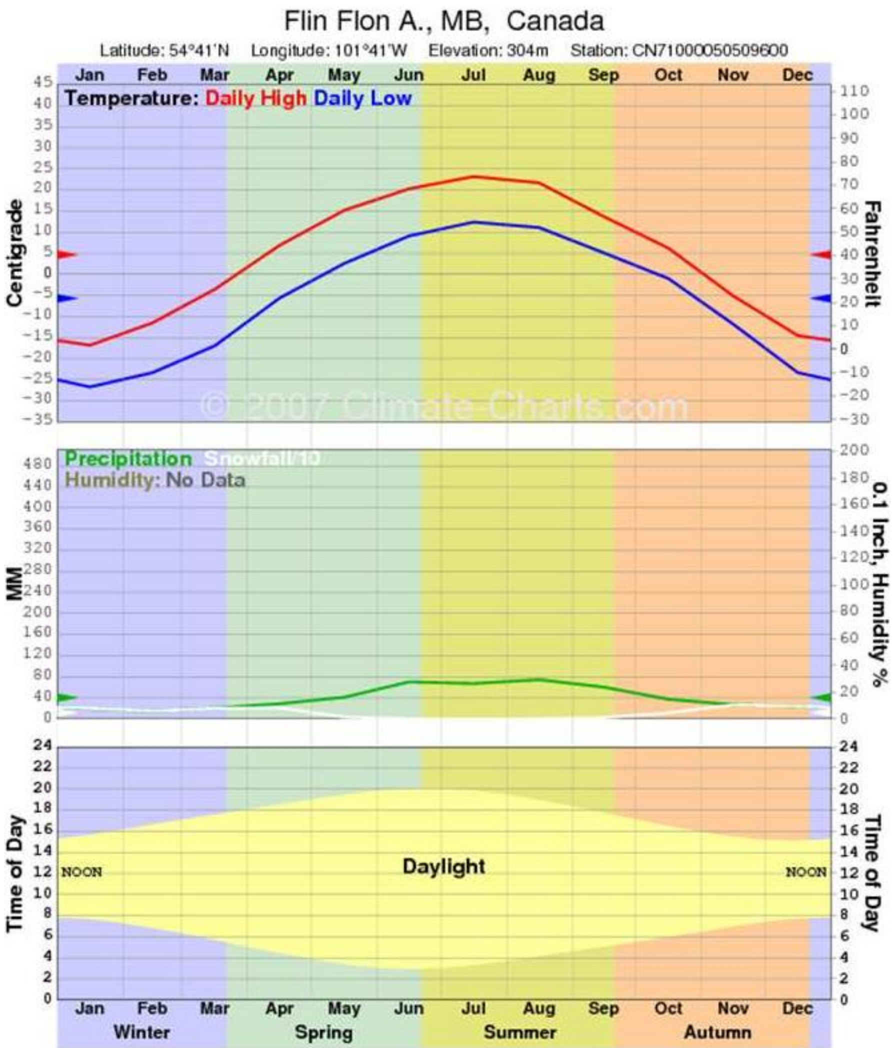


Figure 5.2 Climate chart for Flin Flon, MB
(Source: www.climate-charts.com)

Temperatures show a wide range from daily average minimums in January below -26°C to daily average maximums in July of 23.4°C. The yearly mean daily temperature is -0.3°C.

Total annual precipitation amounts to 476 mm, which is about 10% less than the 525 mm Winnipeg receives. Snow accounts for approximately 30% of total precipitation. More than half of the precipitation

(55%) falls in the four summer months of June, July, August, and September. The driest month is February with barely 4% of the precipitation.

Seasonal specific mineral exploration activities may be conducted year-round at the Project. Geological mapping and field sampling programs are best completed between May and October. Ground geophysical and diamond drilling programs are often preferred between the months of November and March during winter freezing conditions, which allows for improved access for heavy machinery such as diamond drill rigs into wet lowland areas of the Property. Mine operations in the region may be conducted year-round with supporting infrastructure.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The all-weather gravel road, which joins the community of Sherridon to Provincial Highway 10, lies approximately 9 km to the west of the Project. Keewatin Railway Company rail line access to Flin Flon is from the Jungle Lake siding, 14 km northwest of the Project, via Sherridon and Cranberry Portage. The nearest Manitoba Hydro power line is situated at Sherridon, which is on a branch of the transmission line that runs from the Sandy Bay hydro-power station on the Churchill River in Saskatchewan to Snow Lake, Manitoba. This line is connected to Manitoba Hydro's provincial transmission and distribution grid.

5.3.1 SHERRIDON

Sherridon is the closest community to the Project (about 20 km by road). Sherridon is connected by an all-weather road to Provincial Highway 10 (between Flin Flon and Cranberry Portage). In 2016, the population of Sherridon was approximately 108 people. Present sources of employment include fishing, trapping, tourism, and forestry.

5.3.2 FLIN FLON

Flin Flon, Manitoba is located approximately 115 road km and 110 rail km south-southwest of the Project, and 110 km by road from Sherridon. Flin Flon had a population of approximately 4,982 in 2016 and is the site of the large mining and metallurgical centre owned by Hudbay Minerals (formerly HBED), producing copper, zinc, and gold. Numerous mine supply vendors and service companies are located in Flin Flon.

5.3.3 WINNIPEG

Winnipeg, the capital city of Manitoba, sits 750 km south of Flin Flon. It is one of Canada's largest cities functioning as a major transportation, manufacturing, business services, and financial centre. Regular transport routes are well-established between Winnipeg and Flin Flon serving the industrial sector in support of numerous other regional mines in the Flin Flon/Snow Lake/Thompson corridor.

5.3.4 MANPOWER

During previous operations at the PL Mine, there was no appreciable problem in attracting suitable operating personnel. Although a workforce for future operations is not now readily available in the immediate area, it is assumed that a majority of the required personnel could be recruited from the nearby mining communities of Snow Lake and Flin Flon, where mining activity is in decline. As far as we know, there are currently no mining operations in either Lynn Lake or Leaf Rapids. Alamos is looking at developing the old MacLellan Mine as an open pit (purchased Carlisle Gold, 2.5 m ounces).

5.3.5 ON-SITE EXISTING INFRASTRUCTURE

Between May 1986 and April 1989, considerable mining and processing activity took place at the PL Property, including general site development (Figure 5.3) and underground excavation in the form of ramps, drifts, raises, and stopes. Much of this infrastructure remains intact on a care and maintenance basis. The decline ramp and underground workings are currently flooded with water. As can best be determined (Pioneer monthly and other reports), some 350,000 tonnes of ore was mined from the various stopes.

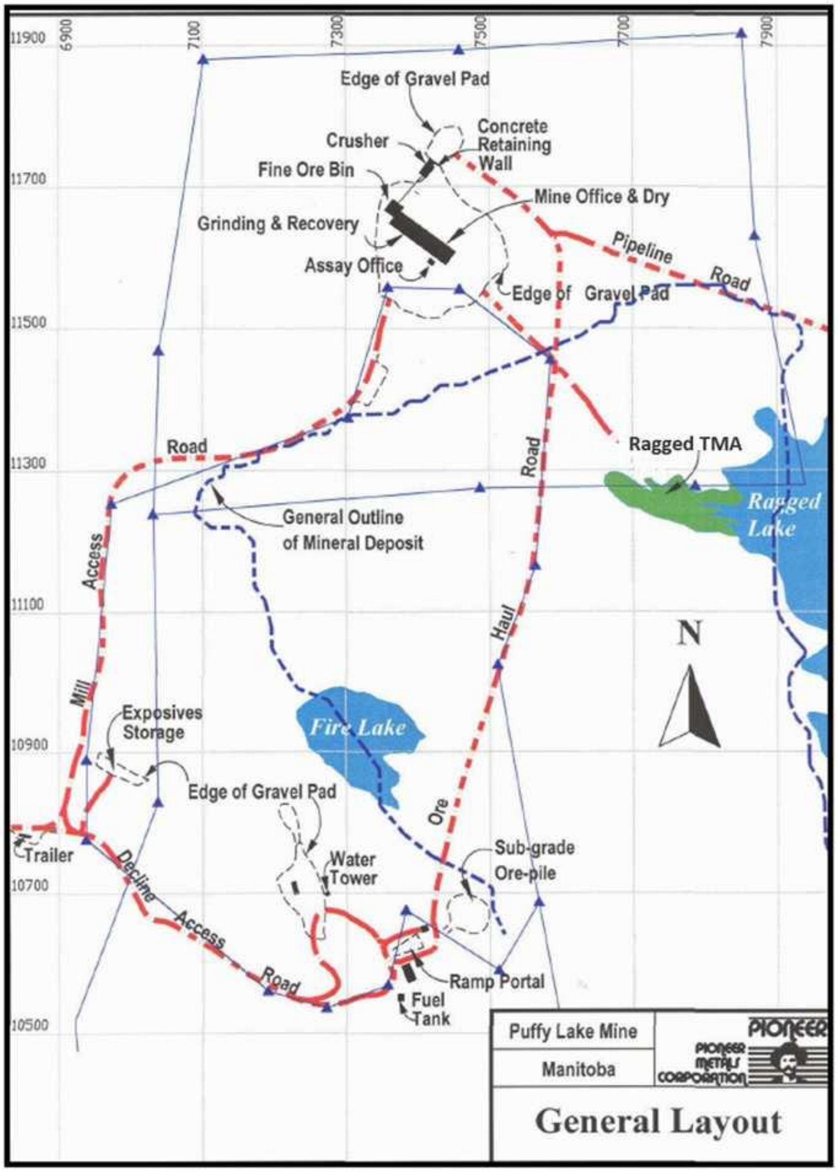


Figure 5.3 General layout of PL mine site (c. 1989)
(Source: Minnova, 2017)

As of the Effective Date, it appears that Minnova's concessions are of sufficient area for proposed future mining operations (including existing processing plant site, tailings management area, and waste disposal areas).

5.3.5.1 Mine

The past producing mine infrastructure includes:

- Surface portal with steel cover;

- Ramp to approximately the 135 m below surface elevation;
- Level and raise development;
- Concrete pads in proximity to the portal; and
- Propane tank pad.

A new core logging building is also located in proximity to the portal.

5.3.5.2 Processing Plant

The processing plant, crushing complex, fine-ore bin, grinding complex, and concentrator/recovery plant site is compact and well-planned. The forest fire of 1989 did not affect the concentrator and associated structures.

The processing plant building remains in good structural condition with minor repairs to cladding and the roof required. The steel structure and concrete foundations remain in very good condition.

A majority of the equipment is in good condition and can be recommissioned. Some equipment has been cannibalized or destroyed by vandals and requires replacement. Some equipment has been removed, primarily in the refinery area, and will be replaced with equivalent or better technology. The equipment's main power feed electrical wiring has been removed from the processing plant building. The motor control centre has been vandalized and will require upgrading and replacement. Section 17 should be referred to for a complete discussion of the process flow diagram and equipment for the PL gold plant.

5.3.5.3 Support Facilities

The office complex and mine dry (clean clothes lockers room, dirty clothes room, showers, and toilets) have been substantially stripped and will require repair and refurbishment. Walls and ceilings have been damaged to different degrees and require refurbishing.

All maintenance tools and equipment were removed from the shop. The warehouse consists solely of the building shell. The assay laboratory building is in reasonable condition, with all useable laboratory equipment removed.

In the mill, shop, and laboratory, some lighting and electrical power is serviceable but will need to be properly tested and checked. Much of the wiring will need to be replaced due to vandalism. The office complex area requires all electrical related services to be re-established. The heating and ventilation equipment is in place and appears undamaged though some ducting requires re-establishing. All the furnaces will need to be properly checked and serviced before being put back into operation.

5.3.5.4 Power Distribution

A 23-km power transmission line was constructed from Sherridon to the mine site. This line follows the provincial access road south from Sherridon to a point close to the take-off point for the mine access road. From there, the power transmission corridor heads east to the mine site parallel to the mine access road and terminates at the main substation near the processing plant. This section of the line was destroyed by the forest fire in 1989 and has not been fully reconstructed. Between January 2012 and June 2012, Minnova invested approximately $550,000 in initial power line upgrades that were undertaken by Manitoba Hydro. Most of the pole line has been replaced to the near vicinity of the mine site but no cabling has been replaced, as yet. The task remains incomplete and has been included in the capital cost estimate. A step transformer will be required at the take off point from the main line.

The main site substation near the processing plant remains in place, requiring minor rehabilitation. The crushing plant substation has been vandalized to a point that it requires replacement. The old underground substation has been removed as has the power line to the portal area; both will need to be replaced. A new transformer will also be required to service the pumps and the Camp facilities at Jay Lake.

5.3.5.5 Water Supply

Adequate supplies of fresh (potable) water are available from several lakes in the immediate vicinity of the mining lease. A new Water Rights Act Licence was granted to Minnova in September 2017 for future operations. In addition, process water will be recycled from the Ragged Tailings Management Facility (TMF).

5.3.5.6 Tailing Management Facility

An approved tailings management area at Ragged Lake exists. Approximately 350,000 tonnes of tailing material was discharged to this area when the mine operated in the 1980s.

An application has been submitted to the federal Department of Fisheries and Oceans to have Ragged Lake added to Schedule 2 of the Metal Mines Effluent Regulation, re-classifying it as a tailings disposal area. A new control structure will be required at the discharge point from the TMF to raise the operating level in the basin to 348 m. Several other structures will be required as well at low points around the perimeter of the basin. No engineering has yet been undertaken on these structures, though costs have been estimated and included in the capital cost estimates.

5.3.5.7 Waste Rock Disposal Area

During previous operations, waste was utilized to construct roads and access ramps. By and large underground access development at the PL Deposit was in near-ore grade material and very little, if any, waste rock was generated.

More than adequate waste rock disposal area is available south of Fire Lake for future waste rock disposal.

5.3.5.8 Camp Accommodation

An exploration camp is located in proximity to the mill and can house and cater to sixteen (16) individuals. There is also a central toilet area with sinks and showering facility and a separate common room with television. There are an additional set of trailers located in Flin Flon, which can house another sixteen (16) individuals and could be added to the existing camp area. Minnova owns all of these camp facilities.

5.3.5.9 Communications

Presently, the Project has voice communication via a satellite telephone and there is no data or internet services to the property.

5.4 PHYSIOGRAPHY

The Project is situated on terrain with limited relief typical of the Canadian Shield. Higher ridges expose extensive areas of rock outcrop and are interspersed with lakes and low-lying swampy areas. Elevations range from 350 m above sea level at the mill site to 340 m at PL.

The PL plant site is located on a finger of high ground (with an average slope of 5%) surrounded on three sides by swamps.

Rocky ridges trend north, following major fault and shear zones. The area is part of a typical boreal forest region in north-central Manitoba. Swamps and lakes occupy a significant part of the surface area. Glacial overburden varies in thickness. Black spruce and tamarack are most common in the lowlands whereas poplar, white spruce, and jack pine dominate uplands. Considerable portions of the western part of the PL Property were burned in the 1989 forest fire.

6 HISTORY

6.1 PL PROPERTY HISTORY

6.1.1 HISTORICAL OWNERSHIP OF THE PL PROPERTY

Table 6.1 summarizes a series of agreements that resulted in Pioneer's 100% interest in the PL ("Puffy Lake") Property and eventual acquisition by Minnova in 2010.

Table 6.1: Summary of Historical Ownership of the PL Property

Year	Ownership
1953	• Mr. A. Oleson staked claims NIP 1-4 in February 1953 over an area that presently includes the PL Mine site.
1960	• Mr. A. Oleson assigns all interests in the claims to Hudson Bay Exploration and Development (HBED) in March 1960. The original NIP claims were allowed to lapse in 1966.
1979	• Mr. J.J. Studer staked Bed Claim CB 10186 over the original Puffy Lake Prospect in October 1979. A 100% interest in CB 10186 was then transferred to Granges for due consideration.
1980-1984	• On June 4, 1980, Maverick entered into an option agreement with Granges to earn an undivided 60% interest in CB 10186, by expending a total of \$350,000 on exploration and development work by December 31, 1983. • Subsequent program expenditures were to be 60% by Maverick and 40% by Granges. The agreement also provided for an area of interest around CB 10186; 3 miles to the north, west, and east of the claim block. • Between August 1980 and September 1981, an additional 20 claim blocks, contiguous to the original Bed Claim CB 10186, were staked and became part of the Puffy Lake Joint Venture. • On June 29, 1984, an Exploration Development and Mining Agreement (ED & M) was made between Maverick and Granges to define the relationship of the parties. On this date, Granges transferred a 60% interest to Maverick in the 21 claim blocks subject to a 20% non-assessable NPI payable to Granges. Maverick was appointed the operator.
1985	• On July 26, 1985, Homestake Mineral Development Company (Homestake) signed an option agreement with Maverick to acquire a 60% interest in the Puffy Lake claims. At that date, Maverick had earned a 71.8% ownership interest in the property and the right to acquire a 100% interest by expending an additional \$300,000 on exploration and development in 1985, under the terms of the 1984 ED & M agreement with Granges. • On December 16, 1985, Granges transferred to Maverick 100% interest in the Puffy Lake Property, subject to a 20% NPI, as defined under the original agreement dated June 4, 1980. • In 1985, a further 11 claim blocks were staked, contiguous to the previous group.

Table 6.1: Summary of Historical Ownership of the PL Property	
Year	**Ownership**
1986	• On January 30, 1986, Homestake terminated the option agreement and Maverick was free to enter into other agreements as follows. • Pioneer entered into an agreement with 301639 B.C. Ltd., a British Columbia corporation, whereby a 10% interest in the Puffy Lake Property was acquired by the expenditure of $300,000 on road building. Pioneer contributed $165,000 toward these expenditures by way of the purchase of the beneficial right to the property interest. • Maverick entered into an agreement with Societe en Comandite Miniere S & S No. 1, a Quebec Limited Partnership, whereby it agreed to sell a 7% working interest in the Puffy Lake Project in consideration of the expenditure of $2 million on the exploration of the property during 1986. • Pioneer entered into an agreement with a British Columbia Limited Partnership, which agreed to spend up to $2 million on further exploration of the Puffy Lake Project in part consideration of earning a 1% working interest in the property. Pioneer agreed to advance up to $1.1 million toward these expenditures. • In September 1986, Maverick and Pioneer Metals Corporation (Pioneer) merged under the latter name.
1987	• In 1987, a third group of 19 claim blocks were staked and added to the property.
2010	• Transfer of PL (“Puffy Lake”) Property to Minnova (then Auriga Gold) is completed.

6.1.2 Historical Exploration of the PL Property

A brief summary of the exploration history of the PL Property is provided in Table 6.2.

Table 6.2: Summary of Historical Exploration on the PL Property	
Year	**Exploration Activities**
1953	• Mr. A. Oleson staked claims NIP 1-4 in February 1953 over an area that presently includes the PL Mine site, approximately 8 km southeast of Sherridon, Manitoba.
1960	• Mr. A. Oleson transfers all interests in the claims to Hudson Bay Exploration and Development (HBED) in March 1960. From January to April 1960, HBED conducted horizontal loop EM surveys on these claims and delineated an approximately 120 m long conductor. HBED drilled 14 diamond drill holes to test the conductor.
1966	• The NIP claims lapsed in 1966.
1972	• An airborne EM and magnetometer survey performed during 1972 by Sherritt Gordon Mines Ltd. included ground in the vicinity of the original Puffy Lake occurrence.
1979-1981	• J.J. Studer staked CB 10186 over the area in October 1979. • In December 1980, 100% interest in the claims was transferred to Granges Exploration Aktiebolag (Granges). • In June 1980, Maverick Mountain Resources Limited (Maverick) entered into an option agreement with Granges. Early in 1981, Maverick completed a 30 hole drill program.

Table 6.2: Summary of Historical Exploration on the PL Property	
Year	**Exploration Activities**
1984-1985	• During the winter of 1984-1985, a 29 hole drill program was performed. The strike length of the main zone was increased to 1,037 m and a second zone of mineralization was discovered 275 m north of and parallel to the main zone. • In 1985, 29 additional diamond drill holes were completed. This confirmed the down dip plunge extension of the Main Zone gold mineralization to a vertical depth of 183 m. Four separate and distinct parallel gold-bearing lenses were identified – Sherridon, Upper Main, Main and Lower Main lenses, uppermost to lowest respectively.
1986	• During 1986, a total of 75 diamond drill holes totalling 9,145 m were completed. A $2 million underground development program was awarded to Canadian Mine Development that included a decline ramp to the 100 and 200 levels, drifting, trial stoping, and bulk sampling (Northern Miner, April 7. 1986). • Underground development began in May 1986. A decline was collared and a crosscut was driven from the footwall to intersect the Main Zone. When the zone was encountered, the decline was turned and driven along strike and slightly down dip. This decline was driven to prove the continuity of the zone. The zone proved to be continuous and the decline was then spiralled down to intersect the zone at a lower elevation. Development totalled 650 m of decline. • In September 1986, Maverick and Pioneer Metals Corporation (Pioneer) merged under the latter name.
1987-1988	• In January 1987, Pioneer announced plans to place the PL (“Puffy Lake”) Deposit into production at a rate of 500 tpd at an estimated capital cost of $18 million (Northern Miner, January 26, 1987). • January 1987, Kilborn Engineering (B.C.) Ltd – “Pioneer Metals Corporation, Puffy Lake Project, Feasibility Study.” The study concluded that the reported resources within four parallel mineralized zones could be mined using modified open stope methods (Northern Miner, January 26, 1987). A 93% rate of recovery was expected at an operating cost of US$175 per ounce (Pioneer news release, January 19, 1987). Reserves were calculated using an average 2 m mining width, 3.43 g/t cut-off grade, and a 10% dilution factor. • Mill site excavation began May 15, 1987 and pre-production underground development commenced near the month end. A raise was driven from the lower portion of the decline up dip to prove continuity in that direction and a test stope was mined. Additional diamond drilling from surface was undertaken. • Completed 225 diamond drill holes. • The claim CB 10186 was converted to a mining lease; October 30, 1987. • Milling began December 4, 1987 and the first doré bar was poured December 15, 1987 (Northern Miner, March 7, 1988).
1989	• In March 1989, Pioneer announced suspension of mining operations due to the failure to reach commercial production levels (Globe and Mail, March 21, 1989). • In the period August 8 to 11, 1989, a series of forest fires swept through the Sherridon – Puffy Lake area causing considerable damage to pipelines from the fresh water intake lake as well as discharge lines to the tailing pond. The crusher/mill/office complex had an adequate firebreak surrounding it and escaped damage from these fires.

Table 6.2: Summary of Historical Exploration on the PL Property	
Year	**Exploration Activities**
1994	• June 1993. Kilborn Engineering Pacific Ltd – "Puffy Lake Mine, Feasibility Study – 1993"
1999-2010	• During February and March 1994, in anticipation of reactivating the PL ("Puffy Lake") mine, a series of eight (8) holes for a total of 633 m was drilled. The program was designed to better define reserves in the area to be mined – as infill drilling at 15 m centres, over an area previously defined at 30 m centres. The drilling confirmed the continuity of all four known zones and returned values similar to those received from past drilling in the area. (News Release dated March 17, 1994).

6.2 PREVIOUS DRILLING

Five gold-bearing zones have been delineated by exploration drilling in the PL Property area. These include, from the top downwards, the Sherridon, Upper, Main, Lower, and Lower 2 Zones.

A total of four hundred eighty-one (481) core holes were drilled by previous operators within the PL Property area (including targets peripheral to the PL Deposit itself) of which three hundred nineteen (319) collars were surveyed at the surface. The coordinates of the remaining one hundred sixty-two (162) holes were located by chain and compass. The actual PL Deposit was explored (1960-1994 inclusive) by approximately four hundred ten (410) drill holes, as shown in **Error! Reference source not ound.**. The PL Property was dormant from 1994 until 2010, when it was acquired by Minnova (then Auriga Gold).

Table 6.3: Summary of Historical Surface Exploration Drilling – PL Deposit			
Year	**Company**	**Number of Holes**	**Total Metres**
1960	Hudson Bay E&D	14	Uncertain
1981	Maverick Mountain Resources	30	Uncertain
1984	Maverick Mountain Resources	29	Uncertain
1985	Maverick Mountain Resources	29	Uncertain
1986	Maverick Mountain Resources	75	9,145
1987	Pioneer Metals Corp	225	Uncertain
1994	Pioneer Metals Corp	8	633
	Total	**410**	

Most of the holes were drilled vertically with the exception of seventy-two (72) holes in areas of difficult or impossible access caused by swamps or lakes.

In September 1980, diamond drilling began at the PL Property along cross sections spaced 30 m apart and aligned N 70° E. Nominal drill spacing along the section lines was 30 m, but varied considerably during various drill campaigns between 1980 and 1987. There are areas of very dense drilling on the west side of an interpreted major fault. On the other hand, the east side of the fault is sparsely drilled with only twenty-nine (29) holes used to explore this area.

During the winter of 1984-1985, a twenty-nine (29) hole drill program was completed. The strike length of the main zone was increased to 1,037 m and a second zone of mineralization was discovered 275 m north of and parallel to the main zone. In 1985, twenty-nine (29) additional diamond drill holes were completed. This confirmed the down dip plunge extension of the Main Zone gold mineralization to a vertical depth of 183 m.

During 1986, a total of seventy-five (75) diamond drill holes totalling 9,145 m were completed in advance of a $2 million underground development program that was awarded to Canadian Mine Development and included a decline ramp to the 100 and 200 levels, drifting, trial stoping, and bulk sampling (Northern Miner, April 7. 1986).

For the purposes of their 1993 Feasibility study, Kilborn (1993) used a diamond drill database consisting of the following numbers of drill holes.

1. Prior to the 1986 Feasibility Study: 192 holes
2. Subsequent to the 1986 Feasibility Study: 281 holes

The underground development and stoping demonstrated continuity in the various zones. No underground drilling is reported on the PL Deposit.

Subsequent to the Kilborn 1993 Feasibility study, and in anticipation of reactivating the PL ("Puffy Lake") Mine, a series of eight (8) holes for a total of 633 m was drilled in February and March 1994. The program was designed to better define reserves in the area to be mined with infill drilling at 15 m centres, over an area previously defined at 30 m centres. The drilling confirmed the continuity of all five known zones and returned grades similar to those received from past drilling in the area. (Pioneer News Release dated March 17, 1994) (3).

6.3 NOKOMIS PROPERTY HISTORY

6.3.1 HISTORICAL OWNERSHIP OF THE NOKOMIS PROPERTY

Table 6.4 summarizes a series of agreements that resulted in Minnova's 100% interest in the Nokomis Property.

Table 6.4: Summary of Historical Ownership of the Nokomis Property	
Year	**Ownership**
1972	• In 1972, the MIS group of 30 claims was staked for Dome Exploration (Canada) Limited to cover the known areas of gold mineralization at Nokomis Lake. Subsequently, Hudson Bay Exploration and Development Ltd. ("HBED") acquired an interest in the claims.
1996	• In September 1996, Pioneer entered into an option agreement with Placer Dome Canada Ltd. ("Placer") and HBED to acquire a 100% interest in the property subject to a 2% NSR. The initial work agreement provided for an infill drill program of no less than 8 holes totalling no less than 300 m to be completed by August 1, 1997 and additional exploration expenditures totalling $500,000 by August 1, 2001. The August 1, 2001 date was subsequently extended.
2004	• In 2004, Pioneer signed an agreement with Claude Resources Inc. ("Claude") on the Nokomis Property, including the MIS claims and eight (8) newly acquired NOK claims. Under the terms of the agreement, Pioneer granted to Claude the sole and exclusive right to earn a 50% working interest in the property for consideration that included a cash payment of $35,000 and a firm exploration expenditure commitment of at least $215,000 on or before August 1, 2004. Claude received the exclusive right to manage and operate the exploration programs during the earn-in period. Upon receipt of the consideration, Pioneer would transfer a 50% right, title and interest in the Property to Claude. Thereafter, Claude and Pioneer would proceed as joint venture partners on the property with Claude serving as the initial operator.
2005	• On January 20, 2005, Pioneer announced that the terms of the 1996 option agreement between Pioneer, Placer, and HBED had been fulfilled due to Claude's exploration program carried out in 2004. Claude now earned a 50% interest in the Property.
2010	• In May 2010, Minnova (then Auriga Gold) completed the acquisition of Pioneer's interest in the PL ("Puffy Lake") Property (see Table 6.1) and the Nokomis Property, which at that time, was 54%. Under the terms of the agreement, the transfer is subject to a 3% net smelter royalty that reduces to 2.5% and 2%, if gold is below US$1,000/oz. and US$750/oz., respectively. The remaining 46% interest in the Nokomis Property was held by Claude Resources Inc.
2011	• In November 2011, Minnova (then Auriga Gold) signed an agreement to purchase the 46% minority interest in the Nokomis Property held by Claude. Under the terms of the Agreement, Auriga agreed to issue to Claude 3,428,571 common shares of Auriga at an issue price of $0.35 per share. Transfer of 100% of the Nokomis Property to Minnova is complete.

6.3.2 Historical Exploration of the Nokomis Property

This Section (Section 6.3.2) is extracted from E. Buhlmann's 2006 NI 43-101 report "Technical Report on the Nokomis Gold Property, Sherridon, Northwestern Manitoba, Canada Prepared for Pioneer Metals Corporation" and dated September 8, 2006.

The main sources of information for the following summary of exploration data is (i) Mineral Deposit Series Report No. 19 by Ostry & Trembath (1992) published by Manitoba Energy and Mines Geological Services, (ii) assessment files in the non-confidential section of the Assessment Files library, (iii) Mineral

Inventory Cards (Minerals Division, Manitoba Industry, Economic Development and Mines, Winnipeg) and (iv) information from other sources which are referenced individually.

A brief summary of the exploration history of the Nokomis Property is given in Table 6.5.

Table 6.5: Summary of Historical Exploration on the Nokomis Property

Year	Exploration Activities
1947-1949	• Between 1947 and 1949, Mr. D.S. Robertson mapped the area around Nokomis Lake for the Geological Survey of Canada ("GSC") and classified the rocks along the eastern shore of Nokomis Lake as Nokomis gneiss, the oldest gneisses within the Kisseynew Complex.
1950	• In 1950, Mr. D.S. Robertson of the GSC suggested the Nokomis area as prospective to Mr. S.C. Simpson, who worked for Sherritt Gordon Mines Ltd. Mr. Simpson went to the area and undertook float sampling, with results over 40 oz. Ag/tonne. Subsequent prospecting identified arsenopyrite zones in gneiss, which occasionally carried visible gold and sometimes up to an ounce of gold per tonne.
1958	• In 1958, Mr. A.L. Parres of Flin Flon optioned the property from Mr. S.C. Simpson. Between May 15 and June 1, 1958, Parres drilled five X-ray holes on the Main West Zone for a total of 95.2 m. Holes intersected encouraging values (Parres, 1958; Hall, 1960).
1960-1961	• In June 1960, G.S.W. Bruce of Rio Tinto Canadian Exploration Ltd examined the property in company with Mr. S.C. Simpson. Arsenopyrite mineralization was sampled and assay results ranged from trace to 0.35 oz. Au/ton. • In August 1960, Rio Tinto started a geological mapping and trenching program on the Nokomis Property (Hall, 1960; AF 90659). In August and September 1960, Mr. H.I. Hall and two prospectors located, lengthened, and sampled old trenches to check previous surface samples while focusing on arsenopyrite zones. Prospecting activity included examining rock specimens on all outcrops and panning the surface soil. Mapping of outcrops on the host gneiss was carried out (AF 90659). • In 1961, Rio Tinto completed grids and 41 drill holes for a total of 2,806 m mainly on the South Showing (Nokomis Deposit; referred to as the upper and the lower mineralized zones) and North Showing areas. Results indicated gold values over a strike length of 180 m and to a depth of 45 m with additional values of interest below 120 m in the South Showing area (Klemenchuk, 1961; AF 90658).
1972-1975	• In 1973, Placer cut a grid and completed a helicopter-borne 4-channel EM and magnetic survey of 947 line km on the claims (MacCormack, 1985). Early in the year 1975, ground EM and magnetometer surveys were completed, followed by diamond drilling of 14 holes for a total of 1,483 m (MacCormack, 1985; Charteris, 1984).
1984-1985	• In 1984, a new metric grid was cut at 50 m line spacing over the entire 30 claims. A ground magnetic survey was completed, and a detailed geological mapping program was undertaken (MacCormack, 1985; Charteris, 1984). Lithogeochemical surveying included 23 drill core samples and 111 samples of host horizon material (Charteris, 1984). • In 1985, from mid-August to late September, 25 drill holes were completed for a total of 2,997 m (MacCormack, 1985).
1991-1999	• The Nokomis Lake area and surrounding areas were mapped as part of the NATMAP

Table 6.5: Summary of Historical Exploration on the Nokomis Property	
Year	**Exploration Activities**
	Shield Margin Project, a multi-disciplinary team effort by the Geological Survey of Canada, Manitoba Geological Surveys Branch, and Saskatchewan Geological Survey. New geological maps and regional structural interpretations resulted (Zwanzig, 1999).
1996-1997	• In 1996, the property was optioned to Pioneer. In early 1997, Pioneer completed 11 drill holes for a total of 1,199 m. Two SPECTREM conductors were tested, nine fill-in holes on the upper mineralized zone, and one hole on the lower mineralized zone in the South Showing area were completed (Buhlmann, 1997; AF 73230).
1999-2000	• In October-November 1999, Pioneer completed drill hole collar surveys of several 1985 holes and the 1997 holes. Old Placer drill cores in the Manitoba Government drill core library were sampled and several outcrops were examined and sampled in the North Showing and South Showing areas (Buhlmann, 2000). • In March 2000, Pioneer deepened two holes, DH-85 and DH-63, and completed 9 additional drill holes for a total of 1,319 m (Buhlmann, 2000).
2004-2006	• 20 Claude/Pioneer diamond drill holes were completed in 2004 (04-106 to 04-108) and 2005 (05-110 to 05-125).

6.3.3 NOKOMIS HISTORICAL DRILLING

Drilling by previous operators at the Nokomis Property delineated a principal gold-bearing horizon. The mineralized horizon is intersected by a normal fault orientated parallel to the strike of the horizon and, downthrows the horizon to the east. The shallow portion of the bisected horizon is termed the 'Upper Host Zone'. The downthrown portion is termed the 'Lower Host Zone'.

A total of one hundred twenty-five (125) diamond drill holes were drilled by previous operators within the Nokomis Property, as shown in Table 6.6. The property was dormant from 1994 until 2010 when it was acquired by Minnova Gold.

Table 6.6: Historical Drilling at Nokomis from 1958 to 2005			
Year	**Company**	**Number of Holes**	**Total Metres**
1958	L. Parres	5	Uncertain
1961	Rio Tinto	41	2,806
1975	Placer Dome	14	1,483
1985	Placer Dome	25	2,997
1997	Pioneer Metals Corp	11	1,199
2000	Pioneer Metals Corp	9	1,319
2004-2005	Claude/Pioneer Metals Corp	20	1,812
	Total	**125**	

The Host Zones have a shallow dip of between 30 degrees and 40 degrees to the east. Approximately half of the historical holes were drilled sub vertically with the remainder dipping between 70 degrees and 45 degrees to the west. Drill holes range between 15 m to 500 m. Hole depths increase to the east in order to intersect the downthrown Lower Host Zone.

6.4 PL DEPOSIT HISTORICAL RESOURCE/RESERVE ESTIMATES

6.4.1 HISTORICAL RESOURCE ESTIMATES - 1981 TO 1987 – MAVERICK-PIONEER - PRE-PRODUCTION

Prior to implementing a production decision in 1988, several phases of resource estimates had been completed by Pioneer/Maverick. Although the PL ("Puffy Lake") Deposit was known and tested by surface drill holes prior to 1980, none of the data obtained prior to 1980 was available; thus, the mineral inventory estimate was based on data obtained from 1980 to 1987. The various pre-production resource estimates are summarized as follows:

- Maverick, 1981 based on an additional 30 ddh: 428,000 t at 7.68 g/t Au
- Pioneer 1985 based on additional 58 ddh: 603,000 t at 6.86 g/t Au
- Pioneer May 31, 1986 and based on additional 154 ddh: 696,700 t at 7.99 g/t Au
- Pioneer (B. Simmons) April 1, 1987: 2,604,500 t at 7.54 g/t Au
- Piteau April 1, 1987. Review of Pioneer estimate: 1,192,000 t at 6.23 g/t Au

No technical documentation of the 1981, 1985, and 1986 resource estimates are available to the Authors; as such, no comment can be made regarding the key assumptions, parameters, and methods used to prepare the historical estimates. The relevance and reliability of the historical estimates are, therefore, unknown to the Authors. Without the technical documentation, the Authors are unable to confirm whether these apparently uncategorized resources meet the current criteria for resource categories set out in CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014) and National Instrument 43-101 (June 24, 2011).

The 1987 Pioneer (Brian Simmons) estimation procedure (reviewed by Piteau Associates Engineering Ltd in April 1987) is reported to have been as follows:

- Geological logging of diamond drill core identified potential gold bearing quartz horizons and selected sample intervals for gold assays.
- The drill holes and gold assays were plotted on cross sections drawn perpendicular to the strike of the quartz veins. Correlations of the quartz-gold intersections, with the four principal horizons (Sherridon, Upper, Main, and Lower), were made on these cross sections.
- All the drill hole and assay data were entered into the computer program together with the corresponding horizon identifier labels (Sherridon, Upper, Main, and Lower). The computer program is specifically designed to handle vertical drill holes; consequently, the small number of inclined drill holes were handled by inserting the coordinates of the vein intersections, as if a vertical drill hole was intersecting the vein at that point.
- The computer program calculated the area of influence of each drill hole on each quartz-gold horizon as a polygon. The computer program approximated true polygon construction by creating a block model and then assigning the drill hole grade outwards in ring increments, with early numbered holes having priority.

- The memory capacity of the computer limits the size of the block model. Therefore, it was necessary to divide the property into three parts: the Main Zone to the south, the Fire Zone to the north, and the Ragged Lake Zone to the northeast.
- The computer calculated the tonnage of each polygon using a specific gravity of 2.77 and calculated a total tonnage and weighted average grade for each of the gold bearing horizons in each of the two halves of the deposit.
- Two determinations of the specific gravity of the PL Deposit were available. Both values were from the Main Horizon in the Main Zone. The first determination, by Britton Research in 1981 on a 20 kg sample from diamond drill core, gave a value of 2.87 gm/cc. The second determination, by Lakefield Research on a 6.8 tonne bulk sample from surface trenches, gave a value of 2.77 gm/cc. A value of 2.77 was selected for the mineral inventory estimation because it was based on a larger sample and was also the more conservative value.
- The reader is cautioned that a Qualified Person has not done sufficient work to classify any of the 1981 to 1987 historical resource estimates as current mineral resources. The Authors, CSA Global and the Issuer, are not treating the historical estimates as current Mineral Resources, and as such, they should not be relied upon.
- No work will be completed to upgrade or verify the 1981 to 1987 historical estimates as current mineral resources since they are superseded by the current 2017 Mineral Resource estimate presented in Section 14.

6.4.2 Post Production Historical Resource Estimates - 1989 – Pioneer

At the time of closing and subsequent to the mine being in production for approximately fifteen (15) months Pioneer reported a resource based on a cut-off of 3.5 g/t as follows:

- Indicated Resource: 1,346,177 t at 7.86 g/t
- Inferred Resource: 883,718 t at 6.35 g/t

No technical documentation of the 1989 resource estimate is available to the Authors; as such, no comment can be made regarding the key assumptions, parameters, and methods used to prepare the historical estimates. The relevance and reliability of the historical estimates are, therefore, unknown to the Authors. Without the technical documentation, the Authors are unable to confirm whether these resources meet the current criteria for resource categories set out in CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014) and National Instrument 43-101 (June 24, 2011).

The reader is cautioned that a Qualified Person has not done sufficient work to classify the 1989 historical resource estimate as current mineral resources. The Authors, CSA Global, and the Issuer are not treating the historical estimate as current Mineral Resources, and as such, they should not be relied upon.

No work will be completed to upgrade or verify the 1989 historical estimate as current mineral resources since they are superseded by the current 2017 Mineral Resource estimate presented in Section 14.

6.4.3 Historical Resource Estimates - 1993 – Kilborn

Kilborn (1993) completed a feasibility study on the PL ("Puffy Lake") Deposit, which included an updated resource estimate utilizing a resource cut-off of 3.5 g/Au (Pieterse, 2010). The resource estimated by Kilborn was:

"Proven and Probable" Resource

- Resource Quantity: 1,346,200 t
- Average Gold Grade: 7.86 g/t
- Contained Gold: 340,200 oz. troy

"Possible" Resource

- Resource Quantity: 883,700 tonnes
- Average Gold Grade: 7.11 g/t
- Contained Gold: 202,100 oz. troy

Kilborn utilized digital drill hole data provided by Pioneer for the resource estimate. The data included a total of four hundred seventy-three (473) core holes drilled in the Puffy Lake area; surface coordinates of three hundred eleven (311) collars were surveyed and the remaining one hundred sixty-two (162) holes were located by chain and compass. Most of the holes were drilled vertically with the exception of seventy-two (72) holes in areas with difficult or impossible access, caused by swamps or lakes. Drill hole spacing was irregular and varied considerably because of various drilling campaigns between 1980 and 1989. There were areas of very dense drilling on the west side of a major fault; the east side of the fault, however, was sparsely drilled with only twenty-nine (29) holes.

Kilborn considered five zones in their study of Puffy Lake area, including from the top downwards, the Sherridon, Upper, Main, and Lower Zones. Kilborn reported that the southern part of the Lower Zones split into two distinct zones separated by a layer of rock of a minimum thickness of 10 mm. All five seams were modelled and their resources estimated. In addition, Kilborn reports that there is an indication of the presence of additional zones based on isolated drill holes, but these lack continuity; therefore, they were not included in the resource estimate.

Kilborn uploaded the Pioneer digital files including drill hole assay and survey data and digitized outlines of the mined-out areas and openings into the Datamine™ software, which was used for manipulation of the data and estimation of the resource. Separate geological models were created for the area west of the main fault and the area east of the main fault. The hanging and footwalls of the Sheridan, Upper, Main, Lower, and Lower Two Zones were modeled by generating surfaces through the top and bottom of the drill hole intersections of the zones. The volumes contained between the hanging and foot walls were filled with Datamine™ cells and subcells. The maximum horizontal dimensions were 25 m by 25 m, the minimum dimensions were one-eighth of the 25 m in both X and Y directions. The vertical dimension of the cells were controlled by the difference in elevation of the hanging wall and the foot wall of the zone. The proposed mining method required a minimum true thickness of 1.0 m, which is equivalent to a vertical thickness of 1.2 m. The vertical thickness in some areas of the original geological models was less than required and was increased to the minimum thickness of 1.2 m."

The interval length of the individual samples within the mineralized zones varied from 0.1-2.0 m. These samples were composited over 1.0 m lengths with a minimum composite length of 0.7 m due to a minimum mining thickness of 1.2 m. High grade assays were not cut.

Experimental semi-variograms were computed for the composites within the mineralized zones. The differential semi-variograms were difficult to define because of the erratic nature of the deposit and showed very high nugget/sill ratios. The Kriging method was, therefore, rejected for grade interpolation and the more traditional method of inverse power of distance weighting method was selected.

The gold grades were interpolated into the cells and subcells using the inverse power of distance weighting method with a preferred direction of ellipsoid. In addition to distance weighting, the length of sample interval was used to weight gold grades in cells and subcells. The search radius was selected from the semi-variograms built for each zone.

Kilborn's 1993 Proven, Probable, and Possible resource categories do not meet the current criteria for resource categories set out in CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014) and National Instrument 43-101 (June 24, 2011).

The reader is cautioned that a Qualified Person has not done sufficient work to classify the 1993 historical resource estimates as current mineral resources. The Authors, CSA Global, and the Issuer are not treating the historical estimates as current Mineral Resources, and as such, they should not be relied upon.

No work will be completed to upgrade or verify the 1993 Kilborn historical estimate as current mineral resources since it is superseded by the current 2017 Puffy Lake Mineral Resource estimate presented in Section 14, which includes the results of the recent 2017 Minnova diamond drilling program.

6.4.4 HISTORICAL RESERVE ESTIMATE – 1993 - TONTO

Utilizing Kilborn's 1993 mineral resource estimate, Tonto Mining, a Division of Dynatec International Ltd ("Tonto"), estimated mineable proven and probable ore reserves for the 1993 feasibility study:

- Ore Reserve Quantity: 855,000 t
- Average Gold Grade: 6.7 g/t
- Contained gold: 184,200 oz. troy

These reserves were estimated based on:

- An in-place cut-off of 3.5 g/t.
- A mining dilution above the design mine height of 15%.
- A loss in pillars and remnants of 20%.

Apart from reserves "on the west side of the fault," no other reserves are referred to.

No detailed technical documentation of the 1993 reserve estimate is available to the Authors; as such, no comment can be made regarding the key assumptions, parameters, and methods used to prepare the historical estimates. The relevance and reliability of the historical estimates are, therefore, unknown to the Authors. Without the technical documentation, the Authors are unable to confirm whether these reserves meet the current criteria for reserve categories set out in CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014) and National Instrument 43-101 (June 24, 2011).

The reader is cautioned that a Qualified Person has not done sufficient work to classify the 1993 historical reserve estimate as current Mineral Reserves. The Authors, CSA Global, and the Issuer are not treating the historical estimate as current Mineral Resources or Reserves, and as such, they should not be relied upon.

No work will be completed to upgrade or verify the 1993 historical estimate as current Mineral Resources or Reserves since they are superseded by the current 2017 Mineral Resource and Reserve estimates presented in Section 14 and Section 15.

6.5 NOKOMIS HISTORICAL RESOURCE ESTIMATE

A historical 'mineral inventory estimate' for the Upper and Lower Host Zones at Nokomis is presented in a 1985 unpublished Dome Exploration report entitled "Project 52, Nokomis Lake, Manitoba: Mineral Inventory Estimates" prepared by L.V. MacCormick. Howe has not been able to review the estimation methodology employed in the 1985 study. The estimate is discussed in the September 8, 2006 E. Buhlmann technical report, where it is stated that the historical estimate does not conform to Ni 43-101 and should not be relied upon (Table 6.7).

Table 6.7: Nokomis Historical Mineral Inventory Estimate (MacCormack, 1985 in Buhlmann, 2006)

Domain	Tonnes	Au (g/t)	Comments on Location
Upper Domain	109,681	8.19	Above Pegmatite
Lower Domain	239,429	5.15	Below Pegmatite
Upper and Lower Domain	349,110	6.10	Combined Above and Below Pegmatite

No technical documentation of the 1985 mineral inventory estimate is available to the Authors; as such no comment can be made regarding the key assumptions, parameters, and methods used to prepare the historical estimates. The relevance and reliability of the historical estimates are, therefore, unknown to the Authors. Without the technical documentation, the Authors are unable to confirm whether these resources meet the current criteria for resource categories set out in CIM Definitions and Standards on Mineral Resources and Mineral Reserves (May 10, 2014) and National Instrument 43-101 (June 24, 2011).

The reader is cautioned that a Qualified Person has not done sufficient work to classify the 1985 historical mineral inventory estimate as current mineral resources. The Authors, CSA Global, and the

Issuer are not treating the historical estimate as current Mineral Resources, and as such, they should not be relied upon.

No work will be completed to upgrade or verify the 1985 historical mineral inventory estimate as current mineral resources since they are superseded by the current 2014 Nokomis Mineral Resource estimate presented in Section 14.

6.6 SUPERSEDED NI 43-101 RESOURCES

PL Deposit mineral resources were estimated in accordance with CIM Standards on Mineral Resources and Reserves by P&E in 2011 and updated in 2014. The 2011 and 2014 mineral resource estimates ("MRE") are presented below for historical reference only. These MREs are superseded by CSA Global's current 2017 PL Deposit MRE, as stated in Section 14.

6.6.1 2011 PL Mineral Resource Estimate (P&E)

On August 4, 2011, Auriga Gold announced its initial Mineral Resource estimate for the Maverick Gold Project All MRE work was carried out by FH Brown, CPG, and Antoine Yassa, P.Geo., both independent Qualified Persons, as defined by NI 43-101. The MRE was carried out using the commercially available Gemcom GEMS™ and Snowden Supervisor™ software programs. The 2011 MRE database contained 84,638 m of drilling from 496 drill holes, including 455 historical holes and 41 holes completed by Auriga, as well as 22,550 gold assay values.

Six mineralization domain models were defined by continuous mineralized structures and assay intervals equal to or greater than 0.50 g/t Au. Interpretation polyline vertices were snapped directly to drill hole assay intervals, in order to generate a three-dimensional representation of the extent of the PL Deposit mineralization. Below the 290 m level, the 0.50 g/t Au mineralization domains were then clipped where possible to exclude material less than 2.50 g/t. Occasional lower grade assay intervals were included to maintain geological continuity. Domain wireframes were then clipped above a constructed overburden surface. The resulting domains were used as hard boundaries during estimation, and for rock coding, statistical analysis and compositing limits.

In order to ensure equal sample support, a compositing length of 1 m was selected. A capping threshold of 30 g/t Au resulted in the capping of 18 composite samples prior to the estimation.

Anisotropic inverse distance cubed ("ID3") linear weighting of capped composite values was used for the estimation of block grades in two passes. During the first pass, 3-12 composites from two or more drill holes, within a search ellipsoid of 40.0 m × 40.0 m × 10.0 m, were required for estimation. All blocks estimated during the first pass were classified as Indicated. During the second pass, 3-12 composites from one or more drill holes were required for the estimation. The search ellipse was expanded to ensure that all remaining blocks within the defined mineralization domains were estimated. All blocks estimated during the second pass were classified as Inferred. A specific gravity of 2.72 t/m^3 was selected for tonnage estimates.

Mineral Resources were classified in accordance with guidelines established by the Canadian Institute of Mining, Metallurgy and Petroleum. In order to ensure that the reported mineral resources have "reasonable prospects for economic extraction," a conceptual floating-cone pit shell was developed. Mineral Resources within the pit shell were accumulated against a cut-off grade of 0.60 g/t Au. Additional resources below the pit-shell limits were accumulated using an underground cut-off of 2.50 g/t Au (Table 6.8).

Table 6.8: Superseded 2011 PL Mineral Resource Estimate

	Resource Category	Tonnes	Gold (g/t)	Gold (oz)
In-Pit Au 0.6 g/t cut-off	Indicated	242,000	4.16	32,000
	Inferred	78,000	3.81	10,000
Underground Au 2.5 g/t cut-off	Indicated	702,000	6.29	142,000
	Inferred	3,018,000	5.65	548,000

6.6.2 2014 PL Mineral Resource Estimate (P&E)

On April 15, 2014, an updated MRE for the PL Deposit was prepared that incorporated an additional 67 drill holes completed during 2012 and 2013. The assay table of the database contained 24,772 Au assays, of which 2,187 Au assays were analyzed since the 2011 MRE.

The 2014 MRE update was undertaken by Yungang Wu, P.Geo. and Eugene Puritch, P.Eng. of P&E Mining Consultants Inc., both independent Qualified Persons in terms of NI 43-101.The MRE was carried out using Gemcom GEMS™ and Snowden Supervisor™ software programs.

In 2014, the ID3 weighting of capped composite values were used to estimate block grades in four passes. During the first pass, 5-12 composites from two or more drill holes, within a search ellipsoid of 20 m × 25.0 m × 5.0 m, were required for estimation. All blocks estimated during the first pass were classified as Measured. During the second pass, 3-12 composites from two or more drill holes were required for the estimation of Indicated Resources within an expanded search ellipse. Subsequent runs up to a maximum range of 120 m × 160 m × 40 m were used to estimate Inferred Resources. A specific gravity of 2.81 g/cm^3 was selected for tonnage estimates based on additional density determinations obtained since the 2011 MRE.

In 2014, a void solid representing underground workings was used to code all historical mined blocks with a rock code 999 that were at least 1 % or greater inside the Void. These blocks were depleted from the resource model.

Mineral resources within the pit shell were accumulated against a cut-off grade of 0.60 g/t Au. Additional resources below the pit-shell limits were accumulated using an underground cut-off of 2.50 g/t Au (Table 6.9).

Table 6.9 Superseded 2014 Mineral Resource Estimate Statement					
	Class	Au Cut-off (g/t)	Tonnes	Au (g/t)	Contained Au (oz.)
In-Pit	Measured	0.6	123,000	4.41	17,400
	Indicated	0.6	445,000	4.40	63,000
	M+I	0.6	568,000	4.40	80,400
	Inferred	0.6	45,000	4.87	7,000
Out-of-Pit	Measured	2.5	27,000	5.12	4,500
	Indicated	2.5	1,057,000	5.95	202,300
	M+I	2.5	1,084,000	5.93	206,800
	Inferred	2.5	2,135,000	6.01	412,500
Total	**Measured**	**0.6+2.5**	**150,000**	**4.54**	**21,900**
	Indicated	**0.6+2.5**	**1,502,000**	**5.49**	**265,300**
	M+I	**0.6+2.5**	**1,652,000**	**5.41**	**287,200**
	Inferred	**0.6+2.5**	**2,180,000**	**5.99**	**419,500**

6.7 SUPERCEDED PEA AND FEASIBILITY STUDIES

Previous PEA and Feasibility studies completed on the Maverick Project since the closure of the PL Mine in 1989 have included:

- 1993 Kilborn Feasibility for Pioneer – studied underground mining of the PL Deposit (Kilborn, 1993)
- 2011 ACA Howe PEA for Auriga – studied open pit mining of the PL Deposit (ACA Howe, 2011)
- 2012 ACA Howe PEA for Auriga – studied underground and open pit mining of the PL Deposit (Orava, et al., 2012)
- 2014 ACA Howe PEA for Auriga – studied underground and open pit mining of the PL and Nokomis Deposits (Orava, et al., 2014)

These studies are now superceded by the Feasibility Study presented in this Report.

6.8 HISTORIC PRODUCTION

Construction of a process plant with a capacity of 1,000 tpd, was completed at the PL Deposit in 1987 and production had begun by January of 1988. Annual gold production was targeted at 72,000 troy ounces. Actual annual production in 1988 and 1989 was 21,000 and 8,500 troy ounces, respectively. The concentrating plant treated a total of 350,000 tonnes of feed. At peak, the PL ("Puffy Lake") Mine employed one hundred seventy (170) people. The first major employee layoff occurred in March of 1989, and the mine was completely closed by June 1989.

![A-Z Mining Professionals logo]

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The following description of the PL Deposit area geology is summarized from Zwanzig and Bailes (2010).

The PL Deposit is located at the boundary of the Kisseynew and the Flin Flon domains of the Paleoproterozoic (Precambrian) Trans-Hudson Orogen (Figure 7.1). Litho-tectonic units in the area have been metamorphosed to middle and upper amphibolite facies. The metamorphic grade increases northward in the region.

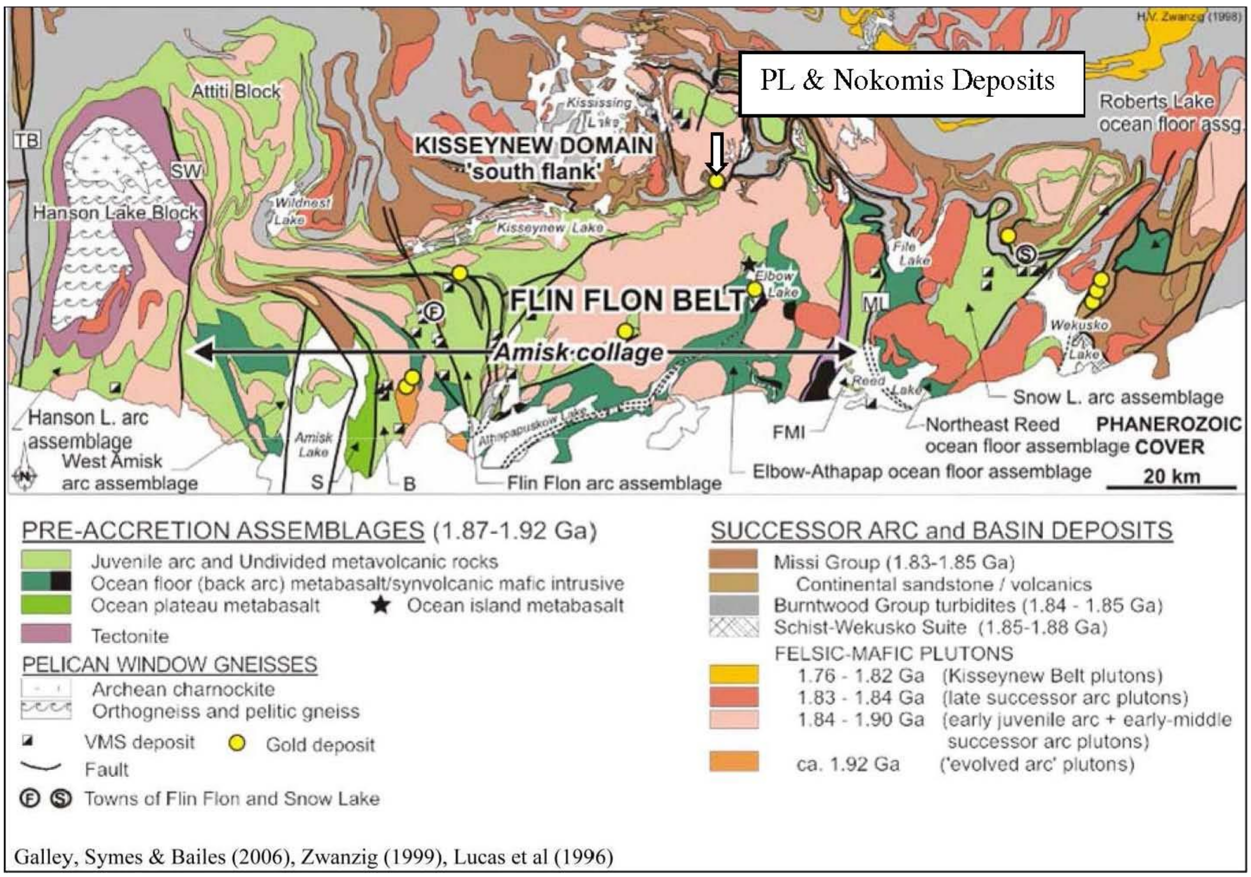


Figure 7.1 Regional geology

The Flin Flon Domain forms a generally east trending 230 km by 80 km belt of complexly folded and metamorphosed volcanic, sedimentary, and intrusive rocks. The belt is bound to the north by the Kisseynew Domain, to the south by Paleozoic rocks, to the east by Archean rocks of the Superior Province, and to the west by the Tabbernor Fault and granitic rocks of the Glennie Domain.

The area around the PL and Nokomis Deposits is underlain by a sequence consisting supracrustal rocks of the Amisk (Flin Flon arc assemblage), Burntwood, and Missi groups and granitoid gneisses of the Sherridon-Hutchinson Lake Complex. The rocks have all been metamorphosed to amphibolite grade.

- **Amisk Group (Flin Flon arc assemblage):** This group is composed of a thick sequence of metamorphosed volcanic strata. Protoliths range from basalt to rhyolite, with intercalated volcaniclastic metasedimentary rocks. Amisk-age volcanism in the Flin Flon arc began with widespread extrusions of thick sequences of mafic flows that are commonly pillowed. The volcanic succession includes abundant agglomerate, tuff, and ash deposits.

- **The Burntwood Group:** This group is generally comprised of graphitic biotite-quartz gneiss, garnet-biotite gneiss, and occasional amphibole-bearing biotite-quartz gneiss that are interpreted to be derived from greywacke sedimentary rocks (Zwanzig, 1999). Regional workers interpret the Burntwood Group to be a distal sedimentary facies coeval with the Missi Group (Syme, et al., 1995; Ansdell, et al., 1995).

- **The Missi Group:** The Missi Group lies unconformably on the Amisk Group. This regionally important angular unconformity is demarcated by a basal conglomerate with stretched clasts. Protoliths of the Missi Group include conglomerate, sandstone, and other related sedimentary rocks.

7.2 PROPERTY GEOLOGY

The Maverick Project occurs at the boundary of the Kisseynew and Flin Flon domains. This complex boundary zone is comprised of structurally interleaved supracrustal rocks of the Flin Flon Domain and high-grade metamorphic gneisses of the Kisseynew Domain. The Project area is underlain by strongly deformed and metamorphosed para- and orthogneisses. The protoliths of the gneisses are interpreted to be supracrustal rocks of the Amisk Group, metasedimentary rocks of the Missi and Burntwood groups, and gneissose intermediate to felsic intrusives (Figure 7.2). The gneisses have experienced prolonged, progressive deformation that has resulted in polyphase folding associated with ductile and brittle-ductile shear zones. Sulphide and gold mineralization occurs in two, possibly three, pulses throughout the deformation history.

The deposit host rocks can be grouped into four mappable units (Figure 7.2):

The lowermost unit is a distinctive blue-grey tonalite gneiss (tonalite). The grey-blue tonalite is strongly foliated and differentiated in microlithons of quartz-plagioclase and biotite. Traditionally, the tonalite has been considered to be structurally below the PL Deposit and used to determine the termination point of drilling within the deposit. However, recent exploration and resource drilling on the property has determined that the tonalite commonly hosts gold-bearing quartz veins and may represent an under tested exploration target.

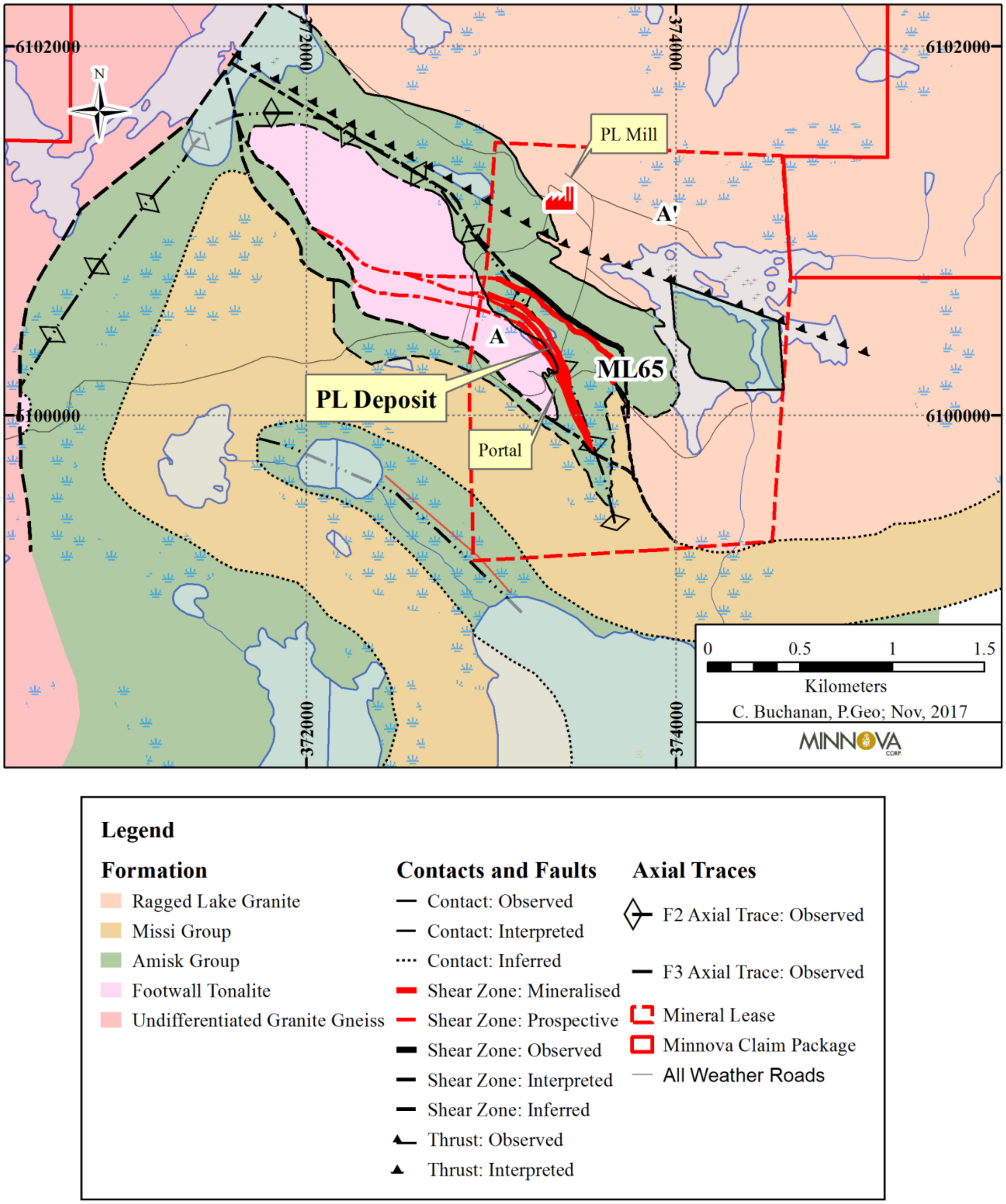


Figure 7.2 Deposit area geology
(Source: Minnova, 2017)

Above the tonalite is a package of transposed Amisk Group metavolcanics interleaved with metagreywacke and occasional metaconglomerate with stretched quartz pebbles. The four main zones of Au mineralization are hosted by brittle ductile shear zones within this 85 m thick package (see Section **Error! Reference source not found.** – Mineralization).

The top of the Amisk Group rocks is demarcated by a sharp contact with an overlying, strongly deformed polymictic metaconglomerate characterised by strongly flattened and stretched oblate cobbles. The metaconglomerate layer is approximately 12 m thick and contains a unique suite of metamorphic minerals comprised of coarse-grain muscovite, sillimanite, and fine-grain magnetite. The metaconglomerate is interpreted to be the regionally significant Missi Group basal conglomerate.

Overlying the metaconglomerate is an 83 m thick package of structurally interleaved biotite-quartz, biotite-garnet, and amphibolite gneisses. The metasedimentary gneisses in the succession are distinguished from the lower package of mineralogically similar rocks by the presence of calcsilicate layers and abundant veins of quartz-plagioclase-tourmaline leucosome. Ostry (1995) interpreted this unit to be a component of the Amisk Group, but it may be a fine-grained member of the Missi Group.

The upper-most unit is a pink granitoid gneiss named the Ragged Lake Granite ("RLG"). The RLG is locally cross-cut by deformed mafic dykes metamorphosed to amphibolite and barren, white quartz veins. The lower contact of the RLG with structurally lower supracrustal rocks is sheared by the Ragged Lake Fault Zone ("RLFZ"). The moderately northeast dipping shear zone is complex and several shear bounded horses along its strike-length have structurally interleaved the granite gneiss and underlying supracrustal rocks (Figure 7.3).

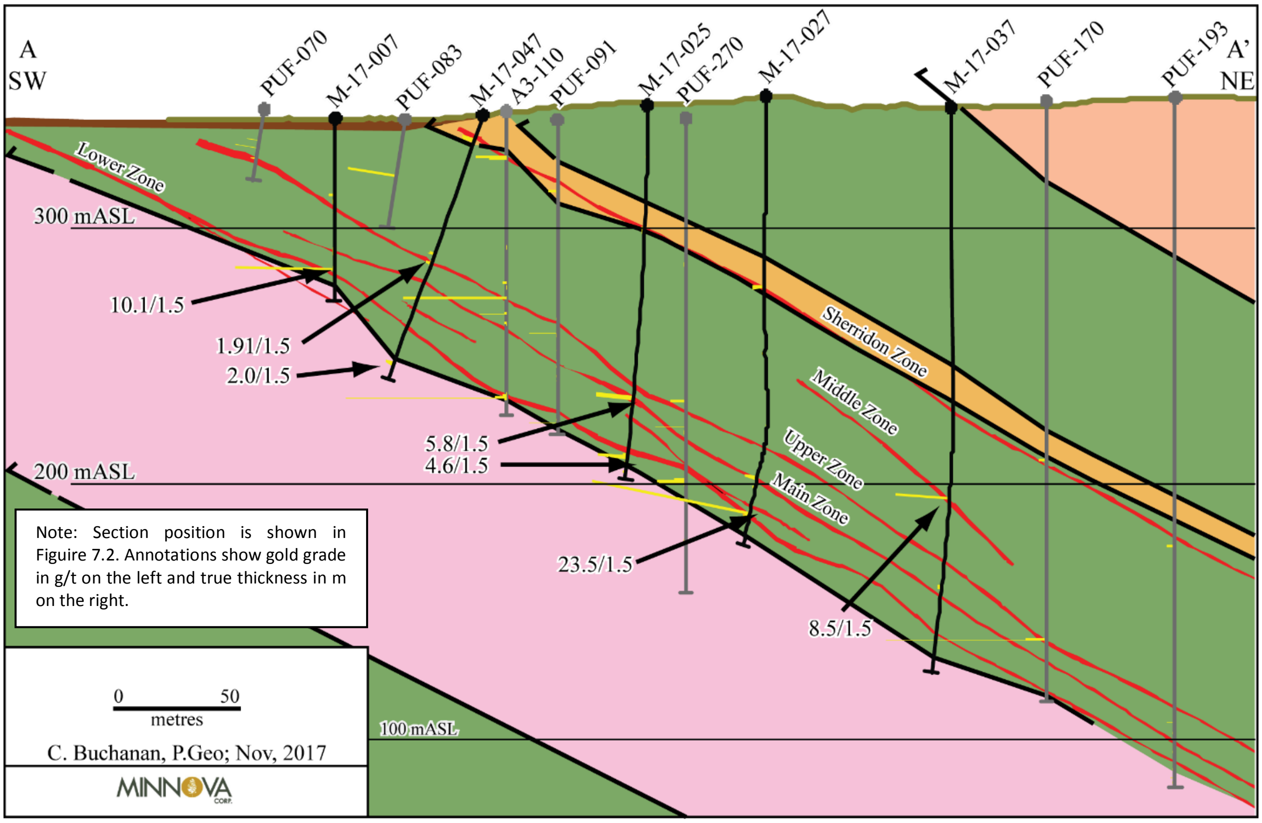


Figure 7.3 PL Deposit cross section A looking northwest
(Source: Minnova, 2017)

The PL Deposit formed on the northeast limb of the Puffy Lake Fold ("PLF"), a large mesoscale F3 fold (Zwanzig, 1999). The PLF is a moderately-plunging, reclined, asymmetric S-fold that is interpreted to be kinematically associated with sinistral displacement along the D3 Nokomis Fault (Figure 7.2). The fold is delineated by a 275 m thick succession of moderately northeast-dipping Missi and Burntwood Group metasedimentary rocks that form a distinctive easterly-trending magnetic anomaly. The metasedimentary rocks were strongly transposed and intercalated by isoclinal F1 folds during the D1 deformation event. The metasedimentary rocks are structurally thickened to approximately 1.7 km across the profile of the PLF.

In the immediate vicinity of the PL Deposit, a Type II fold interference structure, formed by the re-folding of isoclinal F2 folds by the F3-PLF, has preserved the hinge zone of an F2 antiform (Figure 7.2). The F2 fold is southeasterly plunging and tight to isoclinal (Ostry, 1995). The northeastern limb of the fold is comprised of Amisk Group rocks overlain by younger Missi Group metaconglomerates. The geometry of the F2 fold preserves a structurally complicated, but essentially upright stratigraphic section through the Amisk-Missi unconformity.

The northeast limb of the PLF has been structurally dismembered by D3 brittle-ductile shear zones with a reverse-sense of shear. The D3 shear zones, some of which host gold mineralization, are interpreted to have developed after the geometry of the PLF became locked-in and acted as a regional buttress during continued displacement along the Nokomis Fault. The D3 shear zones reactivate and locally crosscut D1 ductile shear zones, forming an imbricate fan on the northeast limb of the PLF (Figure 7.2).The mineralized shear zones are out-of-sequence, higher-order shear zones that form a duplex structure that crosscuts the lower imbricate sheets. The geometry of the duplex structure causes a gentle flexure at the north end of the PL Deposit where the shear zones converge and crosscut the lower tonalite body (Figure 7.2).The complex crosscutting relationships of the D3 shear zones and the out-of-sequence duplex structure indicate progressive deformation over a long period of time (Buchanan, C., 2017).

7.3 MINERALIZATION

The southern flank of the Kisseynew gneiss belt hosts several significant gold deposits including the PL Deposit, the New Britannia Mine, and the Tartan Lake Deposit. These gold deposits occur along the transition between the predominantly early juvenile assemblage of arc related volcanics related to the Amisk Group and the metasediments of the Burntwood Suite – Nokomis Group.

7.3.1 PL DEPOSIT MINERALIZATION

Gold mineralization at the PL Deposit is concentrated along the flexure in the duplex structure and the southeastern strike-extent of the D3 shear zones over a strike length of 1,200 m. The mineralization is controlled by four main shear zones that delineate the duplex structure and are named the Sherridon, Upper, Main, and Lower Zones (Figure 7.4). The shear zones and contained mineralization subcrop, under a northwest-southeast-trending swamp (*i.e.*, Fyre Lake), have been outlined by surface diamond drilling to a vertical depth of 500 m, approximately 1,200 m down dip. The Upper, Main, and Lower Zones occur within 50 m of the sheared tonalite contact and toward the north end of the deposit, the Lower Zone is commonly hosted within the tonalite body.

The gold-bearing shear zones generally strike 330 degree and dip moderately at 30 degree to the northeast, subparallel to the regional foliation (Figure 7.3).The mineralized zones occur in 1.3-2.5 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 25-35 cm. The fault fill veins are generally concordant with the regional foliation, but locally crosscut it. The veins exhibit crack-seal textures and are primarily composed of white quartz. Often, the white quartz is crosscut by a younger phase of translucent quartz that is associated with higher abundance of galena, chalcopyrite, and visible gold. The tension vein arrays have been rotated into masses of white quartz during progressive deformation within the shear zone. The best developed intervals of quartz veins often occur near the contact between metagraywacke and tuffaceous metavolcanics units. This is interpreted to indicate that competency contrast is an important control on the formation of quartz veins and associated gold mineralization.

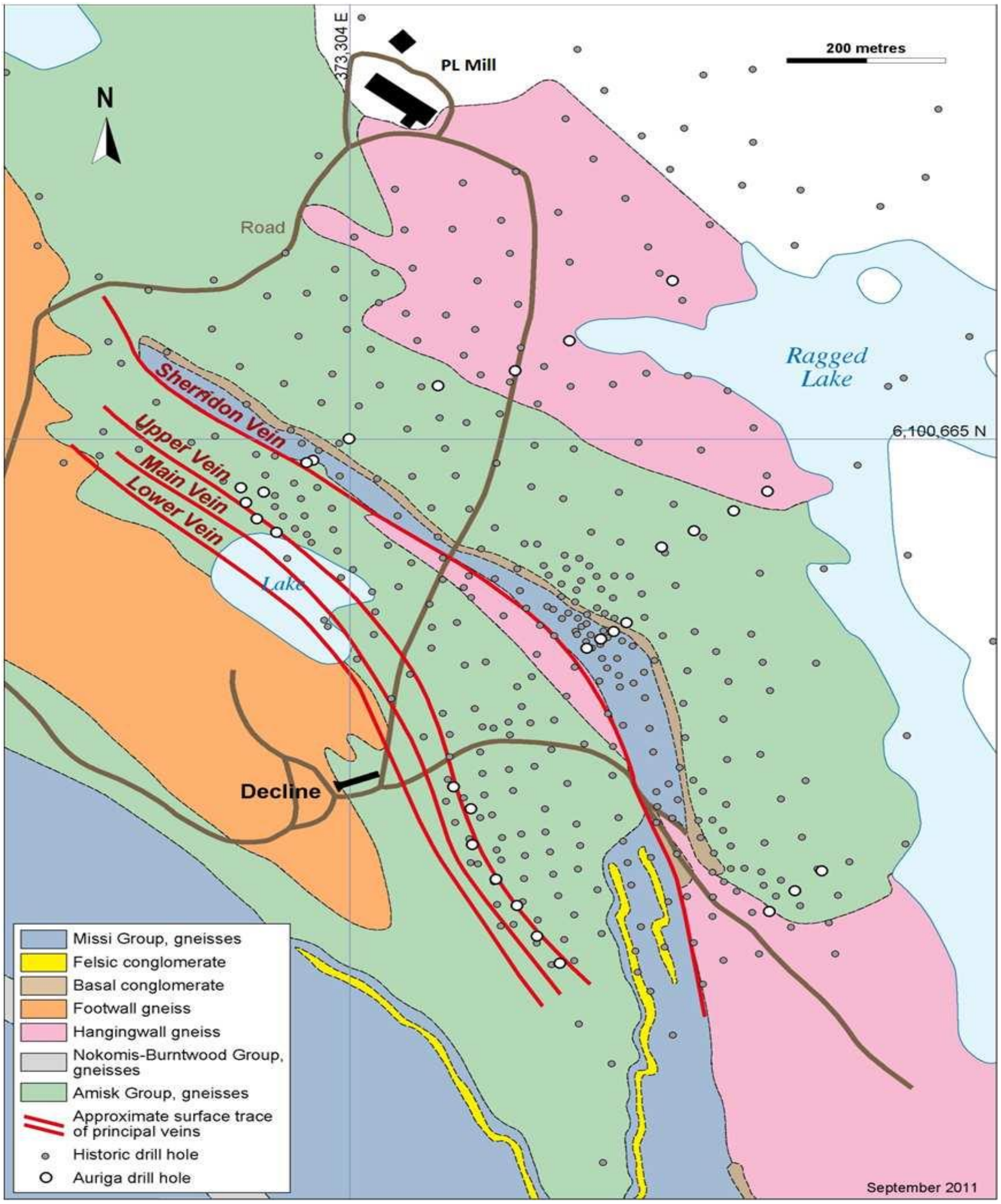


Figure 7.4 Property geology – PL mill area
(Source: Auriga, 2011)

Sulphide mineralization within the shear zones occurs as halos of replacement mineralization around the quartz and in the quartz veins. The assemblage of sulphides is primarily comprised of arsenopyrite-pyrrhotite-pyrite. Trace amounts of galena-chalcopyrite-sphalerite are also present. Visible gold occurs throughout the deposit, but is not common. Arsenopyrite is ubiquitous in the gold-bearing zones occurring as trace to 2%, fine to medium grained disseminations within the host rocks and more importantly, from trace to +5%, associated with the quartz veins, especially along the margins of the veins and along chloritic fractures/slips, or associated with incorporated rafts of the foliated and mineralized wall rock. In general, higher gold grades are associated with increased arsenopyrite content.

Alteration in the deposit is comprised of biotite, muscovite, carbonate, chlorite, and tourmaline. A variety of calc-silicate minerals are also present in the mineralized zones, including: scapolite, diopside, and wollastonite. The paragenetic history of the alteration indicates that the calc-silicate minerals are early and biotite-muscovite-carbonate-chlorite are later associated with the main mineralization event. The relative timing of alteration minerals is interpreted to suggest that the main gold mineralization event occurred during retrograde mineralization and is likely post-peak metamorphism.

The Sherridon Zone occurs structurally higher in the ortho and paragneiss sequence. The central portion of the Sherridon Zone occurs within metres of the sill-like tonalite/granodiorite body. Unlike the Main, Upper, Lower and Lower Two Zones, erratically distributed sulphide minerals (arsenopyrite, pyrite, and pyrrhotite) and free gold at the Sherridon Zone also form veinlets that fill brittle fractures with the quartz veins and occur locally within the host ortho and paragneisses. These veins, that are less than 1 cm in width crosscut the regional schistosity and are interpreted to have formed post peak deformation. They are interpreted as being formed by mobilization of quartz, sulphides and gold from earlier stage mineralization.

7.4 214-TYPE SULPHIDE MINERALIZATION

The 214 Type (Sulphide Type) mineralization has only been observed in the historic underground workings around stope 214. However, it is of some interest since it is reported to have produced the highest grade gold mineralization from the PL Mine. Typically, quartz veins are absent, instead gold is associated with crosscutting, anastomosing, semi-massive, sulphide veins (arsenopyrite ± pyrrhotite, chalcopyrite and pyrite). Ostry and Halden (1995) interpreted the 214 Type mineralization as being formed by the late stage remobilization of the more dominant vein type mineralization described above.

7.5 NOKOMIS MINERALIZATION

The Nokomis gold mineralization is hosted within a shear related, intrusive hosted, lode gold system. The Nokomis Deposit is made up of the mineralized Upper and Lower Host Zones that have been separated by a steeply dipping normal fault that is intruded by pegmatite (Figure 7.5 and Figure 7.6).

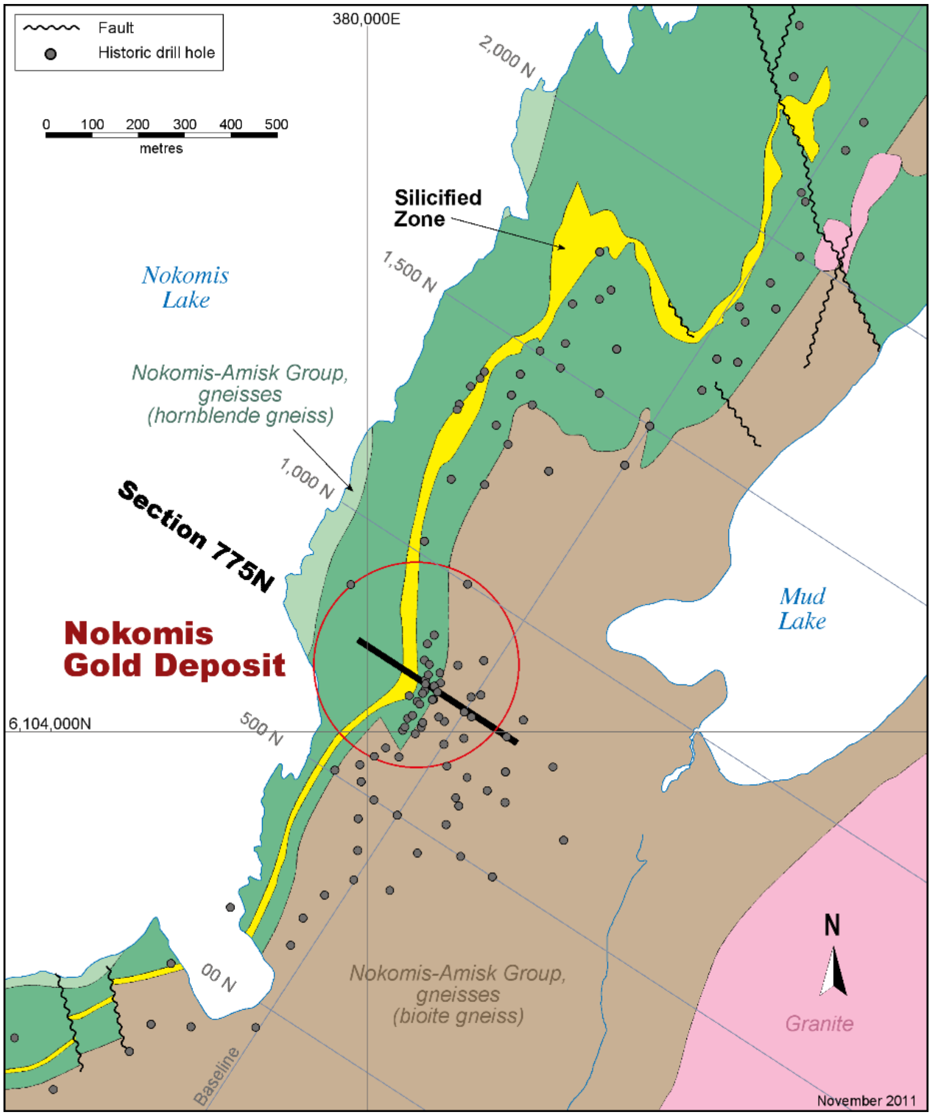


Figure 7.5 Property geology – Nokomis area

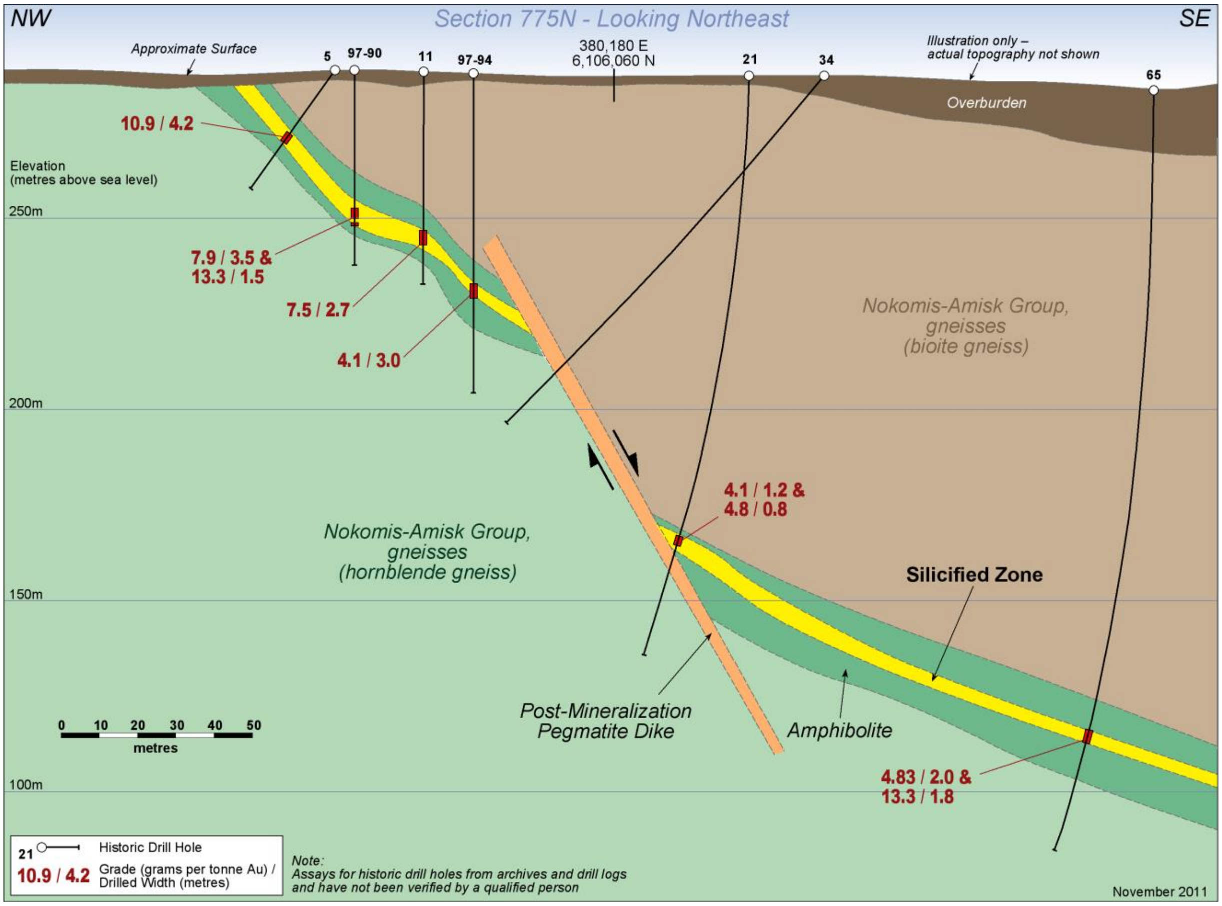


Figure 7.6 Nokomis schematic cross section 775NE

Gold mineralization occurs within a differentiated gabbroic sill emplaced near the contact between metasediments of the Burntwood Suite-Nokomis Group and mafic volcanics of the Amisk Lake group. The host unit for the gold mineralization has been described as a ferrotonalite to diorite. Alteration includes silicification, carbonatization, and albitization.

The Upper and Lower Host Zones range in thickness from <1 m to 4 m and have been intersected by diamond drilling over a surface strike length of approximately 450 m and a down dip extent of more than 350 m, to a vertical depth of 200 m below surface.

8 DEPOSIT TYPES

8.1 PL GOLD PROPERTY

The Maverick Project hosts mesothermal-type gold deposits with multiple veins and lenses.

8.1.1 PL DEPOSIT

Auriferous sulphide-bearing quartz veins and zones of quartz free gold-sulphide mineralization occur in a biotite rich gneiss/schist. Gold-sulphide mineralization also occurs with or without quartz within fine to coarse grained felsic quartzofeldspathic gneiss.

At the PL Deposit, the recognition of an early mineralising event and subsequent remobilization of the sulphides and gold offers an alternate model for the distribution of at least some of the gold sulphide mineralization. The mobilization of arsenopyrite, sulphides, and gold into structural traps that form high-grade 214 type shoots would provide new exploration targets in addition to the typical auriferous sulphide-bearing quartz veins.

8.2 NOKOMIS PROPERTY

8.2.1 NOKOMIS DEPOSIT

At the Nokomis Deposit, gold mineralization occurs within a shear zone cutting a differentiated gabbroic sill emplaced near the contact between the metasediments of the Nokomis group and mafic volcanics of the Amisk Lake group. The host unit for the gold mineralization has been described as a ferrotonalite to diorite. Alteration includes silicification, carbonatization, and albitization. The Nokomis Deposit is made up of the mineralized Upper and Lower Host Zones that have been separated by a steeply dipping normal fault that is intruded by pegmatite (Figure 7.5 and Figure 7.6).

8.2.2 MESOTHERMAL GOLD DEPOSITS

Mesothermal Gold Deposits are mainly quartz vein related, gold-only deposits with associated carbonitized wall rocks. They occur in low-to medium-grade metamorphic terrains of all ages, but only in those that have been intruded by granitoid batholiths (Hodgson, 1993). On a regional scale, the deposits occur in pro-grading arc-trench complexes in association with major transcrustal fault zones, linear belts of fluviatile to shallow-marine sedimentary rocks, and small felsic alkalic and trondhjemitic intrusions, a co-spatial assemblage of structures and rocks that developed after the main period of accretion-related contractional deformation, but before much of the metamorphism and penetrative fabric. At the PL Deposit, the strong lineation of 201-type arsenopyrite and the presence of a mineral assemblage indicative of a metamorphosed alteration (*e.g.*, assemblage of coarse-grained diopside, Ca-amphibole, and calcite associated with sulphide mineralization) strongly suggests a pre-peak metamorphism origin for the deposit (Gagne, et al., 2006).

One broad group of Mesothermal Gold Deposits is hosted in belts dominated by volcanic rocks, such as the greenstone belts in the Precambrian Superior Province of the Canadian Shield. One subclass of this group is auriferous quartz veins and veinlet systems.

Most Mesothermal Gold Deposits in supracrustal rock belts dominated by volcanic rocks are controlled by shear zones and by fracture systems related to shear zones. Shear Zone systems range from brittle to ductile, although at least some ductile shears have overprinted brittle shears and vice versa. Most mineralization is in shear veins within shear zones and in the immediate wall rocks.

Most gold-bearing mineralized zones are localized in fractures and schist zones that have dilated. Patterns of veins indicate that in most deposits, dilation is an integral part of the bulk inhomogeneous strain due to the alternating increments of simple shear along intersecting fracture sets.

Ore fluids are carbon dioxide rich and have been attributed to magmas, metamorphic devolatilization of supracrustal rocks, and mantle degassing. Commonly associated minerals include pyrite (less commonly pyrrhotite), common base-metal sulphides, arsenopyrite, tourmaline, and molybdenite.

9 EXPLORATION

9.1 PAST EXPLORATION

For a discussion of exploration prior to Minnova's acquisition of the property in 2010, see Section 6 – History.

9.2 LIDAR SURVEY

In 2011, Minnova (then Auriga) contracted Eagle Mapping Ltd., Port Coquitlam, British Columbia to undertake a 6,171 ha LiDAR survey of the Maverick Project to provide a detailed digital elevation model for both engineering and exploration purposes.

9.3 GEOPHYSICS

In April 2012, Geotech Ltd. of Newmarket, Ontario conducted an airborne VTEM and magnetic survey covering the PL and Nokomis properties (Figure 9.1 and Figure 9.2). Eight hundred sixty-six (866) line km were flown covering a total area of 66 km^2. The VTEM survey identified several untested conductive anomalies along the regional mineralized structure that hosts the PL and Nokomis Deposits. The anomalies displayed similar characteristics in size and geophysical response to the anomalies associated with the PL and Nokomis Deposits.

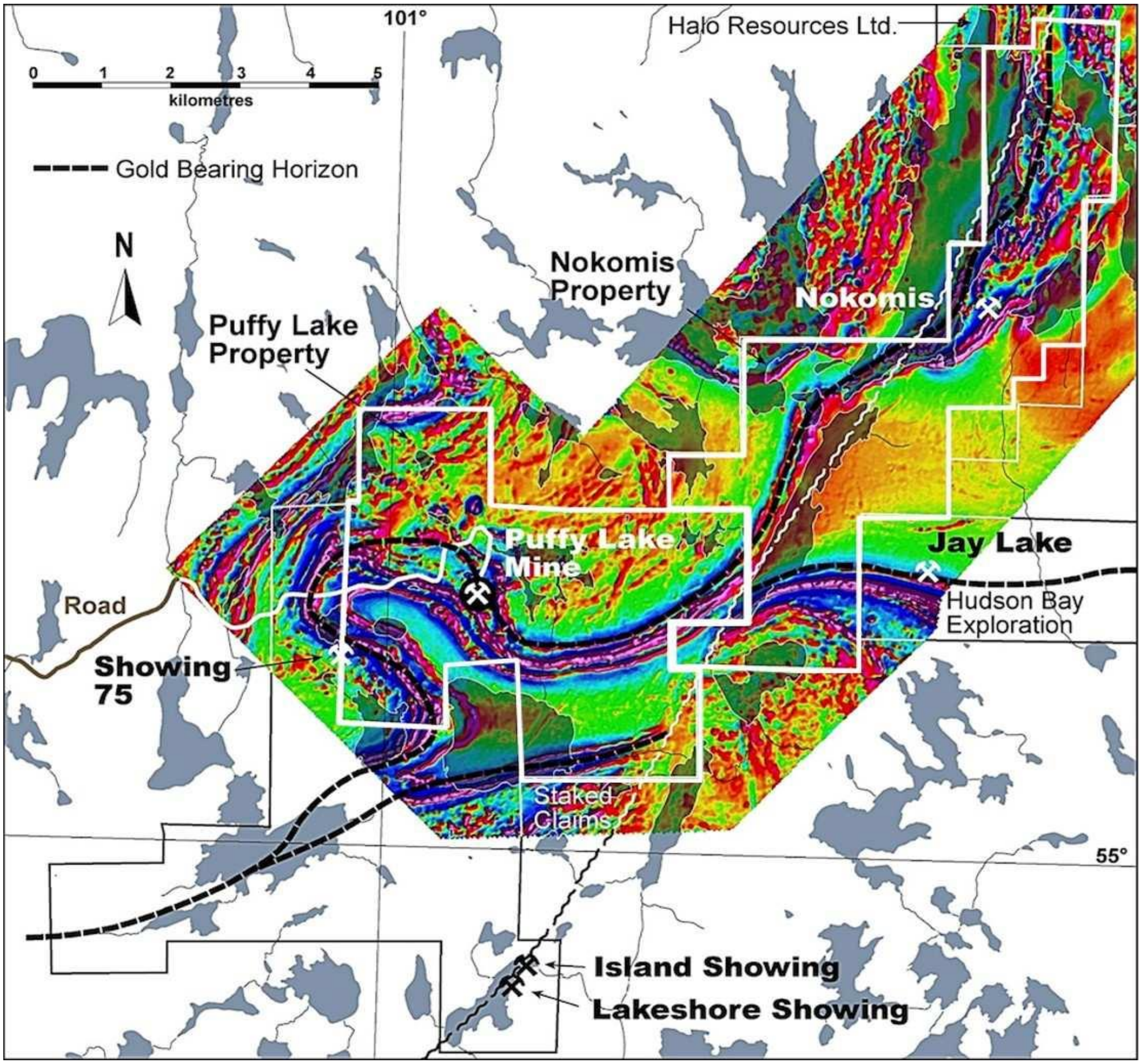


Figure 9.1 Magnetic survey over the PL and Nokomis properties

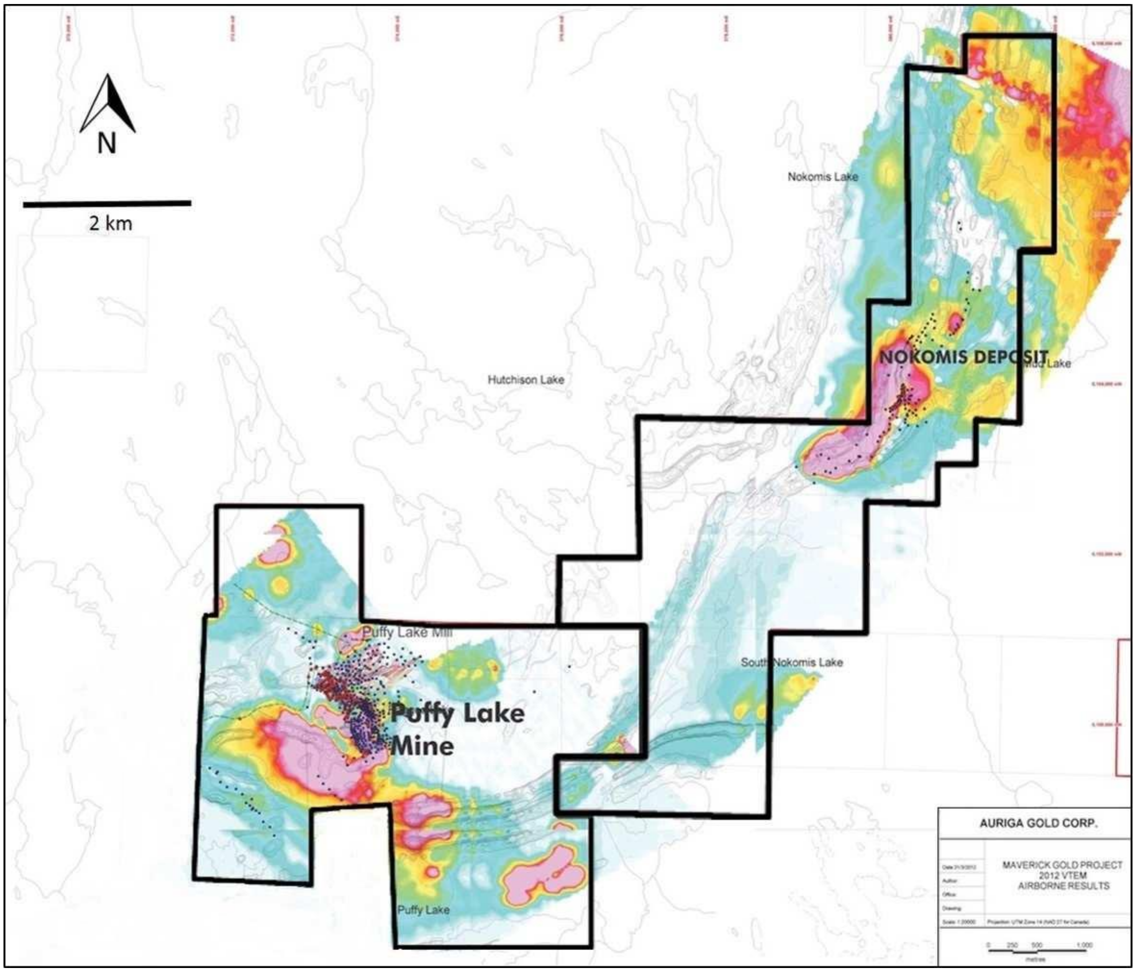


Figure 9.2 VTEM survey over the PL and Nokomis properties

9.4 FIELD PROGRAM

Minnova completed a surface exploration program from May to October 2017. The program focused on property scale geological mapping, prospecting, and re-logging of historical core. The field program resulted in an updated geological map, 88 rock sample analyses, and the discovery of 6 new gold showings. All the samples and showings are within a 1.8 km radius of the existing mill (Figure 9.3).

The showings are primarily hosted in supracrustal rocks that are similar to the Amisk Group rocks that host the PL Deposit. The exception is the Tonalite Showing, which is hosted in the lower tonalite body. Brief summaries of each showing are described below with significant assays presented in Figure 9.3.

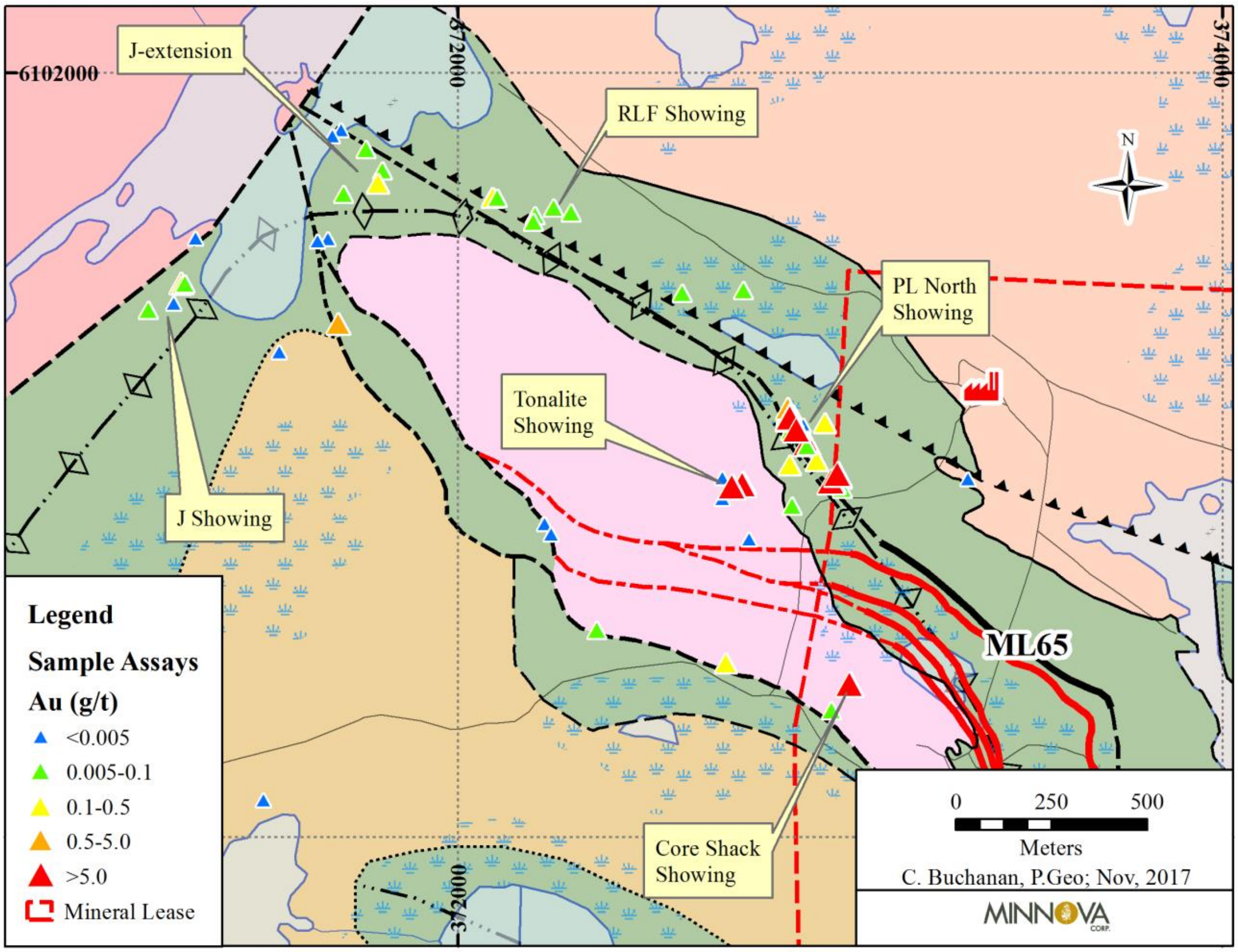


Figure 9.3 2017 Surface rock sampling results
(Source: Minnova, 2017)

The recognition of gossan zones in limited outcrop led to the discovery of the J and J-extension showings hosted by biotite-quartz gneiss along the eastern shoreline of Jay Lake. The host rocks are crosscut by sub-vertical shear zones that contain abundant alteration and sulphide mineralization. Grab sample assay values range from <0.005 g/t Au to 0.954 g/t Au.

The RLF Showing is located along the interpreted trace of the Ragged Lake Fault between the PL Mill and Jay Lake. The showing occurs in scattered outcrop within a swamp. The host rocks are biotite-quartz and biotite-garnet gneisses interpreted to be part of the Amisk Group. Arsenopyrite and pyrite mineralization is coincident with gold mineralization. Grab sample assay values range from 0.007 g/t Au to 0.119 g/t Au. Although the grab samples are low grade, the presence of gold-bearing quartz veins and Amisk Group host rocks make this poorly exposed swamp area a prospective exploration target.

The PL North Showing is located northwest of the main PL Deposit and is shown in Figure 9.4. The showing consists of high-grade grab and chip samples along a 300 m trend that strikes approximately 330 degrees and dips moderately to the northeast. The initial discovery was a 40 cm thick quartz vein exposed on surface that assayed 45 g/t Au in a grab sample. Subsequent chip samples of the vein yielded assay values of 0.020-20.4 g/t Au over 1.0 m. Follow-up prospecting along the strike of the initial vein discovery yielded additional gold-bearing quartz veins and foliation-controlled arsenopyrite-pyrrhotite-pyrite-gold replacement mineralization to the southeast. Samples collected from the PL North occurrences range in value from <0.005-45 g/t Au in the previously described grab sample. The style of mineralization (both quartz vein hosted and replacement) and the Amisk Group host rocks are similar to the PL Deposit. It is unclear if PL North is the strike extension of the PL Deposit or a newly discovered mineralized shear zone within the system. Further surface exploration and diamond drilling is required to resolve these questions.

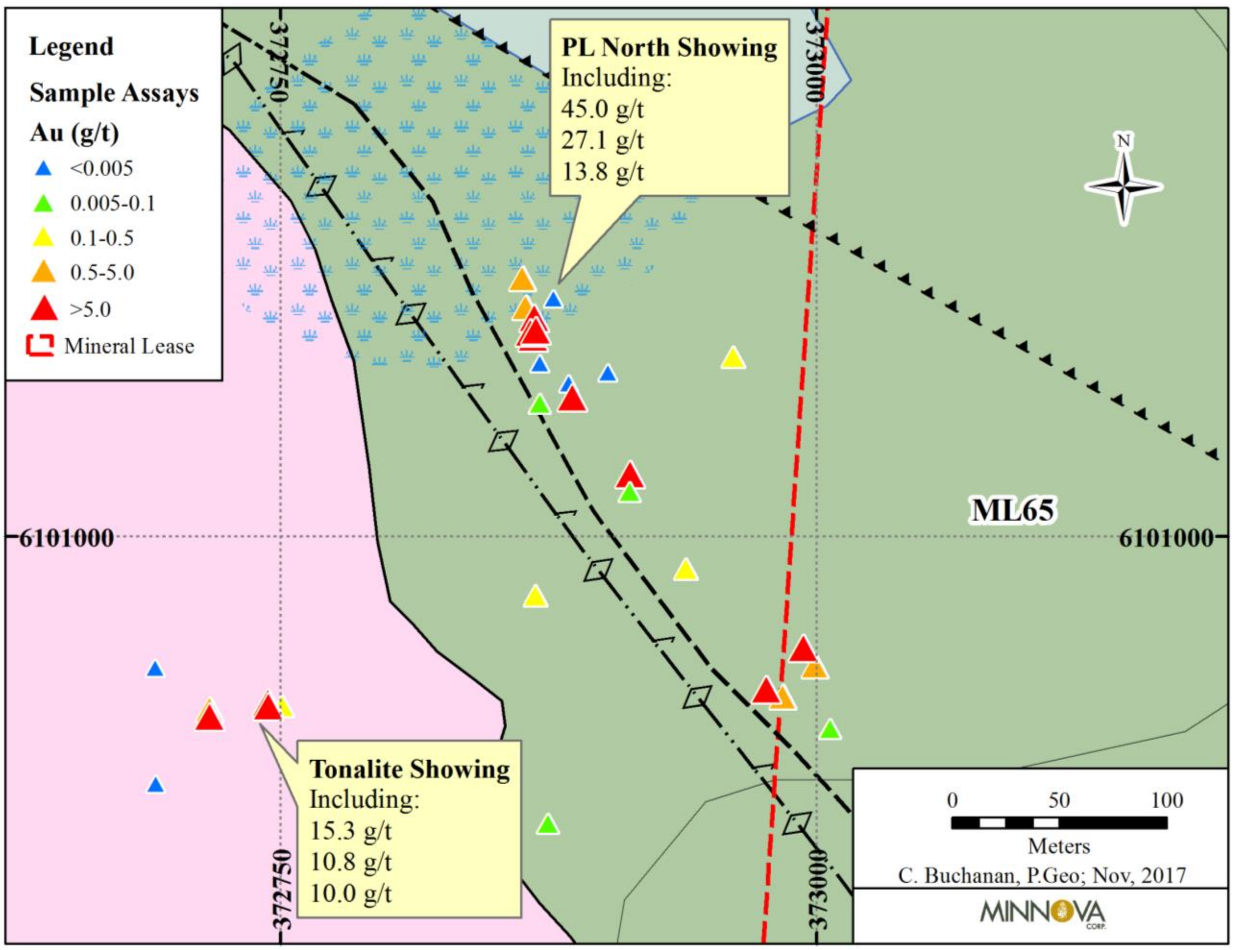


Figure 9.4 2017 surface rock sampling results – PL north and tonalite showings (Source: Minnova, 2017)

Two new tonalite-hosted showings were discovered during mapping of the tonalite contact. The northern showing is located 625 m southwest of the mill and the second is located 400 m northwest of the portal. The gold mineralization is associated with fine-grained arsenopyrite and chalcopyrite disseminated on the foliation planes of the tonalite gneiss. Quartz veins, potassic alteration, tourmaline, and silicification are associated with high-grade gold samples at both showings. The assay results from these two showings range from <0.005-15.3 g/t Au. This is a new style of gold mineralization, alteration, and host rocks near the PL Deposit. Although an encouraging exploration target, additional sampling, trenching, and drilling will be required to determine how these showings relate to the PL Deposit (Table 9.1).

Table 9.1: Surface Samples – Significant Results						
Prospect	**Sample Number**	**Easting**	**Northing**	**Sample Type**	**Length (m)**	**Au (g/t)**
J	E00016974	371275.43	6101449.01	Grab		0.954
J	E00016983	371686.84	6101343.08	Grab		0.564
J ext	389048	371790.69	6101711	Grab		0.147
PL North	389011	372999.3	6100939.56	Grab		2.570
PL North	389030	6101091	372869	Chip	1.00	7.150
PL North	389031	6101091	372869	Chip	0.50	1.660
PL North	389033	6101091	372869	Chip	1.00	0.116
PL North	389036	6101091	372869	Chip	1.00	0.020
PL North	389037	6101091	372869	Chip	1.00	20.400
PL North	389039	372864.47	6101106.86	Grab		1.670
PL North	389040	372862.81	6101120.26	Grab		2.380
PL North	389042	372913.29	6101029.19	Grab		9.270
PL North	389043	372867.98	6101092.95	Grab		27.100
PL North	E00016985	372961.33	6101083.83	Grab		0.358
PL North	E00016986	372868.43	6101101.95	Grab		8.100
PL North	E00016987	372866.7	6101095.1	Grab		13.800
PL North	E00016989	372869.29	6101096.25	Grab		45.000
PL North	E00016992	372984.31	6100925.07	Grab		3.830
PL North	E00016993	372976.74	6100928.73	Grab		5.040
PL North	E00016994	372994	6100948	Grab		5.870
PL North	E00016995	372939	6100985	Grab		0.372
PL North	E00016997	372886.23	6101064.93	Grab		9.390
PL North	E00016999	372868.96	6100972.33	Grab		0.242
RLF	389106	372250.33	6101646.49	OC		0.076
RLF	389110	372091.89	6101672.85	Grab		0.119
Tonalite	388505	372750.92	6100920.91	Grab		0.307
Tonalite	389007	372716.98	6100919.32	Grab		3.200
Tonalite	389008	372744.32	6100922.66	Grab		10.800
Tonalite	389044	372744.46	6100920.98	Grab		10.000
Tonalite	E00016991	372717	6100916	Grab		15.300
	389004	372701.77	6100455.1	Grab		0.362

The 2017 summer exploration program successfully discovered new gold showings in the PL Property package. The discovery of high-grade gold samples and their spatial distribution suggests exploration potential for additional gold mineralized zones in the PL Property and PL Deposit area.

10 DRILLING

10.1 HISTORIC DRILLING

Historic drilling conducted on the Maverick Project (prior to 2010) is summarised in Section 6 – History of this report.

Historical work on the Maverick Project was not reported in the manner currently required under NI 43-101; however, the Author is of the opinion that the geologists of the era followed drill core logging and sampling procedures and assaying techniques that would have been considered industry standard at the time. Detailed written descriptions of the procedures, however, were rarely recorded in technical reports and insertion of QA/QC standards and blanks into the sample batches would not have been standard practice at the time.

10.2 MAVERICK PROJECT DRILLING

Diamond drill programs completed by Minnova and its predecessor, Auriga, are summarised in Table 10.1 and discussed in the following subsections.

Table 10.1: Minnova/Auriga Diamond Drill Programs 2010-2017

Property	Year	Company	Number of Holes	Hole IDS	Total (m)
PL Property	2010-2011	Auriga (now Minnova)	47	A3-001-A3-047	4,950
	2011-2012	Auriga (now Minnova)	61	A3-048-A3-114	5,211
	2013	Auriga (now Minnova)	4	A3-115-A3-118	697
	2016	Minnova	2	M-16-001-M-16-002	915
	2017	Minnova	48	M-17-003-M-17-050	8,523
Total PL Property			**162**		**20,296**
Nokomis Property	2014	Minnova	19	A-4-001-A4-019	1,335

10.3 PL PROPERTY DRILLING

10.3.1 PHASE 1: 2010-2011 DRILL PROGRAM

Minnova completed a two-phase diamond drill program in 2010-2011 totaling 5,816 m in forty-seven (47) holes. An initial diamond drill program of fifteen (15) diamond drill holes (A3-01 to A3-15) totalling 3,429 m commenced in December 2010 and was completed in March 2011. The aim of the program was to confirm historical mineral resources at the past producing PL Mine (formerly the Puffy Lake Mine). Bore holes A3-01 to A3-07, A3-09, and A3-12 intersected gold mineralization associated with quartz veins and silicification within the host rocks. This drilling intersected several high grade mineralized veins and confirmed several parallel gold mineralized zones that dip toward the northeast at approximately 30 degrees.

Drilling continued in March 2011 with the intention of testing the potential of shallow, open pit mining of the mineralized vein system. The program consisted of thirty-two (32) diamond drill holes (A3-16 to A3-47) totalling 2,387 m. Holes A3-16 to A3-29 inclusive tested the near surface potential in the southern area of the mine area near the portal. Holes A3-030 to A3-033 and A3-035 tested the Sherridon Vein where the vein comes close to the surface while Holes A3-034 and A3-036 to A3-047 tested the near surface potential in the northern mine area.

Drill hole locations and collar data are presented in Table 10.2 and Figure 10.1.

Table 10.2: Phase 1: 2010-2011 Diamond Drill Hole Location and Collar Data

DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
A3-001	373517.1	6100748	351.53	-90	360	299.7
A3-002	373416.6	6100724	347.46	-90	360	233.77
A3-003	373312	6100652	346.94	-90	360	169.8
A3-004	373258.5	6100635	344.62	-90	360	142.3
A3-005	373260.1	6100635	344.86	-59	235	114.9
A3-006	373197.5	6100585	342.32	-90	360	87.48
A3-007	373571.5	6100802	348.39	-90	360	328.3
A3-008	373701.2	6100883	351.07	-90	360	352.64
A3-009	373812.4	6100001	350.94	-90	360	185
A3-010	373860.3	6100042	351.37	-90	360	221.58
A3-011	373887.2	6100071	351.69	-90	360	233.77
A3-012	373692.3	6100520	351.87	-90	360	236.8
A3-013	373732.5	6100538	351.69	-90	360	249.01
A3-014	373781.1	6100566	351.36	-90	360	273.4
A3-015	373812.9	6100591	353.22	-90	360	300.82
A3-016	373435.7	6100175	344.04	-90	360	57
A3-017	373435.2	6100175	344.04	-45	235	32.6
A3-018	373457.8	6100152	341.57	-90	360	44.8
A3-019	373457.4	6100152	341.49	-45	235	33.9
A3-020	373455.1	6100102	341.56	-90	360	35.7
A3-021	373454.5	6100102	341.61	-45	235	35.7
A3-022	373484.3	6100053	341.54	-90	360	47.9
A3-023	373483.5	6100053	341.53	-44.1	242.1	41.8
A3-024	373509.1	6100013	341.43	-90	360	72.23
A3-025	373508.8	6100013	341.4	-45	235	69.17
A3-026	373539.2	6099970	341.53	-90	360	44.8
A3-027	373538.9	6099970	341.53	-45	235	17.81
A3-028	373565.1	6099937	341.53	-90	360	41.76
A3-029	373564.3	6099936	341.57	-45	235	8.4
A3-030	373599.5	6100380	345.67	-90	360	26.52
A3-031	373613.7	6100390	345.98	-90	360	32.61

Table 10.2: Phase 1: 2010-2011 Diamond Drill Hole Location and Collar Data						
DDH No.	**Easting**	**Northing**	**Elevation (m)**	**Dip (°)**	**Azimuth (°)**	**Length (m)**
A3-032	373628	6100405	347.8	-90	360	38.71
A3-033	373645.1	6100410	348.73	-90	360	23.47
A3-034	373222.5	6100538	342.1	-90	360	81.38
A3-035	373646.4	6100407	348.1	-90	360	50.9
A3-036	373196.3	6100554	342.16	-90	360	63.09
A3-037	373195.5	6100553	342.18	-45	235	63.09
A3-038	373182.5	6100574	342.18	-90	360	60.04
A3-039	373182.5	6100574	342.15	-45	235	57
A3-040	373177.9	6100597	342.1	-90	360	63.09
A3-041	373177.8	6100597	342.11	-45	235	57
A3-042	373362.4	6100708	352.31	-84	216.1	194.15
A3-043	373623.6	6100881	348.98	-90	0	361.78
A3-044	372912.2	6100778	346.03	-90	0	69.19
A3-045	372942.8	6100861	345	-90	0	63.09
A3-046	373477.6	6101075	355.04	-90	0	389.33
A3-047	373188.7	6100694	342.36	-90	0	109.11
Total	**47**					**5816.39**

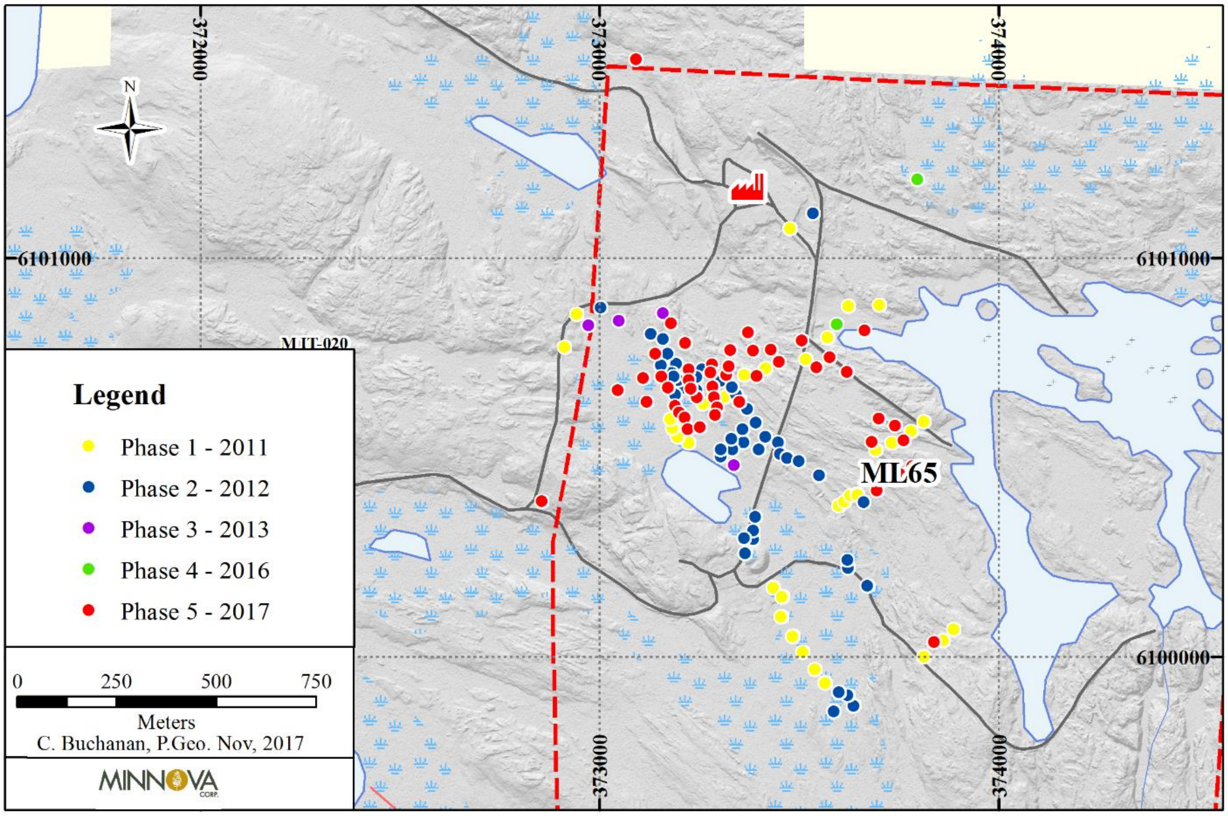


Figure 10.1 PL Property Minnova drill hole locations

A total of approximately 1,931 core samples (excluding QA/QC samples) were collected from the 2010-2011 drill core and sent to Accurassay. A list of significant intersections is presented in Table 10.3.

Table 10.3: Highlights of Drill Intercepts from the Phase 1 2010-2011 Diamond Drill Program				
Hole No.	**From (m)**	**To (m)**	**Core Length[1] (m)**	**Au (gpt)**
A3-01	245.20	245.70	0.50	3.71
and	247.20	247.70	0.50	8.17
and	251.25	251.75	0.50	28.79
A3-05	52.77	55.27	2.5	11.99
including	53.27	53.77	0.50	52.61
and	80.69	81.19	0.50	65.42
and	90.97	92.40	1.43	5.73
A3-06	8.77	9.61	0.84	18.65
and	35.16	35.41	0.25	11.23
and	52.21	52,96	0.75	4.18
A3-02	175.75	177.75	2.00	9.19
including	176.25	176.75	0.50	23.80
A3-04	58.80	59.35	0.55	3.32
and	63.50	65.00	1.50	11.88

Table 10.3: Highlights of Drill Intercepts from the Phase 1 2010-2011 Diamond Drill Program				
Hole No.	From (m)	To (m)	Core Length[1] (m)	Au (gpt)
including	63.50	64.00	0.50	28.55
A3-07	298.10	300.10	2.00	30.44
A3-09	165.13	165.64	0.51	9.64
A3-12	216.80	217.80	1.00	2.17
A3-03	96.98	98.48	1.50	7.73
and	129.07	130.57	1.50	7.88
A3-16	8.20	13.20	5.00	6.59
including	11.20	13.20	2.00	11.98
A3-17	12.10	14.50	2.40	2.77
A3-18	No significant assays			
A3-19	18.50	18.90	0.40	19.06
A3-20	10.80	12.80	2.00	2.99
A3-21	8.60	11.80	3.20	3.60
A3-22	16.10	17.60	1.50	1.29
and	23.60	29.10	5.50	1.18
A3-23	13.20	15.45	2.25	2.29
and	22.50	23.50	1.00	4.06
A3-24	21.29	21.50	0.21	43.85
and	30.90	33.55	2.65	5.08
A3-25	21.29	21.50	0.21	43.85
and	31.47	34.23	2.76	2.99
A3-26	29.86	32.23	2.37	7.08
including	30.36	30.90	0.54	22.74
and	31.47	34.23	2.76	2.99
A3-26	29.86	32.23	2.37	7.08
including	30.36	30.90	0.54	22.74
and	40.01	40.32	0.31	6.30
A3-27	Intersected former mine workings			
A3-28	25.78	27.40	1.62	2.30
and	29.80	30.16	0.36	5.32
and	35.14	35.95	0.81	9.05
A3-34	65.70	66.70	1.00	5.44
A3-36	17.37	17.85	0.48	8.02
and	54.60	56.32	1.72	28.41
including	55.32	56.32	1.00	44.06
and	63.50	64.00	0.50	28.55
A3-37	14.73	16.01	1.28	9.57
and	36.26	38.46	2.20	16.25
including	36.26	36.75	0.49	42.63
and	47.28	48.00	0.72	13.09
A3-08	298.78	299.28	0.50	2.37

Table 10.3: Highlights of Drill Intercepts from the Phase 1 2010-2011 Diamond Drill Program				
Hole No.	**From (m)**	**To (m)**	**Core Length[1] (m)**	**Au (gpt)**
A3-10	198.35	203.13	4.78	7.85
A3-11	208.08	209.2	1.12	3.69
A3-13	156.13	157.69	1.56	10.48
Sherridon Zone – Near Surface Mineralization				
A3-30	13.26	15.60	2.34	21.49
including	13.26	14.11	0.85	57.26
A3-31	17.53	20.42	2.89	4.94
including	20.07	20.42	0.35	21.02
A3-32	31.62	33.29	1.67	15.79
including	32.30	32.75	0.45	55.91
A3-33	Hole terminated before it intersected zone			
A3-08	298.78	299.28	0.50	2.37
A3-10	198.35	203.13	4.78	7.85
A3-11	208.08	209.2	1.12	3.69
A3-13	156.13	157.69	1.56	10.48
A3-35	38.35	42.38	4.03	2.63
including	39.85	41.85	2.00	4.30
Northern – Near Surface Mineralization				
A3-38	55.70	57.25	1.55	6.26
A3-39	23.83	24.16	0.33	9.19
and	36.51	41.56	5.05	4.79
including	37.01	40.51	3.50	6.49
including	37.51	38.01	0.50	23.26
and	46.00	46.25	0.25	27.55
A3-40	No significant assays			
A3-41	46.35	51.05	4.70	0.90
[1]True widths are currently estimated at 85-90% of drilled widths				

10.3.2 Phase 2: 2011-2012 Drill Program

In August 2011, Minnova began Phase 2 of drilling at the Maverick Project. The aim of the drill program was to expand the gold resources and increase the quality and quantity of the optimised near surface in-pit resources on the PL Property, as well as examine other gold occurrences along the 20 km long trend on the property. A total of fifty-seven (57) holes totalling 5,211 m were completed (Table 10.4 and Figure 10.1).

Table 10.4: Phase 2: 2011-2012 Drill Hole Location Data						
DDH No.	**Easting**	**Northing**	**Elevation (m)**	**Dip (°)**	**Azimuth (°)**	**Length (m)**
A3-48	372072	6100286	345	-60	225	377
A3-49	373000	6100880	344.2	-90	0	60.04

Table 10.4: Phase 2: 2011-2012 Drill Hole Location Data						
DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
A3-50	373410	6100550	345.8	-90	0	72.23
A3-51	373443	6100540	348.1	-90	0	50.9
A3-52	373397	6100525	344.7	-90	0	124.05
A3-53	373397	6100525	344.7	-45	235	63.09
A3-54	373453	6100511	346.2	-45	235	63.09
A3-55	373453	6100511	346.2	-90	0	57
A3-56	373469	6100502	346.2	-90	0	53.95
A3-57	373469	6100502	346.2	-45	235	41.76
A3-58	373497	6100494	346.1	-90	0	75.28
A3-59	373548	6100457	346.1	-90	0	47.85
A3-60	373663	6100393	347.5	-90	0	57
A3-61	373352	6100644	348.8	-90	0	63.09
A3-62	373341	6100659	349.3	-90	0	69.19
A3-63	373366	6100626	347.3	-90	0	78.33
A3-64	373370	6100260	347.8	-45	70	35.66
A3-65	373385	6100295	343.7	-90	0	36.35
A3-67	373382	6100320	343.1	-90	0	38.71
A3-68B	373358	6100298	343.2	-90	0	32.61
A3-69	373390	6100355	343.1	-90	0	47.85
A3-71	373301	6100696	348.5	-90	0	170.08
A3-72	373359	6100545	344.3	-90	0	127.1
A3-73	373330	6100530	343	-83	194	130.14
A3-74	373307	6100511	342.8	-90	0	96.62
A3-75	373382	6100594	346.9	-90	0	26.52
A3-76	373357	6100573	346	-90	0	139.29
A3-77	373328	6100550	343	-90	0	124.05
A3-78	373298	6100530	342.8	-90	0	102.71
A3-79	373272	6100732	348.1	-90	0	176.17
A3-80	373245	6100708	345	-90	0	139.59
A3-81	373209	6100681	343.6	-90	0	122.15
A3-82	373226	6100761	346.7	-90	0	182.26
A3-83	373198	6100736	344.2	-90	0	154.83
A3-84	373162	6100706	342.4	-90	0	125.04
A3-85	373162	6100761	344.2	-90	0	176.17
A3-86	373155	6100732	343.3	-90	0	153.92
A3-87	373669	6100183	341.8	-90	0	41.76
A3-88	373669	6100183	341.8	-45	235	35.05
A3-89	373669	6100218	342.2	-90	0	26.52
A3-90	373623	6100218	342.1	-55	235	28.96
A3-91	373621	6100241	342.9	-90	0	21.98
A3-92	373622	6099873	342.4	-90	0	66.14
A3-93	373622	6099873	342.4	-55	235	62.8

Table 10.4: Phase 2: 2011-2012 Drill Hole Location Data						
DDH No.	**Easting**	**Northing**	**Elevation (m)**	**Dip (°)**	**Azimuth (°)**	**Length (m)**
A3-94	373620	6099905	341.9	-90	0	63.09
A3-95	373620	6099905	341.9	-55	235	56.39
A3-96	373578	6099875	342.2	-45	235.1	35.05
A3-97	373578	6099875	342.2	-90	0	53.95
A3-98	373596	6099916	341.8	-90	360	60.35
A3-99	373596	6099916	341.8	-55	235	69.49
A3-100	373334	6100683	349.9	-81	216.7	161.54
A3-103	373131	6100808	345	-90	360	164.58
A3-104	373214	6100659	342.9	-90	360	96.92
A3-105	373161	6100803	345.4	-90	360	155.44
A3-109	373189	6100663	342.5	-90	0	93.87
A3-110	373238	6100673	345	-90	360	118.87
A3-114	373190	6100705	342.8	-82	189.3	106.71
Total	**57**					**5,211.13**

A total of approximately 1,828 core samples (excluding QA/QC samples) were collected from the 2011-2012 drill core and sent to Accurassay. A list of significant intersections is presented in Table 10.5.

Table 10.5: Highlights of Drill Intercepts from the Phase 1: 2011-2012 Diamond Drill Program				
Hole Number	**From (m)**	**To (m)**	**Core Length[1] (m)**	**Au (gpt)**
A3-48	335.65	334.15	0.50	23.68
A3-51	31.86	36.13	4.27	1.09
A3-53	4.30	6.67	2.37	1.45
A3-59	22.63	24.60	1.97	3.70
A3-60	39.05	43.40	4.35	3.72
including	39.05	40.90	1.85	3.63
and	41.90	43.40	1.50	6.16
A3-61	45.06	46.06	1.00	1.64
A3-63	40.80	42.80	2.00	2.47
A3-71	141.32	142.81	1.49	8.11
A3-72	57.85	58.35	0.50	31.21
A3-76	79.59	80.10	0.51	7.77
A3-79	48.87	49.12	0.25	10.40
A3-81	98.80	100.49	1.69	47.21
including	98.80	99.26	0.46	158.45
A3-82	152.54	153.90	1.36	5.56
and	167.55	168.05	0.50	5.73
A3-83	44.13	44.63	0.50	4.40
and	100.30	100.77	0.47	14.30
and	107.03	109.03	2.00	11.96

Table 10.5: Highlights of Drill Intercepts from the Phase 1: 2011-2012 Diamond Drill Program				
Hole Number	From (m)	To (m)	Core Length[1] (m)	Au (gpt)
and	117.25	118.25	1.00	5.27
A3-84	98.17	104.40	6.23	7.55
including	98.17	98.67	0.50	29.61
A3-85	149.67	152.00	2.33	12.60
including	151.51	152.00	0.49	24.33
A3-86	114.22	115.72	1.50	7.54
A3-94	51.50	51.75	0.25	94.35
A3-95	47.86	47.99	0.13	48.68
A3-100	147.45	148.35	0.90	46.95
A3-104	51.89	52.14	0.25	8.30
A3-105	146.18	147.68	1.50	11.38
and	149.34	149.74	0.40	5.41
and	150.58	150.95	0.37	6.00
A3-110	111.49	111.89	0.40	4.20
A3-114	83.00	83.75	0.75	7.79
and	95.06	95.28	0.22	9.77
[1]True widths are currently estimated at 85-100% of drilled widths				

10.3.3 Phase 3: 2013 Drill Program

In August 2013, Minnova undertook a small drill program of four (4) holes: A3-115 to A3 118 (Table 10.6 and Figure 10.1). Drilling was targeted to in-fill areas of shallow gold mineralization that were either previously categorised as Inferred mineralization or not categorised due to insufficient drill density. The area of drilling occurred within ML065, the permitted mining lease that was the site of historical production, and in areas previously identified to be potentially amenable to open-pit mining toward the northern end of the existing PL Deposit resource.

Table 10.6: 2013 Drill Hole Location Data						
DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
A3-115	372971	6100833	342.55	-90	0	99.66
A3-116	373048	6100844	343.73	-90	0	154.53
A3-117	373337	6100482	342.37	-90	0	99.66
A3-118	373158	6100863	347.50	-90	0	343.51
Total	**4**					**697.36**

A total of approximately 67 core samples (excluding QA/QC samples) were collected from the 2013 drill core and sent to Accurassay. A list of significant intersections is presented in Table 10.7.

Table 10.7: Highlights of Drill Intercepts from the 2013 PL Drill Program				
Hole	From	To	Core Length[1]	Au

	(m)	(m)	(m)	(gpt)
A3-115	52.12	52.73	0.61	3.43
A3-116	Granodiorite dike at target depth			
A3-117	46.53	47.85	1.32	7.46
A3-118	72.24	75.64	3.39	8.15
[1]True widths are currently estimated at 85-100% of drilled widths				

10.3.4 PHASE 4: 2016 DRILL PROGRAM

In March 2017, Minnova drilled two holes on the mineral lease (Table 10.8 and Figure 10.1). The targets were designed to improve the geological understanding of deeper zones of the deposit, expand the gold resource, and improve confidence-level in the resource.

Table 10.8: 2016 Drill Hole Location Data						
DDH No.	**Easting**	**Northing**	**Elevation (m)**	**Dip (°)**	**Azimuth (°)**	**Length (m)**
M-16-001	373795.5	6101199	354.497	-90	0	562
M-16-002	373593.5	6100835	348.924	-90	0	353
Total	**2**					**915**

Drill hole M-16-001 targeted the deepest extent of the deposit to the northeast. Drill hole M-16-001 did not return any significant gold values although the Sherridon Zone quartz vein mineralization, with associated trace to 2% arsenopyrite, pyrite, and pyrrhotite sulphides, was intersected at the targeted downhole depth between 526-530 m.

Drill hole M-16-002 was collared on the tailings pond of the PL Deposit and was drilled to infill between historical holes with high-grade intercepts. The intent of the hole was to expand the Measured and Indicated Resource in this area. Drill hole M-16-002 intersected the Sherridon, Upper, and Lower Zones including 17.97 g/t Au over 2.4 m in the Lower Zone.

A total of approximately 80 core samples (excluding QA/QC samples) were collected from the 2016 drill core and sent to Accurassay. A list of significant intersections is presented in Table 10.9.

Table 10.9: Highlights of Drill Intercepts from the 2016 PL Drill Program						
Hole	**Zone**	**From (m)**	**To (m)**	**Length (m)**	**True Width[1] (m)**	**Au (gpt)**
M-16-001	Sherridon Zone					NSV
M-16-002	Sherridon Zone	219.6	224.0	4.40	4.16	1.91
	Upper Zone	285.45	286.2	0.75	0.71	6.24
	Lower Zone	323.6	326.0	2.40	2.27	17.97
	including	323.6	324.4	0.80	0.76	44.31
[1]True widths are currently estimated at 85-100% of drilled widths						

10.3.5 Phase 5: 2017 Drill Program

Minnova completed a drill program of forty-eight (48) diamond drill holes (M-17-003 to M-17-050) in the first quarter of 2017 totalling 8,523 m (Table 10.10 and Figure 10.1). The purpose of this in-fill drill program was to test the extent and continuity of gold mineralization in the PL Deposit with the intent to upgrade and expand the deposit. Holes were collared to improve the Measured and Indicated Resources in the northern, higher grade portions of the PL Deposit while others targeted the down dip extension of the deposit below the historical workings. The drill program supported the updated NI 43-101 resource estimate reported in this Report.

Table 10.10: 2017 Drill Hole Location Data

DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
M-17-003	372855	6100392	346.9	-90	0	140
M-17-004	373350	6100640	349.0	-90	0	170
M-17-005	373294	6100626	346.0	-90	0	131
M-17-006	373171	6100677	342.2	-90	0	80
M-17-007	373189	6100630	342.6	-90	0	71
M-17-008	373506	6100795	352.2	-90	0	293
M-17-009	373199	6100615	342.7	-90	0	73
M-17-010	373213	6100601	342.5	-90	0	77
M-17-011	373221	6100572	342.7	-90	0	92
M-17-012	373250	6100577	342.6	-90	0	101
M-17-013	373543	6100727	353.3	-90	0	278
M-17-014	373046	6100670	342.3	-90	0	53
M-17-015	373109	6100701	342.4	-90	0	98
M-17-016	373154	6100704	342.1	-90	0	95
M-17-017	373289	6100607	344.3	-90	0	128
M-17-018	373619	6100716	352.8	-90	0	302
M-17-019	373287	6100653	352.9	-90	0	140
M-17-020	373283	6100679	347.2	-90	0	146
M-17-021	373577	6100753	353.1	-90	0	314
M-17-022	373243	6100653	344.4	-90	0	116
M-17-023	373223	6100695	345.2	-90	0	122
M-17-024	373223	6100722	344.7	-90	0	134
M-17-025	373278	6100714	347.6	-90	0	146
M-17-026	373664	6100820	348.0	-90	0	395
M-17-027	373324	6100730	351.3	-90	0	176
M-17-028	373282	6100735	348.7	-90	0	161
M-17-029	373317	6100708	350.0	-90	0	173
M-17-030	373740	6100581	350.3	-90	0	275
M-17-031	373139	6100762	343.0	-90	0	131
M-17-032	373214	6100789	346.3	-90	0	149
M-17-033	373700	6100599	351.5	-90	0	275
M-17-034	373179	6100838	346.9	-90	0	173

Table 10.10: 2017 Drill Hole Location Data

DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
M-17-035	373394	6100705	350.8	-90	0	203
M-17-036	373762	6100544	349.6	-90	0	275
M-17-037	373384	6100769	346.8	-90	0	215
M-17-038	373783	6100479	349.6	-90	0	236
M-17-039	373327	6100771	344.6	-90	0	188
M-17-040	373751	6100460	348.2	-90	0	194
M-17-041	373372	6100815	346.3	-90	0	227
M-17-042	373681	6100540	353.6	-90	0	239
M-17-043	373694	6100418	349.5	-90	0	185
M-17-044	373429	6100771	347.8	-90	0	236
M-17-045	373839	6100038	349.4	-90	0	200
M-17-046	373449	6100742	347.8	-90	0	221
M-17-047	373228	6100674	343.8	-70	210	110
M-17-048	373092	6101499	347.4	-90	0	152
M-17-049	373118	6100641	342.6	-60	200	60
M-17-050	371929	6100372	347.6	-60	175	374
Total	**48**					**8,523**

A total of approximately 1,464 core samples (excluding QA/QC samples) were collected from the 2017 drill core and sent to Accurassay, SGS and Actlabs. A list of significant intersections is presented in Table 10.11.

Table 10.11: Highlights of Drill Intercepts from the 2017 Drill Program

Hole	From (m)	To (m)	Core Length[1] (m)	Au (gpt)
M-17-004	110.50	112.00	1.50	11.64
including	111.00	112.00	1.00	17.46
M-17-005	77.00	80.00	3.00	7.67
including	79.00	80.00	1.00	17.56
M-17-006	35.25	36.90	1.65	6.05
M-17-007	57.80	59.30	1.50	10.09
including	58.65	59.30	0.65	23.28
M-17-009	51.50	53.28	1.78	13.03
including	53.00	53.28	0.28	57.64
M-17-010	65.50	67.00	1.50	35.57
including	66.00	67.00	1.00	53.36
M-17-011	12.30	15.85	3.55	20.24
including	12.30	12.60	0.30	131.29
including	13.65	15.85	2.20	14.09
and	61.85	63.35	1.50	6.15
including	62.40	63.35	0.95	9.71
M-17-012	29.00	32.00	3.00	11.10

Table 10.11: Highlights of Drill Intercepts from the 2017 Drill Program				
Hole	From (m)	To (m)	Core Length[1] (m)	Au (gpt)
including	30.00	31.00	1.00	21.23
and	65.00	66.50	1.50	18.98
including	65.50	66.50	1.00	27.99
and	78.00	80.00	2.00	6.07
including	79.00	80.00	1.00	8.62
M-17-014	45.55	47.05	1.50	59.98
including	38.10	38.47	0.37	11.03
including	46.35	46.65	0.30	298.05
M-17-016	57.50	59.00	1.50	14.17
including	58.00	59.00	1.00	21.25
and	84.00	85.50	1.50	10.97
including	84.50	85.50	1.00	16.19
M-17-017	4.50	6.00	1.50	6.25
including	5.00	6.00	1.00	9.37
and	63.00	68.00	5.00	16.38
including	63.00	64.00	1.00	23.79
including	65.00	66.00	1.00	15.76
including	67.00	68.00	1.00	23.28
M-17-018	165.50	167.00	1.50	6.57
including	166.00	167.00	1.00	9.85
M-17-019	89.22	92.00	2.78	37.45
including	89.22	90.00	0.78	81.9
including	90.00	91.00	1.00	32.53
and	122.00	125.00	3.00	9.62
including	124.00	125.00	1.00	21.53
M-17-020	127.50	130.00	2.50	14.03
including	127.50	129.00	1.50	12.62
including	129.00	130.00	1.00	16.14
M-17-023	78.00	79.50	1.50	6.76
and	110.00	112.00	2.00	38.01
including	110.00	111.04	1.04	70.35
M-17-026	345.00	346.95	1.95	12.41
including	346.00	346.95	0.95	21.69
M-17-027	163.07	164.57	1.50	23.50
including	163.07	164.00	0.93	37.9
M-17-033	135.00	137.00	2.00	10.27
M-17-037	151.50	153.00	1.50	8.47
M-17-039	156.00	158.00	2.00	7.46
M-17-044	133.50	135.00	1.50	6.87
including	134	135	1	10.3
[1]True widths are currently estimated at 85-100% of drilled widths				

10.4 NOKOMIS PROPERTY DRILLING

10.4.1 2012 DRILL PROGRAM

In February 2012, Minnova commenced a nineteen (19) hole (A4-01 to 19) 1,335 m diamond drill program. The aim of the program was to confirm historical mineralized intervals at Nokomis. The program was completed in March 2012 and intersected several high grade mineralized zones. Drill holes A4-01 to A4-07, A4-09, and A4-12 intersected gold mineralization associated with the Upper Host Zone. Drill holes A4-01 to A4-07, A4-09, and A4-12 intersected gold mineralization associated with the Lower Host Zone.

A list of drill holes and significant intersections is presented in Table 10.12 and Table 10.13. Drill hole locations are presented in Figure 10.2. The results of the 2012 drill program are used for the Nokomis Resource Estimate prepared by ACA Howe, presented in Section 14.3.

Table 10.12: 2012 Nokomis Drill Hole Location Data

DDH No.	Easting	Northing	Elevation (m)	Dip (°)	Azimuth (°)	Length (m)
A4-01	380144	6104099	320	-90	360	69.00
A4-02	380144	6104099	320	-60	300	59.43
A4-03	380076	6104066	320	-90	360	41.15
A4-04	380104	6104054	320	-90	360	50.29
A4-05	380009	6103913	320	-85	300	88.39
A4-06	380009	6103934	320	-90	360	54.86
A4-07	380040	6103930	320	-90	360	67.05
A4-08	380049	6103982	320	-90	360	25.91
A4-09	380079	6103966	320	-90	360	45.72
A4-10	380091	6103989	320	-90	360	54.86
A4-11	380156	6104073	320	-85	300	80.77
A4-12	380156	6104073	320	-50	300	70.10
A4-13	380149	6104168	321	-86.7	286	42.67
A4-14	380142	6104134	320	-90	360	48.77
A4-15	380156	6104178	321	-90	360	60.96
A4-16	379997	6103960	320	-90	360	32.00
A4-17	380020	6103979	320	-90	360	31.47
A4-18	380220	6103925	320	-90	360	210.30
A4-19	380235	6103998	320	-90	360	201.16
Total	**19**					**1,334.86**

Source: www.minnova.ca

Table 10.13: Significant Drill Intercepts from the 2012 Nokomis Drill Program				
Hole	**From (m)**	**To (m)**	**Core Length[1] (m)**	**Au (gpt)**
A4-01	51.79	59.06	7.27	1.91
A4-02	39.85	47.48	7.63	125.08
Including	45.35	46.48	1.13	820.28
including	45.35	45.85	0.50	1,830.27
A4-03	8.74	13.94	5.20	12.27
A4-04	31.20	35.81	4.61	2.95
A4-05	No Significant Assays			
A4-06	35.79	38.54	2.75	1.12
A4-07	53.42	56.11	2.69	1.04
A4-09	27.14	28.14	1.00	0.94
A4-10	39.06	40.12	1.06	5.46
A4-11	56.76	63.19	6.43	5.10
A4-12	48.81	54.36	5.55	9.65
including	50.31	52.84	2.53	18.85
including	51.81	52.34	0.53	62.23
A4-14	35.39	41.39	6.00	2.79
A4-15	44.05	46.05	2.00	1.81
A4-16	25.29	26.75	1.46	4.89
A4-17	19.81	28.29	8.48	1.18
A4-18	166.95	167.95	1.00	13.72
and	171.80	172.80	1.00	11.64
A4-19	149.58	150.58	1.00	8.25
[1]True widths are currently estimated at 85-100% of drilled widths				

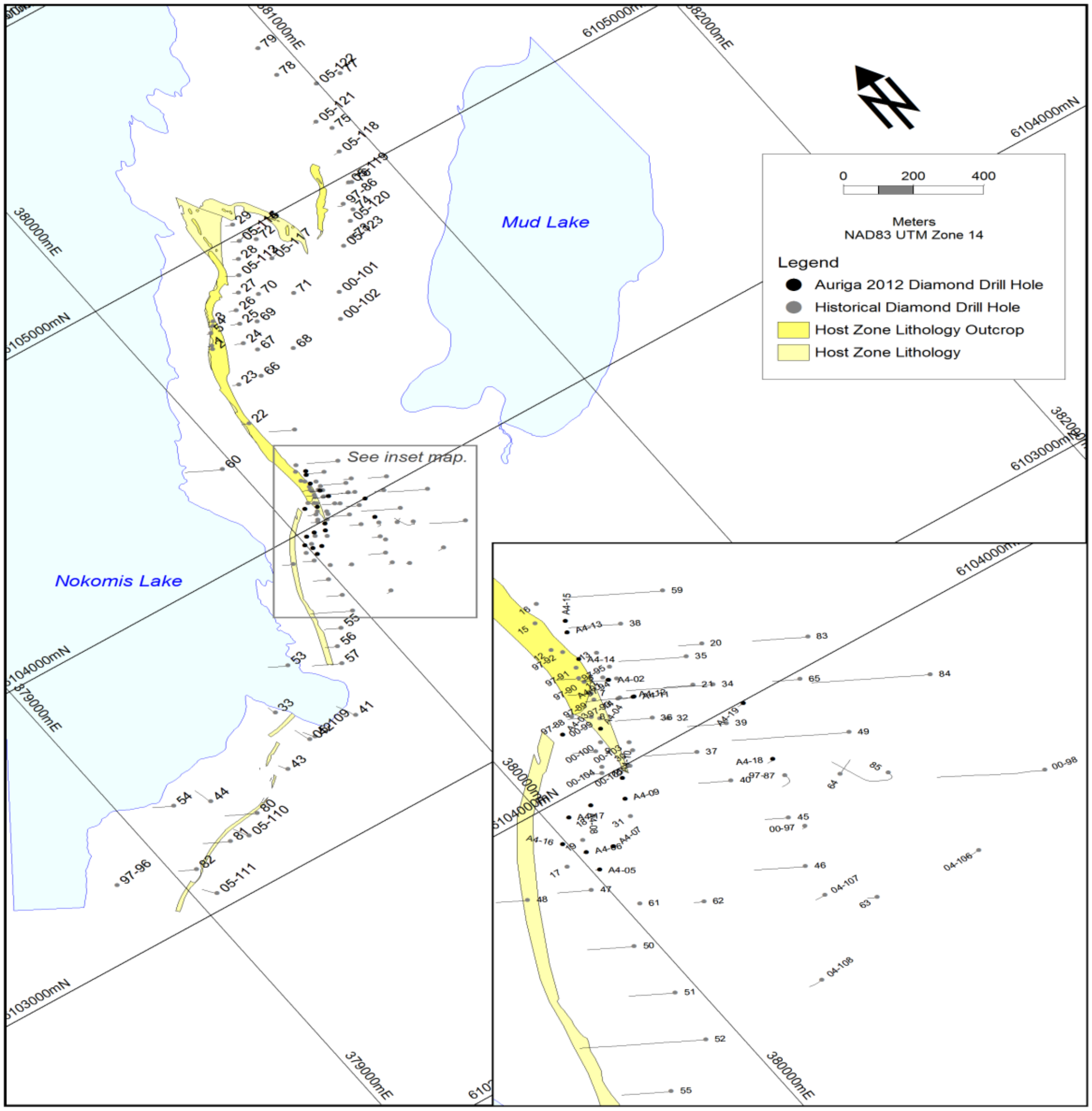


Figure 10.2 Nokomis Property drill hill locations

CSA Global did not find any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

10.5 MINNOVA DIAMOND DRILL PROGRAM METHODS AND APPROACH

10.5.1 DIAMOND DRILL PROGRAMS – GENERAL

Minnova personnel and consultants/contractors designed and supervised the drill programs. The 2011-2013 diamond drill programs, conducted on the PL and Nokomis Properties, were completed under the supervision of Mr. Bruce Mackie. The 2016 and 2017 programs were completed under the supervision of Mr. Chris Buchanan, M.Sc., P. Geo, Minnova Senior Geologist. All 2011 to 2017 diamond drill programs have been contracted to Top Rank Diamond Drilling Ltd. ("Top Rank") of Ste Rose du Lac, Manitoba. Drilling was conducted with 1-2 diamond drills, 7-days a week, 24-hours a day. The drill holes were completed using NTW rods (core diameter of 50 mm). Drill hole lengths varied from 8.4-450 m.

Top Rank constructed drill access trails and drill pads for each setup. Water was supplied by pump and hose from nearby lakes and ponds.

At the completion of the drill hole, casing and rods were removed from the holes.

10.5.2 Drill Hole Survey Methods

Drill holes were initially spotted using a hand-held GPS device and the hole azimuth determined with a Silva or Brunton compass. Upon completion of drill holes, Balchen and Kulchycki Surveys of Dauphin, Manitoba surveyed the drill hole collar coordinates and elevations in UTM coordinates (NAD83).

The drill contractor completed downhole directional surveys on all diamond drill holes at approximately 50 m intervals using a Reflex EZ Shot single shot or similar digital survey tool.

The QP is of the opinion that Minnova's drill hole survey methods meet industry and NI 43-101 standards.

10.5.3 Core Handling, Logging, and Sampling Methods

Core is retrieved from the drill string using conventional wireline techniques. Sample security and chain of custody starts with the removal of core from the core tube and boxing of drill core at each drill. Core is removed from the core tube by the drill contractor's personnel, carefully placed in labeled wooden core boxes, and localized by inserted depth blocks. The boxed core remains under the custody of the drillers until it is transported from the drill to Minnova's secure core processing and sampling facility at the PL Mine site by either the drill contractor or one of the Company's designated personnel.

The core logging and processing facility is located at the PL Mine site near the ramp portal. The facility comprises a portable, secure core logging building used for logging and packing samples for shipment to the assay laboratory.

At the core facility, core boxes are opened and inspected to ensure correct boxing and labeling of the core by the drill contractors then re-closed. Remedial actions are undertaken, if necessary, to correct deficiencies in the spatial information prior to entry into a database.

Processing of the core starts with the core being laid out on workbenches and cleaned prior to logging and sample interval marking. Spatial information related to each box of core is checked for accuracy and consistency at this point. Remedial actions are undertaken, if necessary, to correct deficiencies in the spatial information prior to entry into a database. A geotechnical log of core recovery and RQD measurements is completed by a geological technician under the supervision of a Minnova contract geologist. Core recovery is generally very good. There appear to be no sampling or recovery factors that would negatively impact the sampling procedures.

A Minnova contract geologist then completes a descriptive log. This descriptive log is a detailed description of rock type (maintaining consistency with previous work), structure, alteration, and mineralization (including presence of VG, quartz veining, and its angle with the axis of the core).

The core is next photographed with a high-resolution camera, capturing a JPEG formatted image. This high-resolution photography produces a digital image with a file resolution of sufficient quality to enable re-logging on a monitor at greater than actual size. Up to 4 boxes of core are photographed at the same time.

Following photography, the geologist selects the sample intervals and inputs the intervals into the drill hole database. The selected portions of core are marked and measured for sampling and are identified with one part of a three-part assay tag, placed at the top of the sample interval. Sampling lengths ranged from 0.14-2.8 m and averaged 1.0 m. Sample lengths varied according to geology and mineralization with quartz veins regularly sampled. Sampling is generally restricted to quartz veined/mineralized zones and their unmineralized shoulders. Approximately 17% of the drill core was submitted for assaying. The core not sampled is archived with the sampled half-core.

Drill core and sample information are input into a digital database using portable computer workstations at the workbenches.

The core is sawn with a water-cooled core saw with a 14-inch diamond blade and a mounted jig to assure the core is split equally. The core saw is located in a building separate from the core logging facility. Fresh water is used as a cooling/lubricating fluid; recycled water is not used.

The core is cut in half longitudinally, perpendicular to the foliation (50% split) with one-half placed into plastic sample bags along with part two of the three-part assay tag and sealed. The other half core is returned to the core box for archive and future verification and testing (if required). Each sample bag has the sample number written on the outside of the bag with black permanent marker corresponding to the sample tag placed inside. Information on the third part of the assay tag is entered into the database and the drill log, at which time accuracy and consistency are again reviewed and remedied, if necessary.

Core logging, sawing, sample bagging, and sample shipment preparation is completed either by or under the onsite supervision of a Minnova contract geologist. Certified reference materials (standards), sample blanks, and duplicate samples are inserted by Minnova into each sample batch submitted to the lab for the purpose of quality control (¼ core duplicates were submitted beginning in 2017).

After sampling is completed, the archived core boxes are labeled and cross-piled at the PL Mine site in the ramp portal area.

Sealed sample bags are placed in rice sacks and secured. Minnova personnel maintain possession of the samples in the secure core shack until delivery to the commercial shipping company, Gardwine General Freight. Sample batches are transported to the analytical laboratory in a timely fashion by the shipping company and transferred to the laboratory's chain of custody procedures and protocols.

Following analysis, digital assay files, provided by the laboratory, are merged with a "from" and "to" interval file created by Minnova, with the sample number linking the two files. This methodology limits data entry errors to sample numbering.

Overall, sampling methods are to industry standards for mineralization of this type. The QP is of the opinion that the sampling methods meet NI 43-101 standards.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 HISTORICAL DRILL PROGRAMS

Detailed descriptions of sample preparation, analytical methods, and security protocols and procedures utilized by previous operators prior to 2010 presented in Section 6 – History of this Report were not available to the Authors.

Furthermore, for the majority of previous operators, the Authors were unable to confirm certification of the assay labs used nor their relationship to the previous operators for assay results disclosed in Section 6 – History. Unless specified below, CSA Global has not been able to determine whether the sample preparation and analytical techniques employed by those companies were appropriate for the sample media and mineralization type and conform to past and current industry standards. However, it is probable that the techniques were completed to industry standards considered appropriate at the time of completion.

11.1.1 PL DEPOSIT

Sample information is compiled from assay summary sheets within assessment file compilations for Maverick Mountain drill holes PUF-001 to PUF-156 completed between 1981 and 1986. Assays were analysed by fire assay and metallic screen fire assay techniques.

Laboratory assay certificates are available for a small number of holes: PUF-110 and PUF-141 to 147 analysed at Eco-Tech Laboratories Ltd. Kamloops, British Colombia ("Eco-Tech"); and PUF-117 to 120 analysed at Chemex, Vancouver, British Colombia ("Chemex").

No description of sample preparation and analyses techniques is available for holes completed by Pioneer Metals Corp.

11.1.2 NOKOMIS DEPOSIT

Samples collected by Claude and Pioneer in 2004 and 2005 were prepared and assayed for gold at TSL Laboratories ("TSL") by fire assay and gravimetric methods. At the time of analysis, TSL held certificates of accreditation with the Standards Council of Canada for the procedures used. The reader is directed to Buhlmann (2006) for more information.

Samples collected by Pioneer in 2000 were assayed for gold by fire assay with an ICP finish at the Geoanalytical Services Laboratories of the Saskatchewan Research Council (Buhlmann, 2000).

Samples collected by Pioneer in 1997 were assayed for gold at Eco-Tech by fire assay with an ICP finish (Buhlmann, 2000).

The 25 Dome diamond drill holes completed in 1985 were analysed by fire assay at Pauls' Custom Fire Assaying Limited, Cochenour, Ontario. For samples that returned grades greater than 2 g/t Au, the original pulp and reject material were submitted for reanalysis at the Campbell Red Lake mine, Balmertown, Ontario. (L. MacCormack, 1985).

No description of assay technique is available for earlier drilling, including 14 drill holes completed by Dome in 1975, 41 Rio Tinto diamond drill holes completed in 1961, and 5 Parres diamond drill holes completed in 1958.

11.2 ACCURASSAY LABORATORIES 2010-2017

Accurassay Laboratories ("Accurassay") in Thunder Bay, Ontario was Minnova's primary analytical laboratory from 2010 to early 2017 when it went out of business. Accurassay was accredited to international quality standards through the International Organization for Standardization/International Electrotechnical Commission to ISO/IEC 17025/2005. It was a Standards Council of Canada Accredited Laboratory (No. 434) and conformed to requirements of CAN-P-1579 (Mineral Analysis) and CAN-P-4E.

Accurassay and its employees were independent from Minnova. Minnova personnel and consultants and contractors are not involved in sample preparation and analysis.

11.2.1 SAMPLE PREPARATION

Upon arrival at the Accurassay facilities, samples were entered into Accurassay's Local Information System (LIMS). Primary core samples received at Accurassay were prepared using sample preparation package ALP1, which consisted of conventional drying, if required, in 105°C ovens; crushing, splitting, and pulverizing. After drying, the <5 kg sample was passed through a primary oscillating jaw crusher producing material of 85% passing a 2 mm (-10 mesh) screen. A 500-gram sub-sample was split from the crushed material using a stainless steel riffle splitter. This split was then ground to 90% passing 74 microns (-200 mesh) or better using a ring pulveriser. A silica abrasive was used to clean between each sample.

11.2.2 ANALYTICAL PROCEDURES

Primary core samples were prepared and assayed for Au using the lead fire assay procedure with an AAS geochemical finish (Accurassay Code ALFA2 – 50 g charge). The lower and upper detection limits were 0.005 and 30 g/t Au, respectively. All samples that assayed higher than 2.5 g/t were re-assayed for Au and Ag using a gravimetric finish (Accurassay Code ALFA8 – 50 g charge) to improve accuracy of the higher-grade samples. Accurassay's lower and upper detection limits for reporting 50 g charge fire assay-gravimetric results were 1 and 1,000 g/t Au and 10 and 10,000 g/t Ag, respectively.

Accurassay also had its own internal QA/QC protocols, including standards, blanks, and duplicates and the results of these analyses were also reported along with the results on Minnova's samples.

Laboratory pulps and rejects from the 2017 diamond drill program were backhauled to the Property and stored at the mill.

11.3 SGS LABORATORIES 2017

Due to the unforeseen closure of Accurassay Laboratories in early 2017, samples were re-routed to SGS Canada Inc. ("SGS") Mineral Services and their labs in Burnaby, British Columbia and Cochrane, Ontario. SGS is an accredited laboratory (PTP-MAL – Accredited by Standards Council of Canada – in accordance with ISO/IEC 43-1).

SGS Cochrane is a Standards Council of Canada Accredited Laboratory No. 841 and conforms with requirements of CAN-P-4E (ISO/IEC 17025:2005): General Requirements for the Competence of Testing and Calibration Laboratories (ISO/IEC 17025: 2005) and CAN-P-1579:2014: Requirements for the Accreditation of Mineral Analysis Testing Laboratories.

SGS Burnaby is a Standards Council of Canada Accredited Laboratory No. 744 and conforms with requirements of CAN-P-4E (ISO/IEC 17025:2005): General Requirements for the Competence of Testing and Calibration Laboratories (ISO/IEC 17025: 2005) and CAN-P-1579:2014: Requirements for the Accreditation of Mineral Analysis Testing Laboratories. The Burnaby lab is accredited for the gold fire assay procedures used on Minnova's samples:

- GE_FAI313-FAI515 Determination of Gold, Platinum and Palladium by Lead Fusion Fire Assay and Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) [Au; Pt; Pd; HNO_3; HCl]
- GO_FAG303-FAG505 - Determination of Ore Grade Gold by Lead Fusion Fire Assay and Gravimetric Finish [30g.; 50g.; Au; HNO_3; NH_4OH]

SGS labs utilize industry standard quality control procedures.

SGS Mineral Services and its employees are independent from Minnova. Minnova personnel and consultants and contractors are not involved in sample preparation and analysis.

11.3.1 SAMPLE PREPARATION

Primary core samples received at SGS are prepared using its sample preparation package PRP89, which consists of conventional drying, if required, in 105°C ovens; crushing, splitting, and pulverizing. After drying, the sample is passed through a primary oscillating jaw crusher producing material of 75% passing a 2 mm screen. A 250-gram sub-sample is split from the crushed material using a riffle splitter. This split is then ground to 85% passing 75 microns or better using a ring pulveriser.

11.3.2 ANALYTICAL PROCEDURES

Primary core samples are prepared and assayed using the lead fire assay procedure on a 30 g sample with an AA geochemical finish (SGS Code GE FAA313). The lower and upper detection limit of method GE FAA313 is 0.005 and 10 g/t Au. All samples that assay higher than 3.0 g/tonne are re-assayed using a 30 g sample with a gravimetric finish (SGS Code GE FA303) to improve accuracy of the higher grade samples. The lower and upper detection limit of method GE FAG303 is 0.5 and 3,000 g/ Au.

SGS also has its own internal QA/QC protocols including standards, blanks, and duplicates and the results of these analyses are also reported along with the results on Minnovas's samples.

Laboratory pulps and rejects from the 2017 diamond drill program were backhauled to the Property and stored at the mill.

11.4 ACTIVATION LABORATORIES 2017

Due to laboratory backlogs at SGS, Minnova shifted to Activation Laboratories Ltd. ("Actlabs") at Thunder Bay, Ontario during the latter half of the 2017 diamond drill program and summer field sampling program.

Actlabs is a Standards Council of Canada Accredited Laboratory No. 673 and conforms to requirements of CAN-P-4E (ISO/IEC 17025:2005): General Requirements for the Competence of Testing and Calibration Laboratories (ISO/IEC 17025: 2005) and CAN-P-1579:2014: Requirements for the Accreditation of Mineral Analysis Testing Laboratories. The Thunder Bay lab is accredited for the gold fire assay procedures used on Minnova's samples:

- QOP AA-Au Procedure for analysis of Gold and/or Silver by Fire Assay with AA or Gravimetric Finish

Actlabs utilize industry standard quality control procedures.

Actlabs and its employees are independent from Minnova. Minnova personnel and consultants and contractors are not involved in sample preparation and analysis.

11.4.1 Sample Preparation

Primary core samples received at Actlabs are prepared using its sample preparation package RX1, which consists of crushing, splitting, and pulverizing. A <7 kg sample is passed through a primary jaw crusher producing material of 80% passing a 2 mm screen. A 250 g sub-sample is split from the crushed material using a riffle splitter. This split is then ground to 95% passing 105 microns or better using a mild steel ring pulveriser.

11.4.2 Analytical Procedures

Primary core samples are prepared and assayed using the lead fire assay procedure on a 30 g sample with an AA geochemical finish (Actlabs Code 1A2-30). The lower and upper detection limit of method 1A2 is 0.005 and 5 g/tonne Au. All samples that assay higher than 5.0 g/tonne are re-assayed using a 30 g sample with a gravimetric finish (Actlabs Code 1A3-30) to improve accuracy of the higher grade samples. Actlabs' lower and upper detection limits for reporting 30 g charge fire assay-gravimetric results are 0.03 and 10,000 g/tonne Au and 3 and 10,000 g/tonne Ag, respectively.

Actlabs also has its own internal QA/QC protocols including standards, blanks, and duplicates and the results of these analyses are also reported along with the results on Minnova's samples.

Laboratory pulps and rejects from the 2017 diamond drill program were backhauled to the Property and stored at the mill.

11.5 QUALITY ASSURANCE/QUALITY CONTROL ("QA/QC")

The following section describes the various QA/QC practices employed by Minnova/Auriga and previous operators at the Maverick Project. The availability of QA/QC data is described below for the following campaigns:

- 1,806 Minnova samples analysed by Accurassay, Actlabs, and SGS account for 6% of sample data for the Maverick Property provided to the Author. Certified Reference Materials ("CRM") blanks and duplicates amount to 19% of these samples.

- 4,653 Auriga samples analysed at Accurassay account for 16% of sample data for the Maverick Property provided to the Author. CRMs and blanks amount to 7% of these samples

No independent QA/QC data is available for analyses of samples collected from the Maverick Project prior to 2011 and which account for 78% of available sample data. At the PL Deposit, historical drill programs include the Maverick Mountain and Pioneer exploration campaigns. At Nokomis, historical drill programs include Pioneer, Dome, Rio Tinto, and Parres diamond exploration campaigns.

11.5.1 MINNOVA 2017

Minnova implemented a QA/QC program for sampling undertaken in 2017. In addition to quality control procedures employed by the laboratories, a series of duplicate samples, blank material and CRMs were added to the sample stream. A ratio of one in every eight samples sent for analysis at the SGS and Actlab laboratories was a QA/QC sample.

11.5.1.1 Performance of Certified Reference Materials

To monitor accuracy, CRMs were inserted sequentially into the sample stream before shipment from the field at a rate of one in every 20 samples submitted. A range of gold and silver CRMs were obtained from CDN Resource Laboratories Ltd. of Langley, British Columbia, Canada. CRMs were received in individually vacuum sealed tin-top kraft bags containing 60 g of pulverized blended material.

To check the accuracy of the 2017 laboratory results, control limits were established above and below the certified mean at the certified ± 2 standard deviation value. CRM results are plotted in sequence with control limits and the certified CRM value for gold along with error bars that represent ± 1 standard deviation for CRM results returned by the laboratory. CRMs and the assay results they returned are summarized in Table 11.1.

Table 11.1: Minnova CRM Data									
Standard	**Type**	**Control Grade**	**Cert Std. Dev**	**Count**	**Mean**	**Max**	**Min**	**Lab Std. Dev**	**Average % Diff**
CDN-GS-4B	Gold Accua	3.77	0.35	1	3.95	3.95	3.95	-	
CDN-GS-4B	Gold SGS	3.77	0.35	26	3.92	4.60	2.42	0.22	5%
CDN-GS-7E	Gold SGS	7.40	0.57	23	7.68	8.61	6.78	0.42	4%
CDN-GS-13A	Gold SGS	13.2	0.72	9	13.40	13.70	12.70	0.30	1%
CDN-GS-4B	Gold Actlab	3.77	0.35	14	3.86	4.03	3.65	0.13	2%
CDN-GS-7E	Gold Actlab	7.40	0.57	17	7.49	7.77	7.20	0.20	1%
CDN-GS-13A	Gold Actlab	13.2	0.72	7	13.26	14.00	12.40	0.61	0%

Control plots for gold CRMs are presented in Figure 11.1. Most CRM analyses returned values within the accepted ± two standard deviation control limit.

SGS analysis results for CRM CDN-GS-4B were predominantly within control limits. Two samples were beyond limits included one outlier, sample 268182, which returned a value of 2.42 g/t Au, significantly lower than the expected CRM grade of 3.77 g/t Au. Ignoring outliers, on average results, showed a slight positive bias of 0.5% relative to the control grade. SGS analysis results for CRMs CDN-GS-7E and CDN-GS-13A returned values that were predominantly within control limits.

Actlab analysis results for all CRMs showed good accuracy and precision, with 95% of CRM results falling within the ± 1 standard deviation and no significant bias to under or over reporting of gold grades.

For both laboratories, average CRM gold results and the deviations identified, are within the certified confidence limits for each CRM.

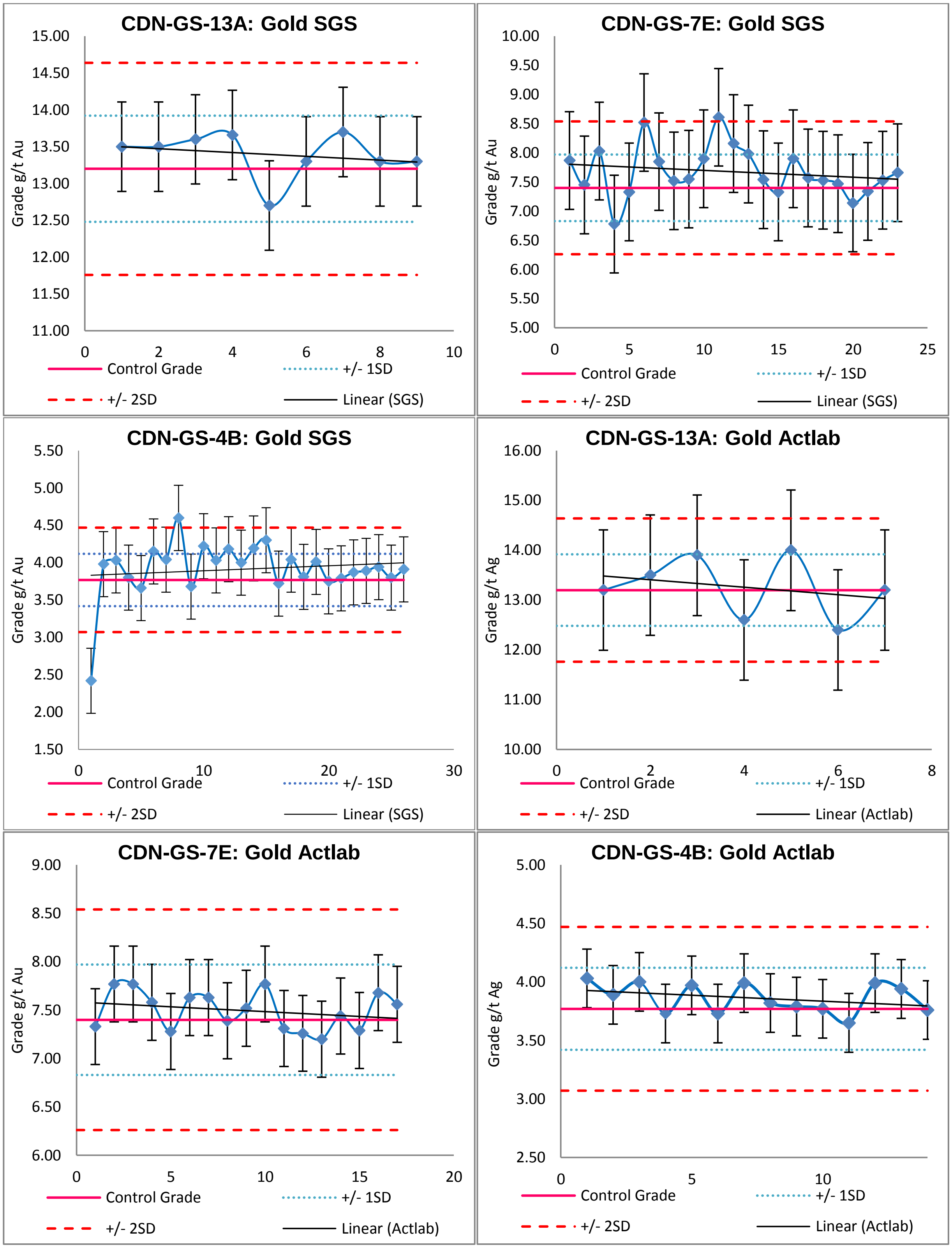


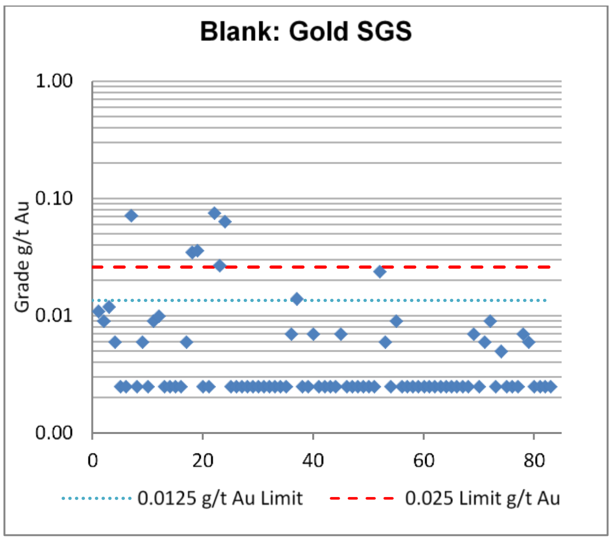
Figure 11.1 Control plots for 2017 Gold CRMs

11.5.1.2 Performance of Blanks Material

Minnova inserted 135 blank samples into the sample stream before shipment from the field a rate of approximately one in every 20 samples. In 2016, blank material was black silica sand. In 2017, blank material was changed to 2-3 cm fragments of marble that underwent the same preparation procedure as core fragments.

As a rough guide, the Author suggests that blank samples should have analyses of less than five times the detection limit; the maximum acceptable value for the blank material be 25 ppb or 0.025 g/t Au. A blank sample that assays greater than the maximum acceptable value should be considered a failure.

Blanks sample analysis results for SGS and Actlabs are shown in Figure 11.2.



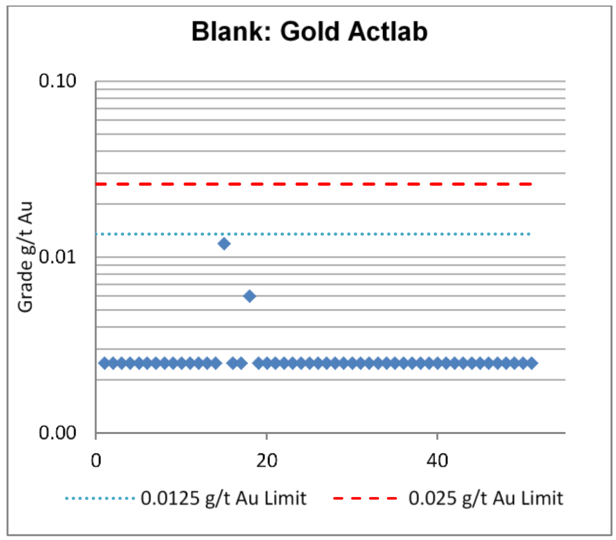


Figure 11.2 Control plots for 2017 blank samples

Early in the 2017 sampling program, six blank analysed at SGS returned elevated gold grades beyond the 0.025 g/t Au limit. In each case, a sample with a high gold value preceded an anomalous blank. For example, blank sample 268255 returned a value of 0.075 g/t Au and was immediately preceded by samples of 14 g/t Au, 1 g/t Au, and 131 g/t Au. Although this indicates possible smearing of gold during sample preparation, the level of contamination is not considered significant. Blank samples returned an average grade of 0.01 g/t Au and no blank samples returned a greater than 0.1 g/t Au.

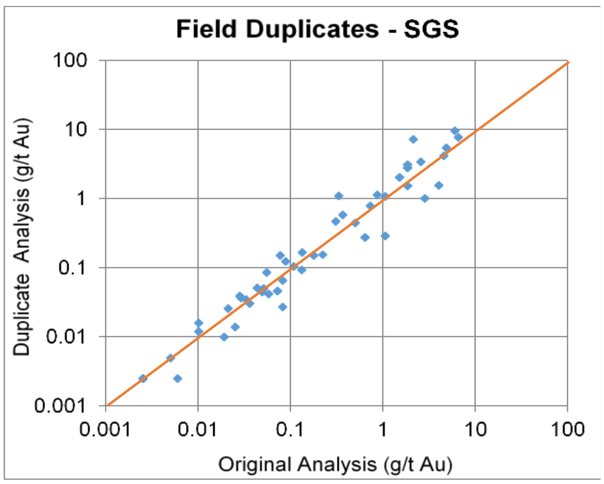
Blanks sample analysis results for Actlabs were very good. All but two samples returned grades at the lower limit of detection.

Overall blanks sample analysis results are considered acceptable.

11.5.1.3 Performance of Duplicates

The 2017 drill program included quarter core duplicate samples that were used to assess the presence of a 'nugget effect' or heterogeneity of gold mineralization in drill core samples. Minnova inserted one hundred twelve (112) quarter core duplicates into the sample stream such that one in every twenty (20) samples was a duplicate.

The difference between the original analysis and the quarter core duplicate analysis is presented in the scatter plots in Figure 11.3. Any values that plot significantly away from the scatter chart correlation line may indicate a potential nugget effect or sample handling errors. Control charts show a reasonable correlation between original and quarter core duplicates.



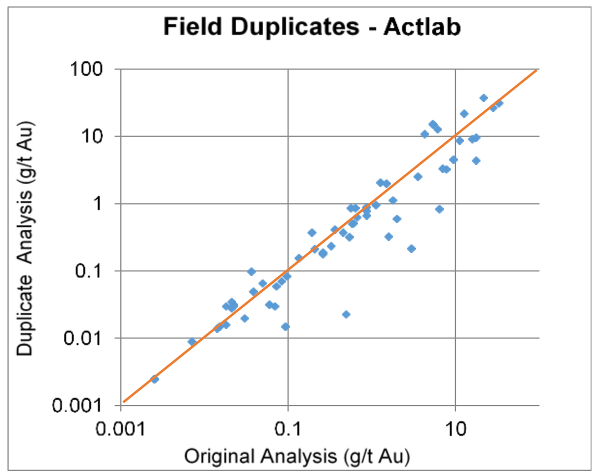


Figure 11.3 Scatter plots for 2017 core duplicate samples

11.5.2 Auriga (Now Minnova) 2010-2013

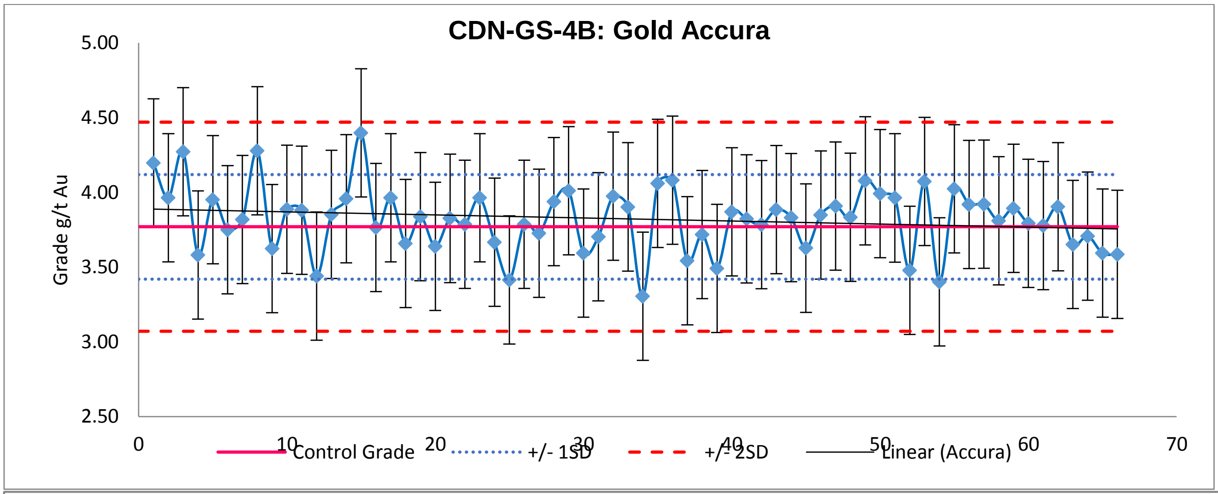
Auriga implemented a QA/QC program for the drilling completed between 2011 and 2013. In addition to the standard quality control of the laboratories, a series of blank and certified reference materials were added to the sample stream. A ratio of one in every twenty (20) samples sent for analysis at the Accurassay Laboratory was a QA/QC sample.

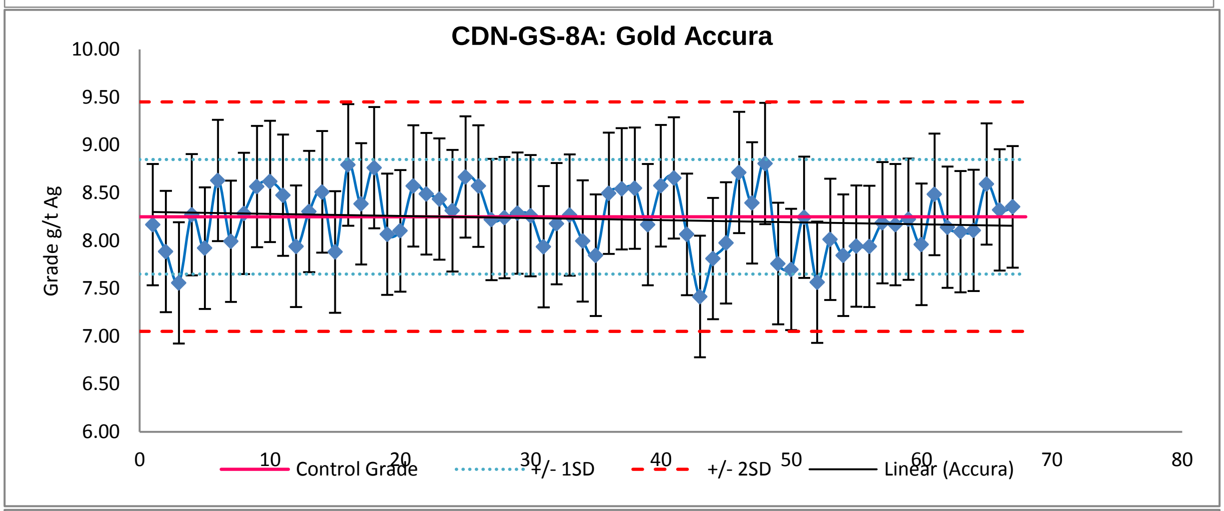
11.5.2.1 Performance of Certified Reference Materials

Three CRMs purchased at CDN Resource Laboratories were used for sampling programs completed between 2010 and 2013. CRM grades ranged from 3.77 to 13.2 g/t Au. CRMs were received in individually vacuum sealed tin-top kraft bags containing 60 g of pulverized blended material. CRMs and their assay results are summarized in Table 11.2.

Table 11.2: Auriga CRM Data									
Standard	**Type**	**Control Grade**	**Cert Std. Dev**	**Count**	**Mean**	**Max**	**Min**	**Lab Std. Dev**	**Average % Diff**
CDN-GS-4B	Gold SGS	3.77	0.35	66	3.82	4.40	3.31	0.21	1%
CDN-GS-13A	Gold SGS	13.2	0.72	65	13.36	14.72	12.52	0.38	1%
CDN-GS-7E	Gold SGS	8.25	0.6	68	8.23	8.81	7.42	0.32	0%

Control plots for gold CRMs are presented in Figure 11.4. Most CRMs analytical values are within the accepted ± 2 standard deviation control limits. Samples plot evenly about the expected control grade with no significant bias. For each CRM type analysed at Accurassay, average gold results and the deviations identified, are within confidence limits.





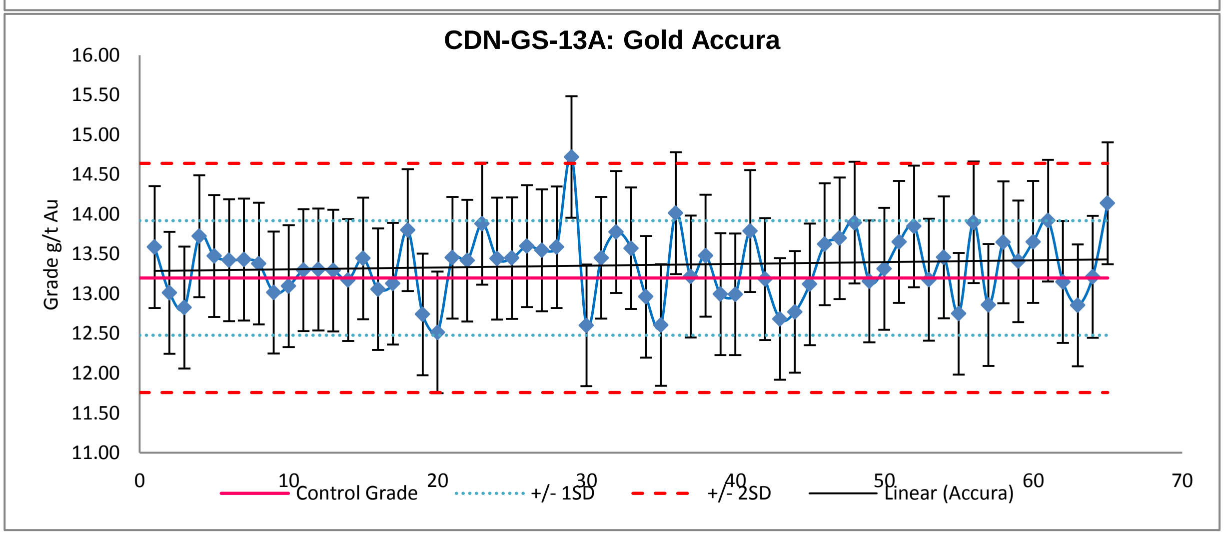


Figure 11.4 Control Plots for Auriga Gold CRMs – 2011 to 2012

11.5.2.2 Performance of Blank Material

Auriga inserted one hundred thirty-three (133) blank samples into the drill core sample batches before shipment at a rate of approximately one in every thirty (30) samples.

The blank material used initially was a "barren" gneissic rock that went through the same sample preparation method as core samples. The initial twelve (12) samples returned elevated gold values (the highest being 1.03 g/t Au); Auriga recognized that the gneiss was not completely barren of gold and discontinued its use. At that time, Auriga switched to a pre-pulverized barren CRM for the remainder of the program. Because the barren CRM did not go through the sample preparation process, it monitored possible analytical contamination only. It is not known when the use of the barren CRM was discontinued (Figure 11.5).

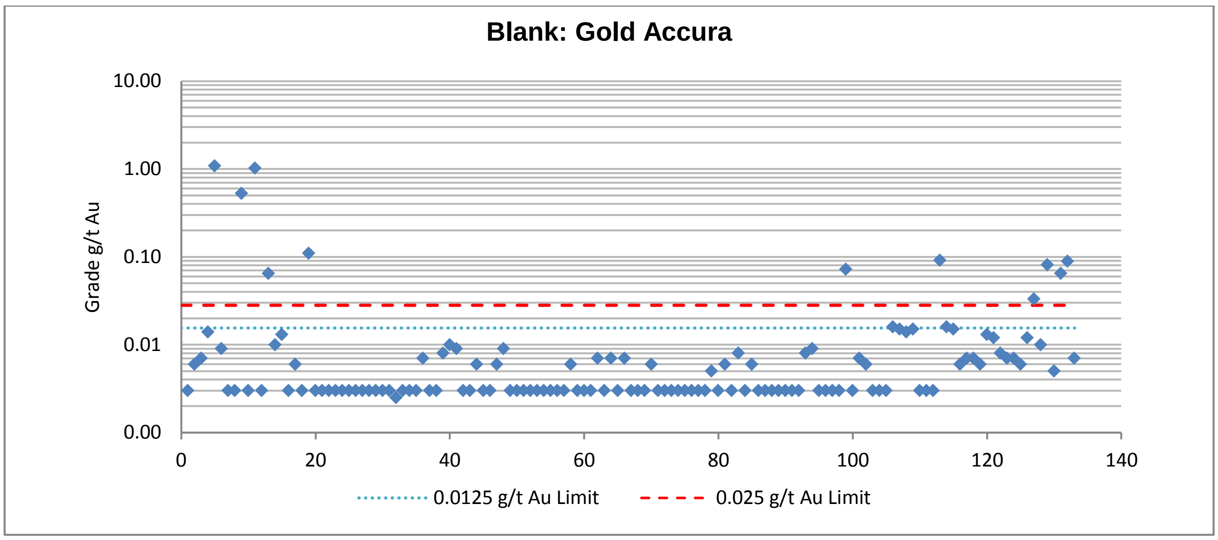


Figure 11.5 Control Plots for Auriga Blank Samples – 2011 to 2012

11.5.3 Historical

No independent Maverick Project QA/QC data, prior to 2011, was made available to the Author. However, the QA/QC assessment for a selection of historical drill programs are presented in technical reports on the property.

11.5.3.1 PL Deposit

In 2011, P&E undertook a comparison of coarse reject duplicate analyses that were undertaken during historical sampling programs:

> *"P&E compiled 1,674 reject split duplicate analyses from Eco-Tech, and Chemex that were completed during the historical programs. Only values greater than 1 g/t Au were compiled. Considering the duplicate pairs were derived from a coarse reject split, the*

resulting precision of 11%, when measured using the Thompson-Howarth precision plot is considered excellent.

P&E declares the data acquired and analyzed by Auriga Gold to be satisfactory for use in a resource estimate."

11.5.3.2 Nokomis

At Nokomis, the 2004 program was conducted under the supervision of E. Buhlmann, P.Geo. and the 2005 program under J. O'Sullivan, P.Eng. of Rainbow Resources Limited; both are qualified persons under NI 43-101. The results of the QA/QC program employed at that time are discussed in the 2006 technical report on the property, as follows:

> *"In all, 30 duplicates and 11 standards were inserted into the sample stream. The highest difference between sample pairs was 18% (0.41 g/t and 0.48 g/t gold) and the standard deviation for the standards was 4% (0.17 g/t; standard AuM-2 gold content: 4.94 g/t gold). The assay results for all samples were reviewed by the author. Based on (i) the results for duplicates and standards, (ii) the comparison of sample lengths and visual descriptions of mineralization in drill logs, (iii) the author's experience with the tenors of mineralization in outcrops and drill cores from several field and drilling programs on the Property, and (iv) the author's past experience with numerous sample assays and check assays from the property, the author formed the opinion that the assay results as reported can be relied on for the purpose of evaluating the exploration potential and recommending the future exploration"* (Buhlmann, E., 2006).

11.5.4 Quality Assurance/Quality Control Conclusions

CSA declares the data acquired and analyzed by Minnova and its predecessor, Auriga Gold, to be satisfactory for use in a resource estimate.

12 DATA VERIFICATION

12.1 PL DEPOSIT SITE VISIT

Mr. Leon McGarry completed a two-day field visit to the PL Deposit Property from May 29-30, 2017 as part of due diligence in the preparation of this Report. Travel days from Toronto to Flin Flon and return included the evening of May 28 and May 30, 2017. Mr. McGarry met with and was accompanied in the field by Mr. Chris Buchanan, M.Sc., P.Geo, Minnova Senior Geologist, and with whom previous operators and the Company's exploration activities, methodologies, findings, and interpretations were discussed. Mr. McGarry reviewed selected sections of the 2017 drill core from the PL Deposit archived on the property. All technical observations were as reported by Minnova. Several verification samples of drill core were collected. Mr. McGarry stayed at the exploration camp at the PL Deposit on the night of May 29, 2017.

12.1.1 REVIEW OF FIELD EXPLORATION

The Author was able to locate and verify the position of 13 drill holes in the field, using a handheld Garmin GPS60™ unit. Collar monuments employed at the PL Deposit project include wooden stakes approximately 5 cm in section with aluminium tags.

Coordinate differences between database drill collars and those collected by the Author, range from 2-5.7 m. The GPS device reported a margin of error of ±3 m for all measurements. Accordingly, the collar locations recorded in the database are considered reliable (Table 12.1).

Table 12.1: 2017 Site Visit Drill Collar Location Check

Hole ID	Company	Database		CSA		Distance Apart (m)
		Easting	Northing	Easting	Northing	
A3-051	Auriga	373,447	6,100,539	373,449	6,100,535	4.1
A3-061	Auriga	373,352	6,100,644	373,350	6,100,646	2.8
A3-083	Auriga	373,191	6,100,735	373,190	6,100,733	2.6
A3-110	Auriga	373,242	6,100,670	373,241	6,100,675	5.0
M-17-004	Minnova	373,350	6,100,640	373,354	6,100,636	5.7
M-17-005	Minnova	373,294	6,100,626	373,296	6,100,631	4.9
M-17-017	Minnova	373,289	6,100,607	373,287	6,100,607	2.0
M-17-019	Minnova	373,287	6,100,653	373,285	6,100,656	4.1
M-17-022	Minnova	373,243	6,100,653	373,238	6,100,652	4.9
M-17-023	Minnova	373,223	6,100,695	373,221	6,100,695	2.5
M-17-024	Minnova	373,223	6,100,722	373,223	6,100,728	5.8
M-17-031	Minnova	373,139	6,100,762	373,137	6,100,763	2.6
M-17-047	Minnova	373,228	6,100,674	373,228	6,100,672	2.3
Note: Coordinates in WGS84 UTM Zone 6N						

12.1.2 VERIFICATION SAMPLING

The Author conducted limited verification sampling during the 2017 site visit, which included nine (9) samples of quarter core from drill holes M-17-005, M-17-010, and M-17-026.

The Author collected and sealed the sample bags with ladder lock ties and maintained possession of all samples until personally delivered to Gardwine General Freight for onward delivery to Activation Laboratories Thunder Bay, Ontario for sample preparation and analysis. The sample preparation and analysis techniques described in Section 11.2 were also used for verification sample analysis.

Overall, the average assays from quarter core check samples collected from the 2017 drill holes correlate with average original assays, returning average grades of 4.1 and 4.2 g/t Au, respectively. However, on a sample pair basis, there is a large range of variation as shown in Table 12.2. Given the nuggety nature of mineralization at the PL Deposit, this variation is to be expected. Accordingly, these samples provide an independent confirmation of the presence of significant gold mineralization at the PL Deposit.

Table 12.2: 2017 Duplicates versus Original Samples

Hole ID	From (m)	To (m)	Original Sample ID	Lab	Original Assay (g/t Au)	Check Sample	Check List	Check Assay (g/t Au)
M-17-005	77	78	1252819	SGS	4.34	102822	ACTLAB	20.70
M-17-005	78	79	1252820	SGS	1.1	102823	ACTLAB	0.38
M-17-005	80	81	1252825	SGS	2.04	102824	ACTLAB	10.1
M-17-010	24	25	268313	SGS	3.16	102819	ACTLAB	0.82
M-17-010	25	26	268314	SGS	0.948	102820	ACTLAB	0.92
M-17-010	26	27	268317	SGS	0.165	102821	ACTLAB	0.15
M-17-026	343	344	1252494	SGS	0.903	102816	ACTLAB	0.60
M-17-026	345	346	1252497	SGS	3.6	102817	ACTLAB	3.06
M-17-026	346	346.95	1252498	SGS	21.69	102818	ACTLAB	0.04
Standard Sample CDN-GS-4B (Expected Grade 3.77 g/t ± 0.35)						*102825*	*ACTLAB*	*4.03*
					Blank	*102826*	*ACTLAB*	*0.0025*

12.1.3 DATA VALIDATION

Hard copy records selected for verification included drill hole logs for historical Maverick Mountain and Pioneer exploration campaigns for drill holes distributed throughout the deposit. Scanned hard copy records were cross checked against digital records for typographic or data entry errors. Assay summary sheets are available within assessment file compilations for drill holes PUF-001 to PUF-156 with laboratory assay certificates for holes PUF-110, PUF-117 to 120, and PUF-141 to 147. Within scanned historical log documents provided by the Company, diamond drill hole assay analysis summary sheets are available for PUF-190 to PUF-420. For historical holes, approximately 10% of the digital gold assay data was cross checked against historical logs. No significant typographic errors were identified, indicating a suitably accurate level of data entry.

For Minnova and Auriga drilling, digital geological logs and assay tables were cross checked against available drill cores to confirm the rationale for assigned lithology codes and to review the association between quartz veining, mineralization, and gold grades. Geological logging was found to be of sufficient quality to allow the use of assigned lithological codes for interpretation of the deposit.

The Author considers available data for the PL Deposit adequate for the estimation of Mineral Resources and disclosure within this technical report.

A lithology file that incorporates the historical observations recorded in available geological logs would benefit the Project. Minnova should continue to compile, review, and digitise historical geological logs. Where possible, original assay certificates should be compiled to allow verification of all historical assays.

12.2 NOKOMIS DEPOSIT SITE VISIT

Mr. Leon McGarry, P.Geo., visited the Nokomis Project between the January 20-21, 2014 to collect verification samples and confirm the location of drill collars. At the time of the site visit, Mr. McGarry was employed by ACA Howe International Limited, now part of CSA Global Canada Geosciences Ltd.

12.2.1 DATA VALIDATION

The Nokomis core was examined, and five verification samples were selected from three holes by taking ¼ splits of the remaining half core. An effort was made to sample a range of grades. The collection of samples was overseen by the Author. Sample bags were sealed with zip ties, placed in a rice bag, and brought by Mr. McGarry to the AGAT Laboratory ("AGAT") in Mississauga, Ontario for analysis.

AGAT maintains ISO registrations and accreditations, which provide independent verification that a Quality Management System designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards. Gold was determined using fire assay on a 30 g aliquot, with an AAS finish. Samples assaying greater than 10 g/t Au were rerun using fire assay with a gravimetric finish.

A comparison of the results is presented in Table 12.3.

Table 12.3: ACA Howe Check Samples

Drill Hole	From (m)	To (m)	Original Sample ID	Minnova (g/t Au)	ACA Howe (g/t Au)	Difference (g/t Au)
A18	166.95	167.45	1273398	8.548	3.43	-5.1
A4	32.70	33.20	1273135	4.262	6.36	2.1
A11	58.26	58.76	1273068	10.740	8.8	-1.9
A11	58.76	59.26	1273069	7.989	13.1	5.1
A3	12.24	12.74	1273157	20.051	21.1	1.0

The Author considers available data for the Nokomis Deposit adequate for the estimation of Mineral Resources and disclosure within this technical report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 HISTORIC PUFFY LAKE MILL OPERATION

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 tpd and rates of up to 1,500 tpd for short periods were reported, although average production was generally well below 1,000 tpd.

The process consisted of conventional crushing and grinding to a target grind of 100 µ followed by flotation to produce a sulphide concentrate containing gold. A jig in the grinding circuit recovered coarse free gold.

The flotation concentrate was reground and leached in a conventional cyanidation plant using drum filtration for liquid/solid separation followed by Merrill-Crowe recovery of gold from solution. Precipitated gold was smelted to yield a doré gold product.

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

Table 13.1 summarizes the performance of the mill during its operation. Figure 13.1 and Figure 13.2 present recoveries.

Table 13.1: Historical PL Mine Production				
Month	**Tonnage (tpd)**	**Gold Produced (oz)**	**Silver Produced (oz)**	**Gold Recovery (%)**
Dec-87		450.3	190.8	72.93
Jan-88	881	1,368.9	386.0	72.93
Feb-88	799	1,216.3	383.1	83.83
Mar-88	901	1,446.9	380.5	81.69
Apr-88	979	1,600.9	334.1	81.18
May-88	1025	1,804.0	310.0	73.08
Jun-88	833	2,227.0	722.0	80.01
Jul-88	613	1,110.9	437.1	87.79
Aug-88	784	1,306.5	782.2	81.44
Sep-88	662	1,498.2	741.1	89.22
Oct-88	737	2,617.0	941.0	86.99
Nov-88	728	1,827.7	623.3	82.57
Dec-88	652	1,720.7	550.0	82.07
Jan-89	830	1,862.8	568.3	86.24
Feb-89	583	2,003.3	507.9	88.5
Mar-89	745	1,646.7	446.1	83.77
Apr-89	694	936.4	275.7	86.04
May-89		1,834.9	441.5	
Totals		**28,479.4**	**9,020.7**	

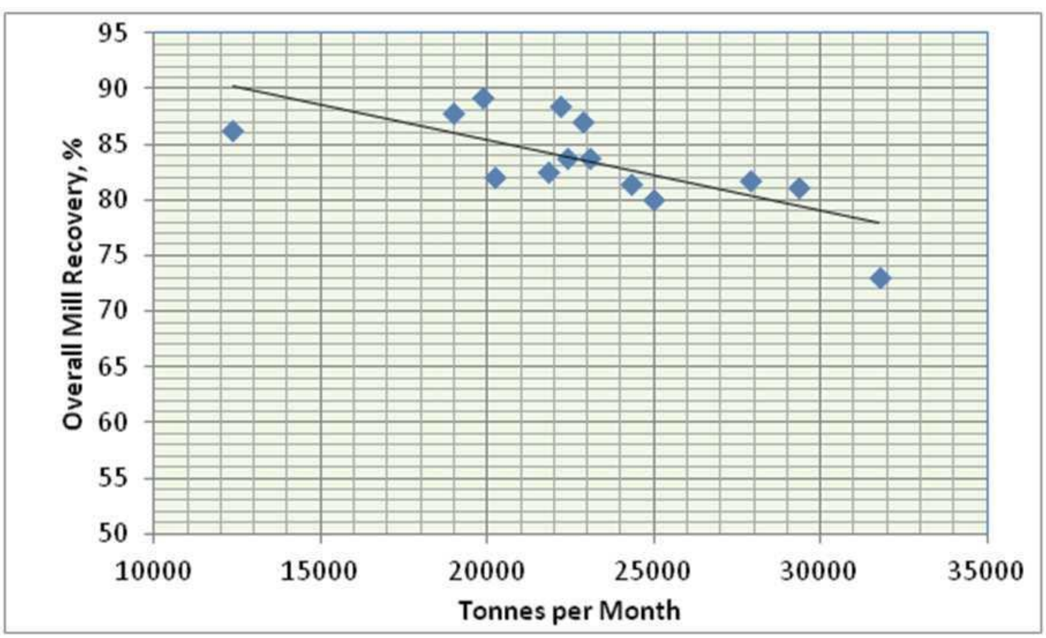


Figure 13.1 Recovery versus Tonnage, 1988-89

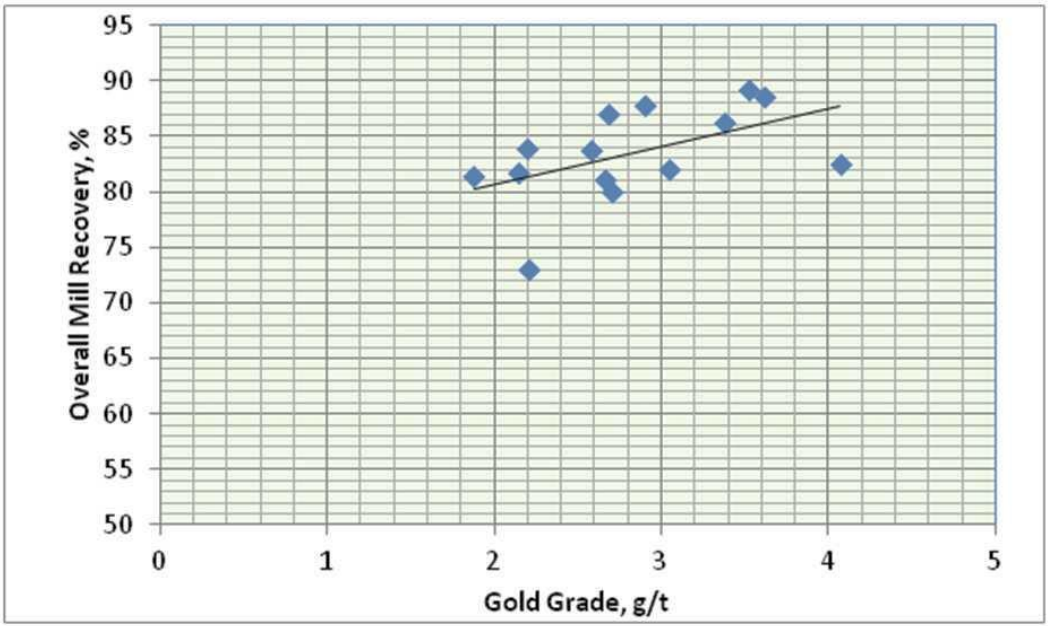


Figure 13.2 Recovery versus Grade, 1988-89

13.2 METALLURGICAL TESTING

Information in this section was taken from the "Technical Report on Puffy Lake Mine," dated June 30, 2006 and Authored by Karel Pieterse. No recent metallurgical testwork has been reported.

On April 13, 1981, Britton Research Limited ("Britton") reported on a series of metallurgical tests (April 13, 1981, Project B495. Britton Research Limited, Vancouver, B.C. Metallurgical tests on a Sample of Gold Ore) 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 10 mesh to 63% -200 mesh.
- The sample had a specific gravity of 2.87 g/cm^3.
- Although an appreciable amount of arsenopyrite was present in the sample, it was not refractory to cyanidation. Cyanidation was carried out for 72 hours, after grinding to 63% -200 mesh 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% -200 mesh raised the extraction to 92.7%. A further increase to 95% -200 mesh yielded an extraction rate of 95.4%.

- Three-stage rougher flotation, after grinding to 63% -200 mesh, followed by amalgamation of the tailing, recovered 93.1% of the gold, in combined concentrates, assaying 73.7 g/t. Finer grinding to 95% -200 mesh increased the recovery to 94.8%.
- Possible methods of treating the ore include the following. In each case, a jig would be installed in the grinding circuit to recover coarse gold:
    - 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. The choice of the method to be used would depend on the results of additional metallurgical investigations, to be followed by a preliminary feasibility study, which would include estimates of capital and operating costs, as well as environmental studies. The possibility of shipping the ore direct to a smelter would also be investigated.

On October 4, 1984, Lakefield Research ("Lakefield") reported on an investigation of the recovery of gold on a bulk sample (October 4, 1984, Project L.R.2873. Lakefield Research, Lakefield, Ontario, Canada (An Investigation of the Recovery of Gold). The results are summarized as follows:

- Thirty (30) drums of material were received by Lakefield. Shipping weight was 6,818 kg (6.818 tonnes).
- The overall 'calculated' grade of the 6.8 tonne bulk sample 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.
- The flowsheet evaluated in the pilot plant is shown in Figure 13.3.
- The 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 200 mesh. The calculated work index was 13.4.
- **Flotation and Gravity Circuits**. Upon completion of the three days of tests, the pilot plant was cleaned out and a mass balance was established. Gold recovered from tests together with that from clean-up (assumed to be recoverable during gravity concentration), an overall metallurgical balance was projected.
- 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%.

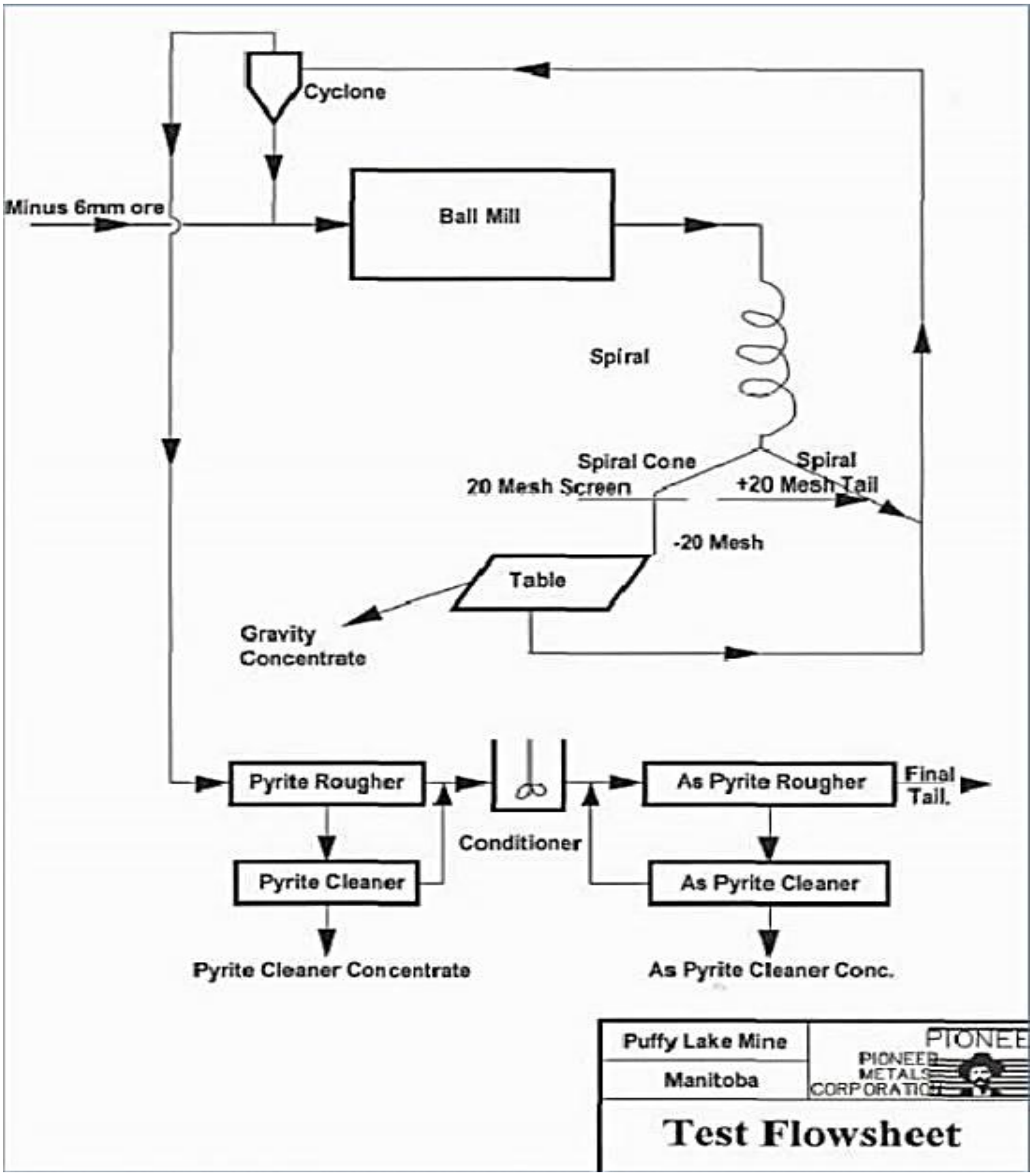


Figure 13.3 Lakefield test flowsheet

On November 25, 1986, Coastech reported on confirmatory metallurgical testwork (25 November 1986. Coastech Research Inc., Vancouver, B.C. 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 grams per tonne. 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, except Lot 1. 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.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

During the period July to October 2017, CSA carried out a mineral resource estimate ("MRE") update study for the PL Deposit based on the collection of cored geologic data from a 2017 exploration drilling program undertaken by Minnova. In the opinion of the Author, 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 November 1, 2017 and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. The MRE is generated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice Guidelines" (CIM Council, 2003).

The MRE for the PL Deposit has been prepared by Mr. Leon McGarry, CSA Senior Resource Geologist and a Qualified Person (QP) for the reporting of Mineral Resources, as defined by NI 43-101. Mr. McGarry is responsible for the geological domaining, block modelling, MRE studies presented in this Report Section. Mr. McGarry visited the PL Project site between May 29, 2017 and May 30, 2017 to review diamond drill core, confirm the location of drill collars and outcrops and to collect the verification samples detailed in Section 12 – Data Verification.

Previous NI 43-101 MREs generated for the PL Deposit are described in Section 6 – History and include the 2011 MRE prepared by P&E and reported in Orava, D., et al., 2012, and the 2014 MRE prepared by P&E and reported in Burga, et al., 2014. The current MRE presented in this Report supersedes all past estimates and benefits from the changes that are summarized in Section 14.2.14 – Comparison with Previous Mineral Resource Estimate.

The 2012 mineral resource estimate of the Nokomis Deposit, also prepared and reported by Mr. L. McGarry as a Qualified Person (QP), has not been updated and is restated herein. A brief summary of the Nokomis MRE is given in Section 14.3 – Nokomis Mineral Resource Estimate. The reader is referred to Burga, et al., 2014 for more details.

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 PL DEPOSIT

14.2.1 INFORMING DATA AND VALIDATION

14.2.1.1 PL Deposit Drill Data

The Author has relied on the following drill hole data for the PL Deposit provided by Minnova by way of a digital data export containing a Microsoft Access™ database transferred to CSA on July 23, 2017.

- 416 diamond drill holes completed by Maverick Mountain between 1981 and 1986 and Pioneer between 1980 and 1988 totaling 74,885 m. Holes PUF-01 to PUF-055 were AQ size, all others were NQ size. Drilling comprised predominantly vertical holes (376 vertical and 40 inclined) drilled on a northwest-southeast orientated grid at 30 m to 60 m collar spacings. Vertical holes typically intercepted mineralized horizons at an angle of 50 degrees, with downhole intercepts equal to approximately 130% of horizon true thickness. Hole collar locations were surveyed by theodolite and chain and compass methods in Mine Grid coordinates.
  - Down hole survey data derived from Topari and dip acid tests is available for 3 holes;
  - Gold analyses by fire assay are available for 1,380 samples from 416 drill holes representing a combined length of 21,550 m;
  - Lithology data is available for 350 drill holes; however, for 184 drill holes only the depth to overburden is recorded. For the remaining 166 drill holes, dominant lithologies are recorded; and
  - A proportion of historical drill hole logs were checked and compared to the Minnova database "MCIPLD_DDH_Queries.mdb." No obvious historical transcription issues were identified.

- 106 NQTW diamond drill holes completed by Auriga between 2010 and 2011 totaling 11,280 m. Drilling comprised predominantly vertical holes that infilled the historical grid. Drill holes were surveyed by Balchen & Kulchycki Surveys of Dauphin, Manitoba.
  - Down hole survey data, derived from a *Reflex single shot*, is available for 91 holes;
  - Gold analyses by fire assay are available for 3,742 samples from 94 drill holes, representing a combined length of 2,206 m; and
  - Lithology data is available for 102 holes and includes codes for major rock types.

- 48 NQ size diamond drill holes completed by Minnova between 2016 and 2017 totaling 8,900 m. Drilling comprised predominantly vertical holes that infilled between historical holes at the northern end of the deposit. Drill holes were surveyed by Balchen & Kulchycki Surveys of Dauphin, Manitoba.
  - Down hole survey data, derived from an EZ-Shot device, is available for 46 drill holes;
  - Gold analyses by fire assay are available for 1,380 samples from 48 drill holes representing a combined length of 1,347 m; and
  - Lithology data is available for all holes and includes: major and minor rock types, mineralization, alteration, structural, and geotechnical data.

The current grid system used is NAD83 UTM Zone 14N. Drill hole azimuths are recorded in True North. Measurements are in metric units.

All drill hole data was imported into Micromine™ software and interrogated via Micromine™ 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 was performed to confirm drill hole depths, inconsistent or missing sample/logging intervals, and survey data. The data was validated – checked for logical or transcription errors, such as overlapping intervals. There were a few, very minor errors that were corrected.

The Author 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.

14.2.1.2 Assay Data by Operator

Drill data is generated by multiple operators with variable sampling and analysis relative to methods. Relative to historical gold assay data, the Auriga and Minnova gold assay data sets have a larger proportion of samples with gold grades above 0.2 g/t Au. The relative bias, shown in the left cumulative distribution plot in Figure 14.1 can be attributed to the focusing of those programs on higher-grade portions of the deposit.

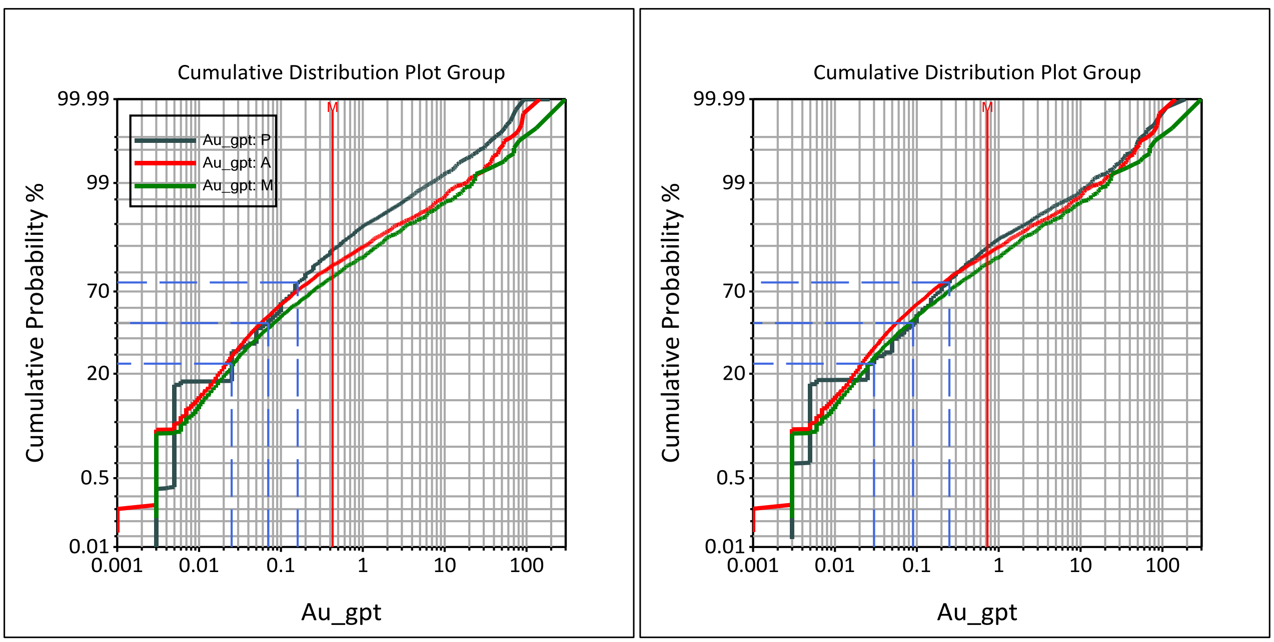


Figure 14.1 Cumulative Distribution of Assay Data by Exploration Campaign, within the entire deposit (left) and within 60m of recent drill holes (right)

A comparison between Auriga and Minnova gold assay data and historical data within 30 m of recent holes shows no material difference between the distribution of gold assay grades and no apparent bias, as shown in the right cumulative distribution plot in Figure 14.1. Historical Pioneer and Maverick gold assay data can be compiled with Auriga and Minnova assay data for the purpose of estimating gold mineral resources.

14.2.1.3 Topography

A digital elevation model ("DEM") with a lateral resolution of 5 m and vertical resolution ±1 m was provided by Minnova. The DEM was generated by an airborne LIDAR Survey completed in 2011.

The elevation of historical holes was not systematically surveyed. For 174 historical PUF series holes, a default elevation of 350 masl is recorded in the digital database. For these holes, the DEM elevation is considered more accurate and is entered as the preferred elevation.

14.2.1.4 Model of Historical Workings

A three-dimensional model of underground workings in .dxf format was provided by Minnova. The model represents mine development and mined stopes. A cross check of the model against the LIDAR DEM indicates a datum shift of approximately 5 m is required to align the digital model of the mine portal with the DEM surface.

Accounting for this shift, the digital model of mine workings aligns with projected mineralized intervals in drill holes throughout the deposit and supports the drill hole database information.

The position of historical mine workings with interpreted vein domains is shown in Figure 14.2.

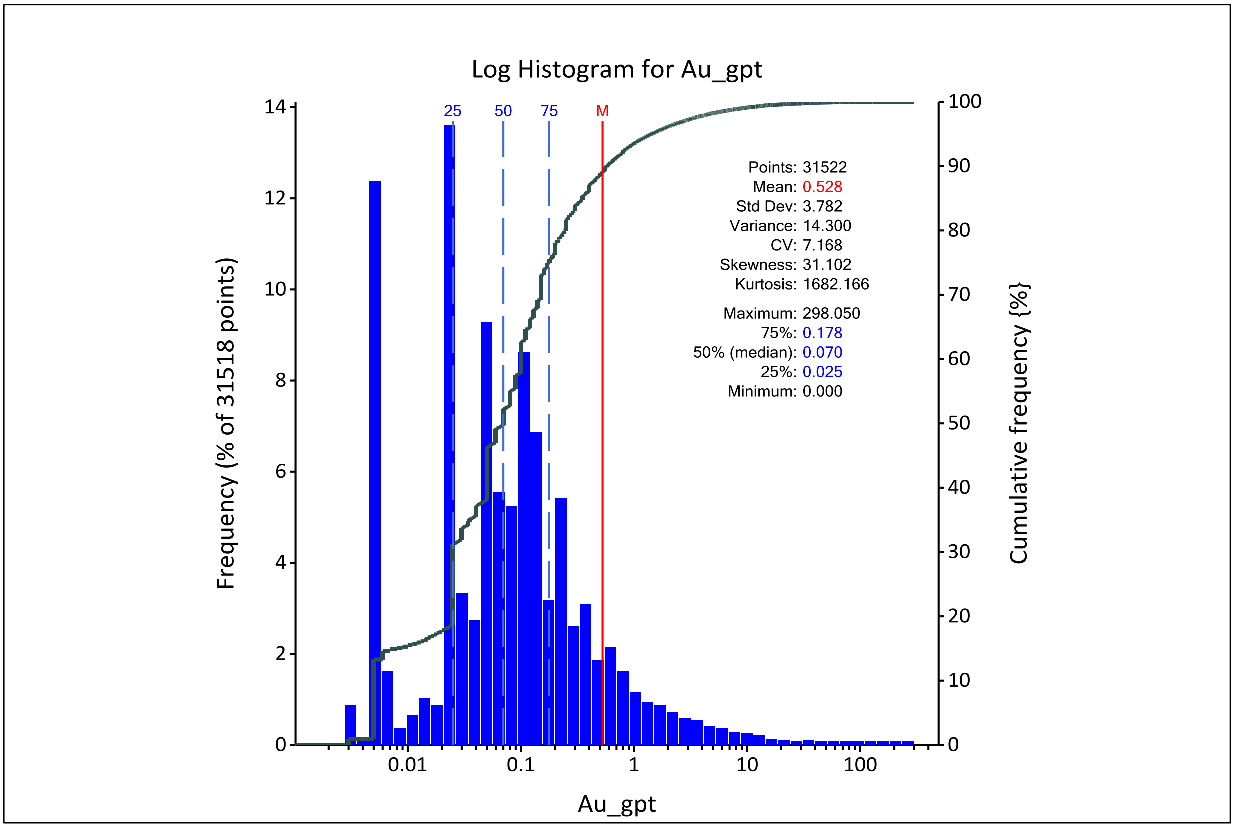


Figure 14.2 Histogram of raw assay data from the PL Deposit

14.2.2 Geological Interpretation

14.2.2.1 Software

Geological modeling was undertaken by CSA Global using Micromine™ and Leapfrog™ software.

14.2.2.2 Preliminary Statistical Assessment

Descriptive statistical analysis of assay data is undertaken for the identification of assay populations, which may represent separate styles of gold mineralization. Specifically, this analysis is undertaken to estimate the natural gold cut-off grade that defines mineralized envelopes and to determine the distribution parameters for gold.

A review of the drill hole assay histogram, shown in Figure 14.2, indicates that gold assays for the Project show a logarithmic distribution that is positively skewed with a large high-grade tail. The mean grade of 0.53 g/t Au is significantly higher than the median grade of 0.07 g/t Au. It is not possible to discern a natural population break associated with a particular style or generation of mineralization at an economically significant grade.

14.2.2.3 Lithology and Mineralization Modeling

Lithological and mineralization features were defined from logged and interpreted geology and assay data.

In Micromine™ software, drill hole traces were colour-coded according to lithology and gold assay values were reviewed against available geological maps in three-dimensional and cross section views. Cross sections at 20 m spacings were displayed with a clipping window equal to a half the distance from the adjacent sections.

Mineralized intervals were domained by manually assigning assay intervals to vein set domains:

- Intervals were selected to exceed a notional cut-off grade of 1 g/t Au over at least 1.5 m true thickness; and
- Where this grade and thickness cannot be achieved, a 1 m interval of less than 1 g/t Au was assigned to the domain to honour interpreted vein continuity.

The domained assay file was imported into Leapfrog™ to generate wireframe models of mineralization. For each vein set, hanging wall and footwall contact depths were contoured using Leapfrog™ functions, to generate closed wireframe solids that were snapped to drill hole intervals.

The extent of each vein model was limited using a boundary string. Wireframe solids are projected from drill hole intervals by up to 50 m along strike and 100 m down dip. If a vein set did not extend to the adjacent drill hole section, the wireframe was projected half way to the next section and terminated. The base of overburden depth was contoured using Leapfrog™ functions. Vein sets were extended up to the overburden surface.

The interpreted direction and dip of the vein sets was maintained in wireframe models, although the wireframe thickness was reduced from the last known intersection. The domained assay file was reviewed to ensure that no unassigned intervals fell within the wireframe.

Fourteen (14) domains were modelled with variable volumes and drill densities. Of these, six (6) were considered major domains having between 232 and 636 samples assigned to each. The following features are noted:

- All veins pinch and swell along strike, with lenses or portions above 1.5 m in thickness ranging in extent from 50 m to 150 m in the along strike and down dip directions;
- Plunging trends were observed locally in the Sherridon Vein where the majority of mineralization is constrained to plunging trends at 10/105E, which extend over distances of 350 m, and the Lower Vein – South where plunging trends at 20/110E, which extend over distances of300 m;
- The Middle, Upper, and Main Veins are predominantly discontinuous and less than 1 m thick with no discernable trends to gold mineralization. Vein segments greater than 1.5 m in thickness and with gold grades above 2 g/t Au range from 50 m to 150 m in extent in all directions; and

* The Lower Vein is continuous throughout the deposit; however, it has been divided into a north and south domain due to a change in dip direction at 6,100,500 m North where the vein thins and becomes poorly mineralized. This is also the northern limit to historical underground workings shown in Figure 14.3.

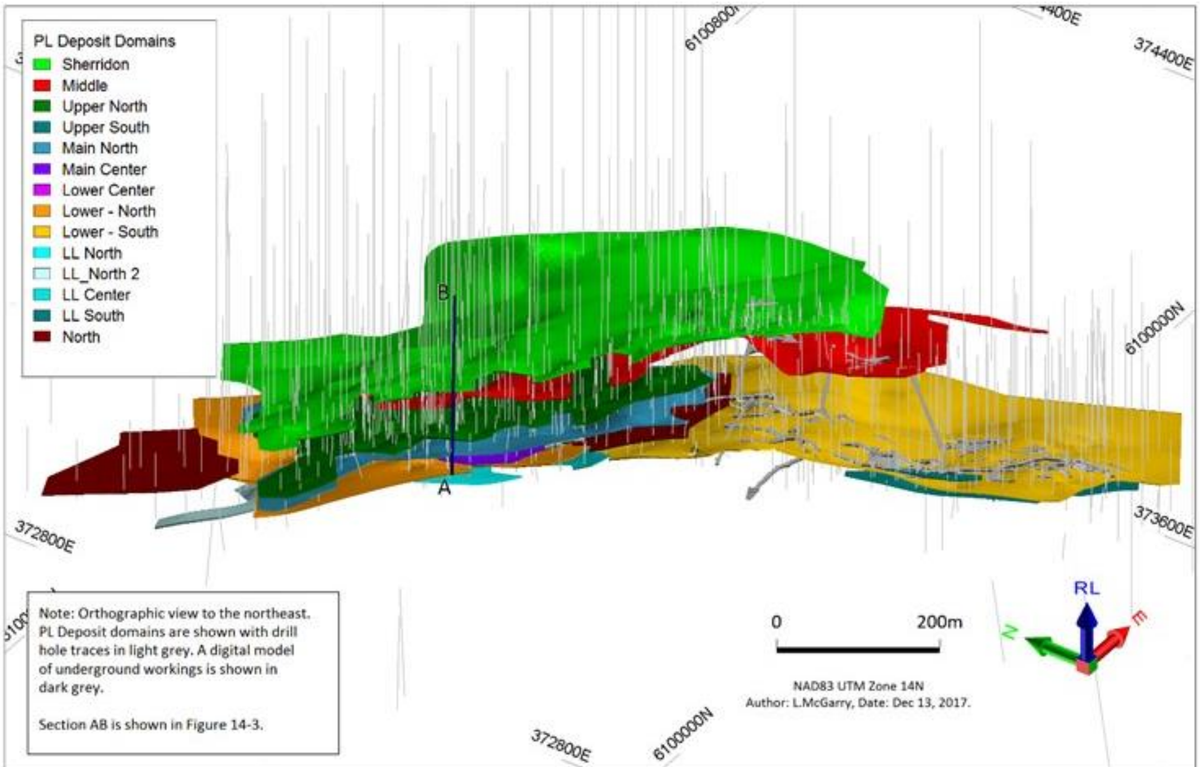


Figure 14.3 PL Deposit wireframe domains – Orthographic view to the northeast

Eight minor domains are also modelled with between 15 and 98 samples assigned to each. Minor domains are interpreted to represent main vein splays or parallel sheets of limited extent. Those that occur in the footwall of the Lower Vein were historically called the 'Lower Lower' Veins (Kilbourn, 1993) and are prefixed with 'LL' in this study.

Figure 14.3 shows modelled domains in a three-dimensional view to the northwest. Figure 14.4 presents a typical cross section through the deposit with drill hole traces annotated with gold assay bar charts.

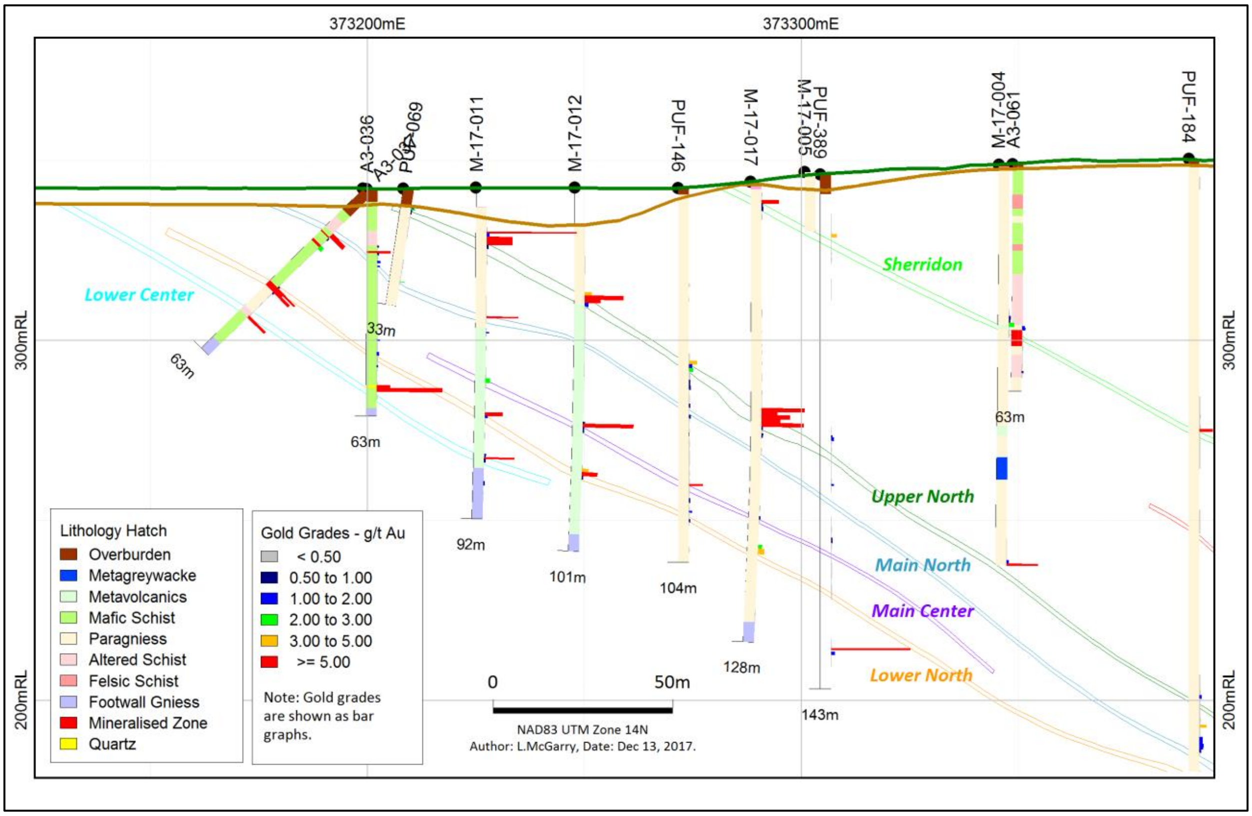


Figure 14.4 Example of interpretation of mineralization and geological features – Section 760 m N

Modelled vein domains are listed in Table 14.1 with dimensions and approximate drill hole spacing.

Table 14.1: PL Deposit Vein Set Domain Details										
Domain Name	**Code**	**Volume (m^3)**	**Area (m^2)**	**Strike Extent (m)**	**Down Dip Extent (m)**	**Depth Extent (m)**	**Average Thickness (m)**	**Number of Drill Holes**	**Number of Samples**	**Average DH Spacing (m)**
Sherridon	SH	1,103,000	701,000	770	1,270	580	1.57	261	529	50
Lower – South	Slo	690,000	420,000	730	780	380	1.64	227	636	45
Lower – North	NLo	674,000	397,000	500	1,080	560	1.70	166	406	50
Main North	Nmn	511,000	347,000	550	1,070	560	1.47	191	313	45
Upper North	Nup	502,000	345,000	520	1,090	540	1.46	181	370	45
Middle	Mid	829,000	505,000	860	1,160	530	1.64	124	232	65
LL South	Lso	50,000	45,000	335	200	110	1.11	41	64	35
Lower Center	Clw	61,000	40,000	170	425	180	1.53	38	98	30
LL Center	Lcd	47,000	45,000	150	400	190	1.04	34	38	35
Main Center	Cmd	29,000	24,000	160	190	130	1.21	26	41	30
Upper South	Sup	11,000	6,000	120	75	50	1.83	15	43	20
LL North	Lcn	34,000	22,000	230	205	100	1.55	10	19	45
LL North 2	Lno	48,000	27,000	150	210	100	1.78	6	17	65
North	Nth	57,000	38,000	230	230	70	1.50	6	15	80

14.2.3 DOMAINED SAMPLES

Sample data was grouped into lithological domains for statistical analysis of gold assay grades in g/t. A summary of the 14 domain codes used to distinguish the data during geostatistical analysis and estimation is shown in Table 14.1.

Univariate statistics for gold, collected from each mineralized resource domain, are presented in the box and whisker plots in Figure 14.5.The following features are observed:

- Positively skewed symmetrical logarithmic distributions for all domains, with mean grades significantly higher than the median values;
- Comparable mean gold grades across most domains. This is in part due to the compositing process to achieve minimum grade and thickness; and
- Of the six major domains, the North Upper, North Main, and Middle Veins have a larger spread of grades and incorporate a greater number of lower grade samples due to lower vein continuity.

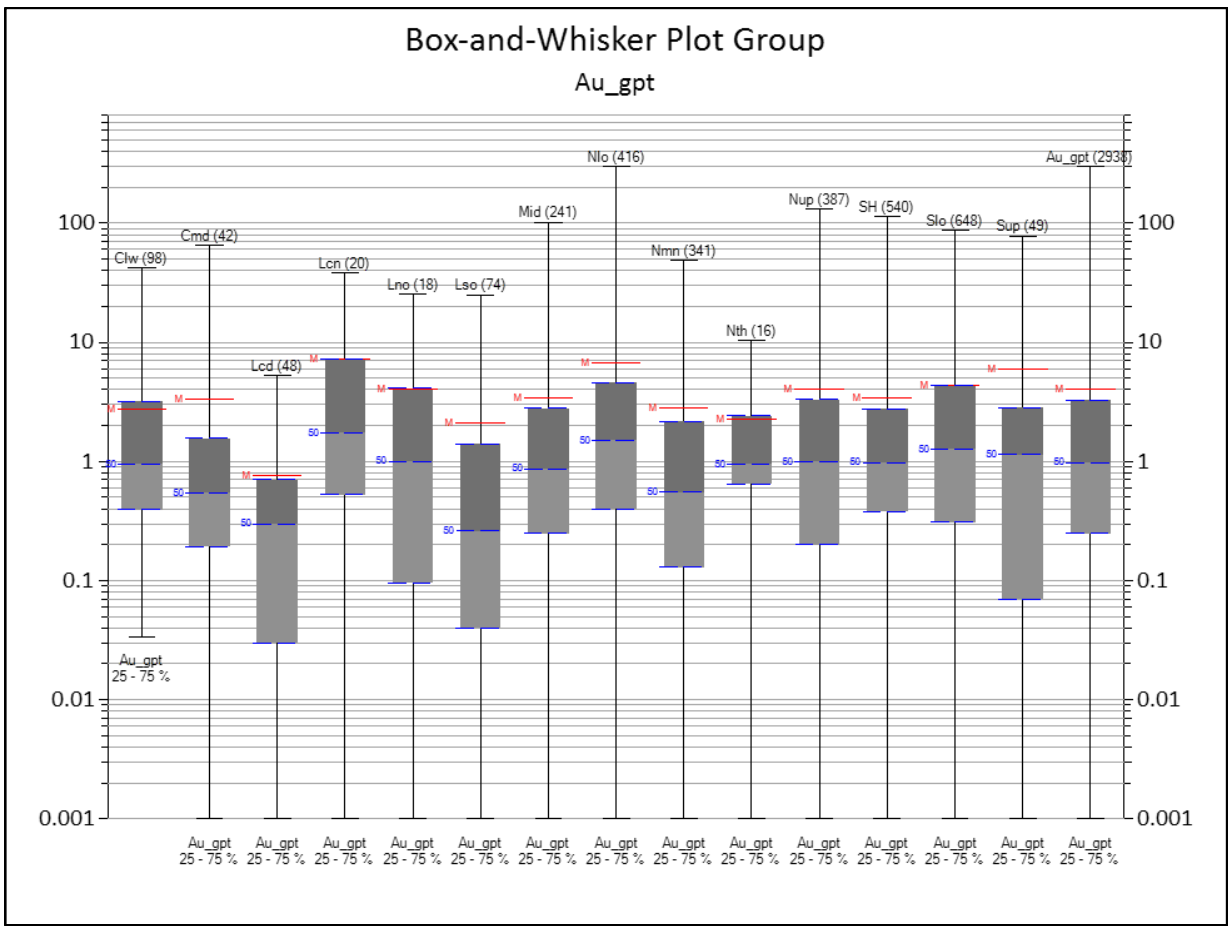


Figure 14.5 Box and whisker plot of domained assay data

14.2.4 DENSITY

For the 2014 resource estimate, an average bulk density of 2.81 g/cm^3 was selected based on 33 samples collected by P&E (Burga, et al., 2014). Density measurements for these samples were undertaken at AGAT Laboratories in Mississauga, Ontario and ranged from 2.50 g/cm^3 to 2.96 g/cm^3, with an average density of 2.81 g/cm^3.

In 2017, Minnova undertook density determinations on 10 cm core fragments using the displacement method. Density determinations of 158 samples were made on intervals within or close to mineralized zones. Measured densities ranged from 2.26 g/cm^3 to 3.33 g/cm^3 with an average of 2.81 g/cm^3 and a standard deviation of 0.12 g/cm^3. Of a total of 158 determinations, 79 fell within MRE domains with an average of 11 determinations per domain.

There is insufficient data to determine density on a rock type basis. Instead, the average bulk density from the P&E program and the 2017 Minnova program has been utilised by the Author.

14.2.5 SAMPLE COMPOSITING

14.2.5.1 Sample Length Analyses

To generate representative length-weighted composites, a sample length analysis was conducted. As shown in Figure 14.6, of the 26,053 assays in the resource database, 48% of samples are 1 m in length, 17% are 0.5 m, and 11% are 1.5 m long. For domained assays, the number of 1 m samples decreases to 24% and the number of 0.5 m samples increases to 28%.

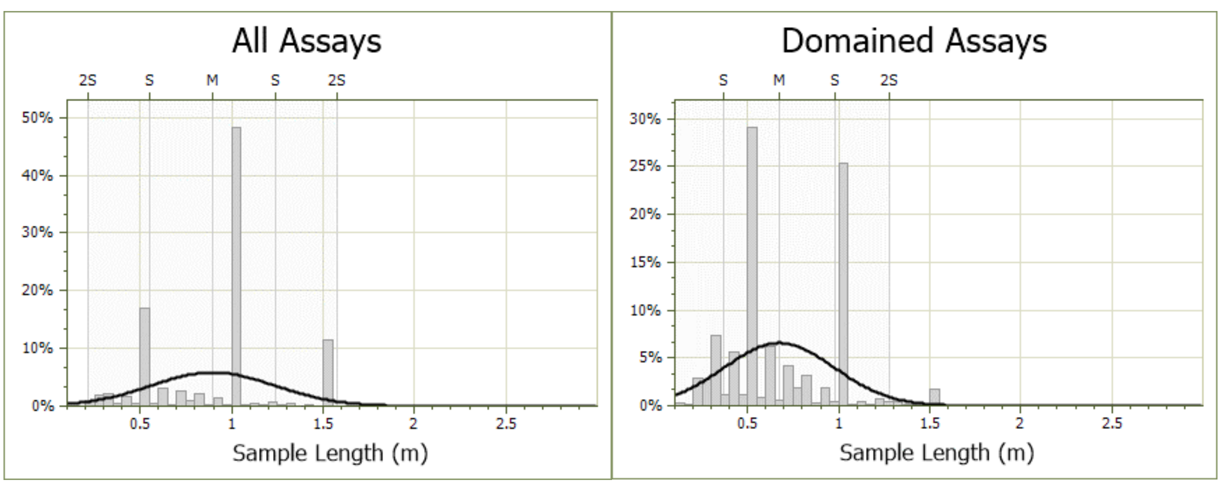


Figure 14.6 Histogram of sample lengths

To ensure equal sample support, and to avoid splitting assay intervals, a composite interval length of 1 m has been selected. This length is equal to the second most common domained assay interval and is a multiple of the dominant interval length. Domained assays were regularized using the length-weighted averages of gold grades.

Gold grade can rapidly change over short distances and veins can pinch out to a few centimetres in thickness. Therefore, it is appropriate to assume that un-sampled intervals did not contain any obvious mineralization. Prior to compositing, a nominal grade of 0.001 g/t Au was used to populate un-sampled assay intervals for gold on the basis that un-sampled core was perceived as barren by past operators and is more likely than not to be barren.

The number of dummy 0.001 g/t Au values added to each domain included: Sherridon - 11, Middle – 9, North Upper – 17, North Main -28 Lower South – 12 and Lower North – 10. For un-sampled intervals projected to be within or close to vein models, Minnova should sample available core to minimize the application of these default barren grades.

Composites that were less than 0.3 m in length were discarded so as to not introduce a short sample bias into the estimation process.

14.2.6 Statistical Analyses

Before undertaking the resource estimate, univariate statistical assessment of the composite data was undertaken. Exploration sample data were statistically reviewed and variograms were calculated to determine spatial continuity for composited sample gold values and density determinations.

Statistical analysis was carried out using Snowden Supervisor 8.7™ software. Statistics for each domain are presented in Table 14.2. Histograms for the six major domains are presented in Figure 14.7.

Table 14.2: PL Deposit Composite Summary

Domain	Count	Min	Max	Mean	Standard Development	Uncut CV	Capping Value	Number Capped	Capped Mean	Capped Standard Development	CV
SH	445	0.00	54.90	2.94	5.34	1.82	40.00	2.00	2.89	4.95	1.71
Slo	427	0.00	74.80	3.58	6.03	1.69	40.00	2.00	3.48	5.12	1.47
Nlo	307	0.00	103.47	5.25	10.97	2.09	45.00	5.00	4.82	8.37	1.74
Nup	301	0.00	71.04	3.57	6.84	1.92	35.00	2.00	3.43	5.81	1.69
Nmn	287	0.00	48.65	2.48	5.22	2.11	35.00	1.00	2.43	4.85	1.99
Mid	190	0.00	66.00	2.87	7.11	2.48	40.00	2.00	2.65	5.43	2.05
Clw	67	0.03	24.42	2.58	3.73	1.45					
Lso	54	0.00	9.45	1.31	2.03	1.55					
Lcd	43	0.00	3.92	0.71	0.97	1.36					
Cmd	36	0.00	32.87	3.22	7.27	2.26					
Sup	28	0.00	35.80	4.55	8.56	1.88	35.00	1.00	4.52	8.45	1.87
Lcn	17	0.00	36.18	6.97	8.83	1.27	35.00	1.00	6.90	8.59	1.25
Lno	12	0.51	25.16	5.14	7.67	1.49					
Nth	12	0.35	10.40	3.88	3.76	0.97					
Total	**2,226**	**0.00**	**103.47**	**3.38**	**6.90**	**2.04**	**45.00**		**3.25**	**5.80**	**1.79**

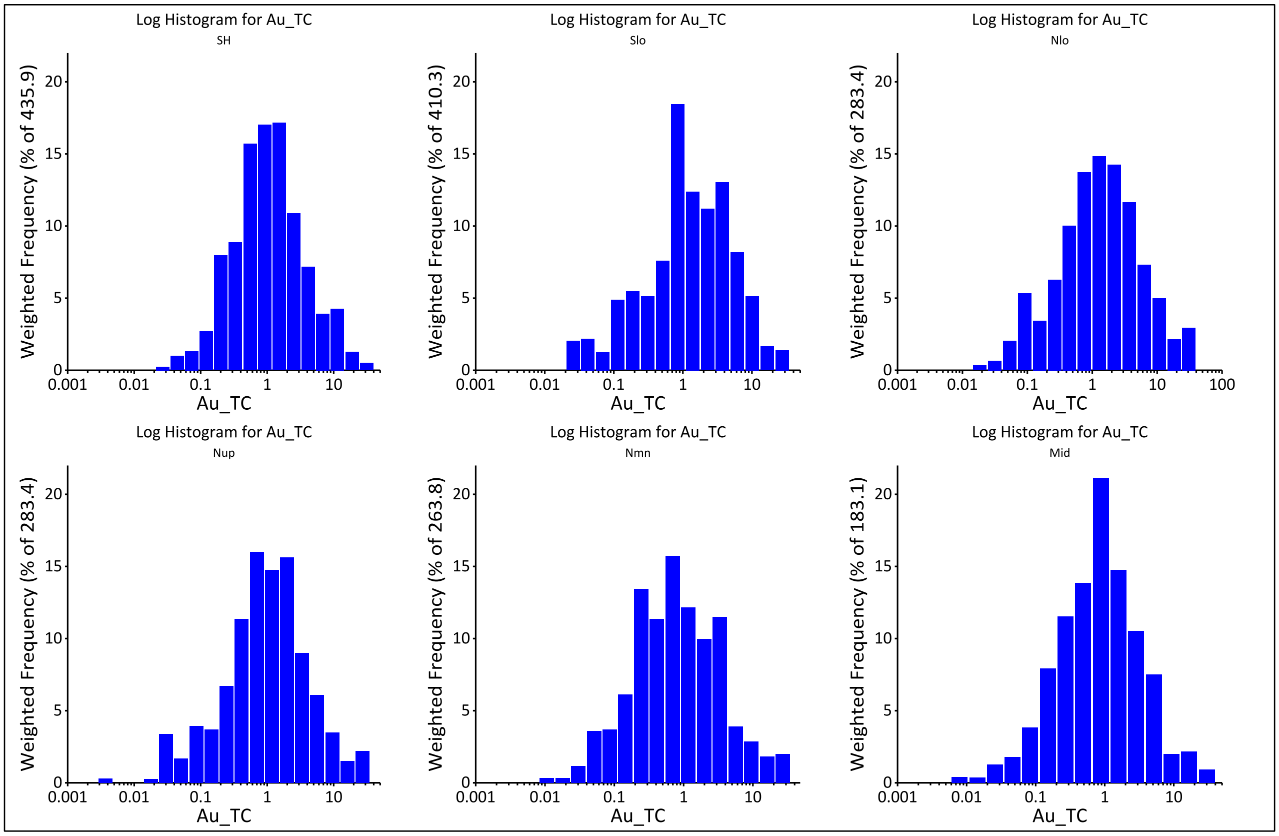


Figure 14.7 Histogram of capped composite gold grades for major PL Deposit domains

14.2.6.1 Summary Statistics – Sample Assays

The following features are observed:

- No discernable trends associated with depth or distance along strike were identified;
- For the major domains, average gold grades range from 2.48 to 5.25 g/t Au, with minor domains showing greater variability with average gold grades ranging from 0.71 to 6.97 g/t Au;
- For all domains, the coefficient of variation is high. For the six major domains, the uncapped Coefficient of Variation, or the standard deviation divided by the mean, ranges from 1.82 to 2.48. A high CV indicates a large spread of values about the mean and that capping of high values samples may be required;
- All major domains populations show symmetrical logarithmic distributions for gold;
- The thicker and more continuous Sherridon and Lower North Domains have a more log normal distribution attributed to those domains, containing a large number of holes that are focused on higher grade portions of each vein; and
- Where a significant number of lower grade intercepts are included in the wireframe to facilitate vein modeling for the Middle, Upper, and Main Domains; histograms are positively skewed.

14.2.6.2 Grade Capping

A review of grade outliers was undertaken to ensure that extreme grades were treated appropriately during grade interpolation. Although extreme grade outliers within the grade populations of variables are real, they are potentially not representative of the volume they inform during estimation. If these values are not capped, they have the potential to result in significant grade over-estimation on a local basis.

In general, very high grades (at the PL Deposit typically greater than 30 g/t Au) are located within the higher-grade portions of the deposit. Except for a 39.95 g/t sample in hole PUF-195 and a 74.8 g/t sample in hole PUF-297 at depth, most very high-grade samples are well constrained by surrounding drill holes. Accordingly, a relaxed approach to the application of top cuts was taken.

The cutting strategy was applied based on the following:

- Probability plots were reviewed to identify inflection points at the upper end of the distribution;
- Inflection points were rounded to the nearest 5 g/t Au interval to identify a capping value; and
- In addition, sample data were sorted into descending order and several capping scenarios applied to see what effect the capping value would have on the mean, standard deviation, and coefficient of variation (CV), as well as the loss of metal from the sample population. The impact of the capping value was assessed with the goal of bringing the CV below a value of 2.

The capping thresholds presented in Table 14.2 were selected, resulting in the capping of sixteen (16) gold composite assay values prior to estimation.

14.2.7 GEOSTATISTICS

The six major domains have a sufficient number of samples to generate meaningful variation models for composited and capped gold grades. Veins that did not have sufficient data for variography were modelled using the variogram model of the stratigraphically closest vein.

To honour the population distributions shown in the histograms in Figure 14.7, composite gold values undergo a normal score transform prior to being assessed for anisotropy, or directional dependence. Maps of gold value continuity were used to investigate the strike, dip, and pitch direction axis of the six major vein domains.

The grade variation between sample pairs orientated along each direction axis ±15 degree was reviewed using semi-variogram charts. Example semi-variograms for the Lower Vein South Domains are shown in Figure 14.8. Sample pairs are grouped by their separation distance, or 'lag interval' on the X-axis. For each lag interval assessed, half of average variance value of paired samples is plotted on the Y-axis. The resulting empirical semi-variogram chart can show if there is a relationship between grade variance and distance along each axis that can be modelled. Normal score variograms are back transformed to give the semi-variogram parameters presented in Table 14.3.

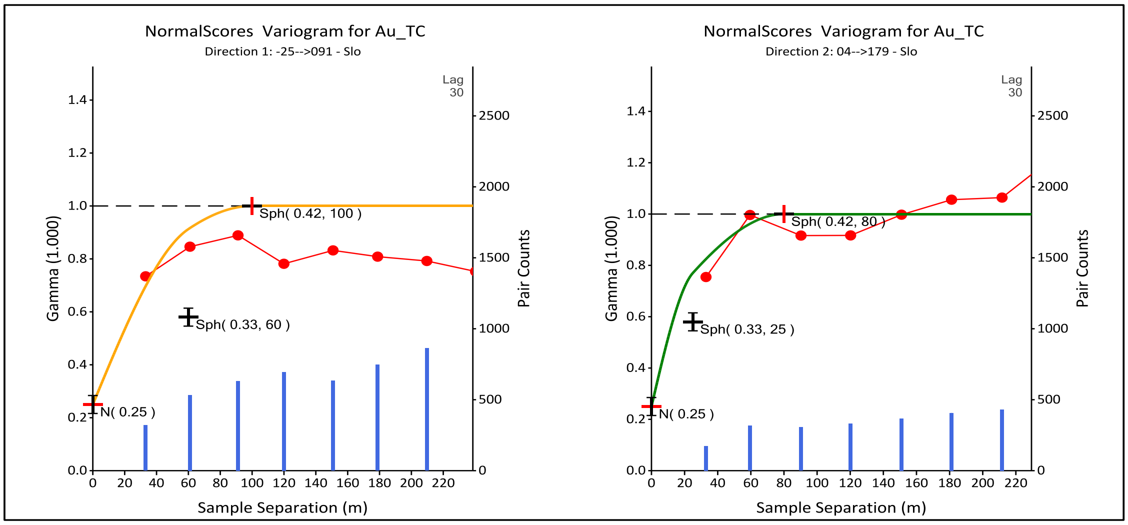


Figure 14.8 Example of major and semi-major axis variograms for the Lower Vein South Domain

Table 14.3: Modeled Semi-Variogram Parameters for PL Deposit Gold Grade Interpolation										
Zone	**Ellipise Rotation**			**Nugget Value**		**Model**	**Partial Sill**	**Range (m)**		
	Z	**Y**	**x**					**Major**	**Semi-Major**	**Minor**
SH	35	30	0	5.86	42%	1. Spher	5.75	40	20	15
						2. Spher	2.43	60	80	30
Slo	96	24	-6.3	6.8	34%	1. Spher	5.62	62	24	15
						2. Spher	7.65	100	80	30
Nlo	33	23	8	20.17	44%	1. Spher	19.55	50	47	15
						2. Spher	6.07	80	76	30
Nup	59	33	-11	9.64	42%	1. Spher	5.95	72	25	15
						2. Spher	7.11	120	62	30
Mid	72	23	-8	9.91	48%	1. Spher	6.09	70	60	15
						2. Spher	4.66	120	100	30
Nmn	63	28	-10	7.97	45%	1. Spher	2.96	40	60	15
						2. Spher	6.86	150	110	30

For all domains, semi-variogram charts for gold were modelled using two spherical functions. The semi-variogram models, described in Table 14.3, are sufficiently well defined to allow meaningful kriging calculations.

For all domains, semi-variogram models for gold have high nugget values, describing expected variation between samples at the same place, ranging from 34% to 48% of the total variance. High nugget values are in line with the style of mineralization and known levels of continuity where significant changes in grade and thickness across vein are observed.

Ellipses were visualised in Micromine and fit with vein orientations and apparent mineralization trends. Example search ellipses are shown in plan view for the Lower Vein North and South domains in

Figure 14.9 and Figure 14.10. Variogram models are used to define search ellipse anisotropy during estimation.

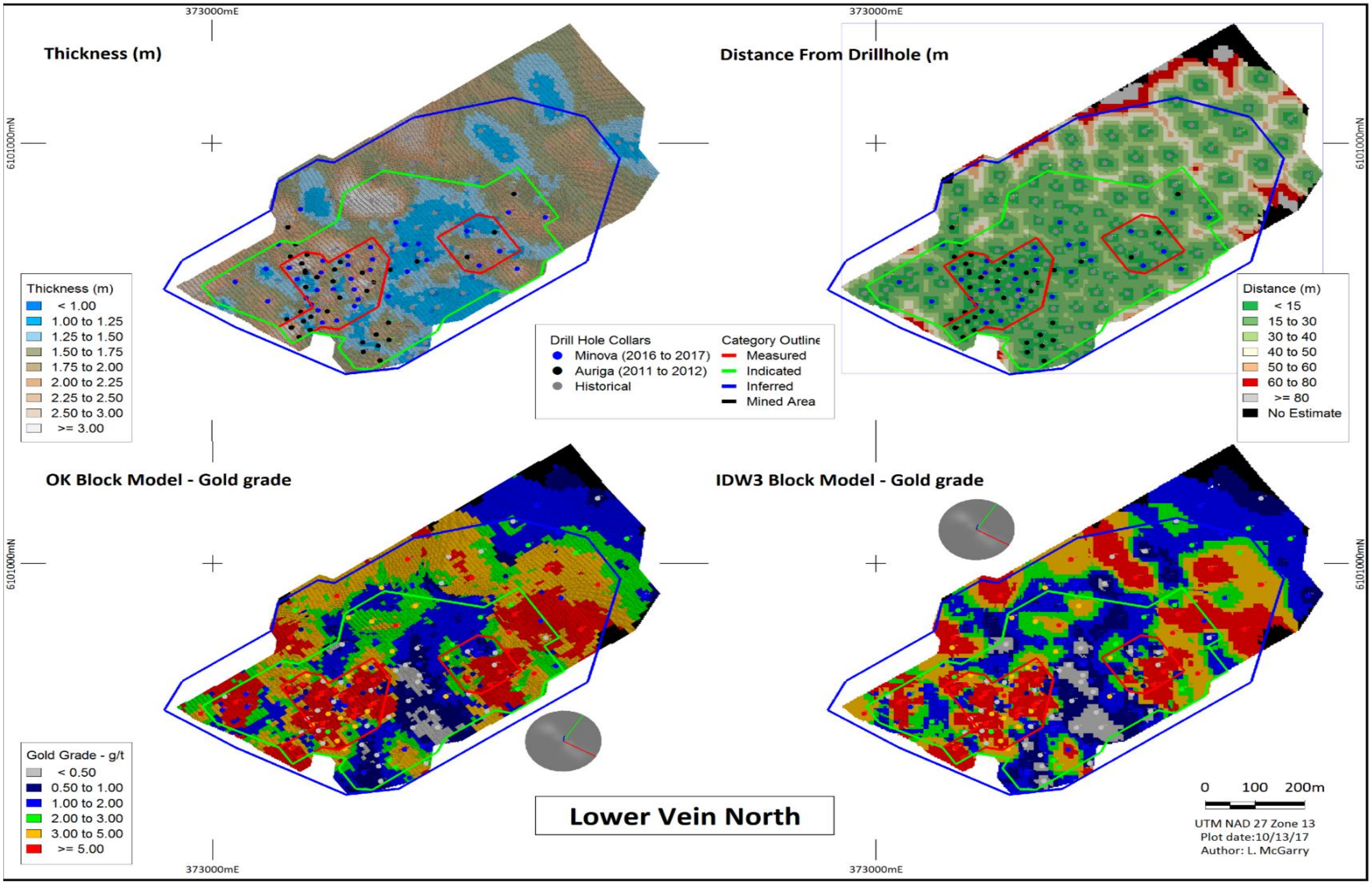


Figure 14.9 Plan view of the Lower Vein North

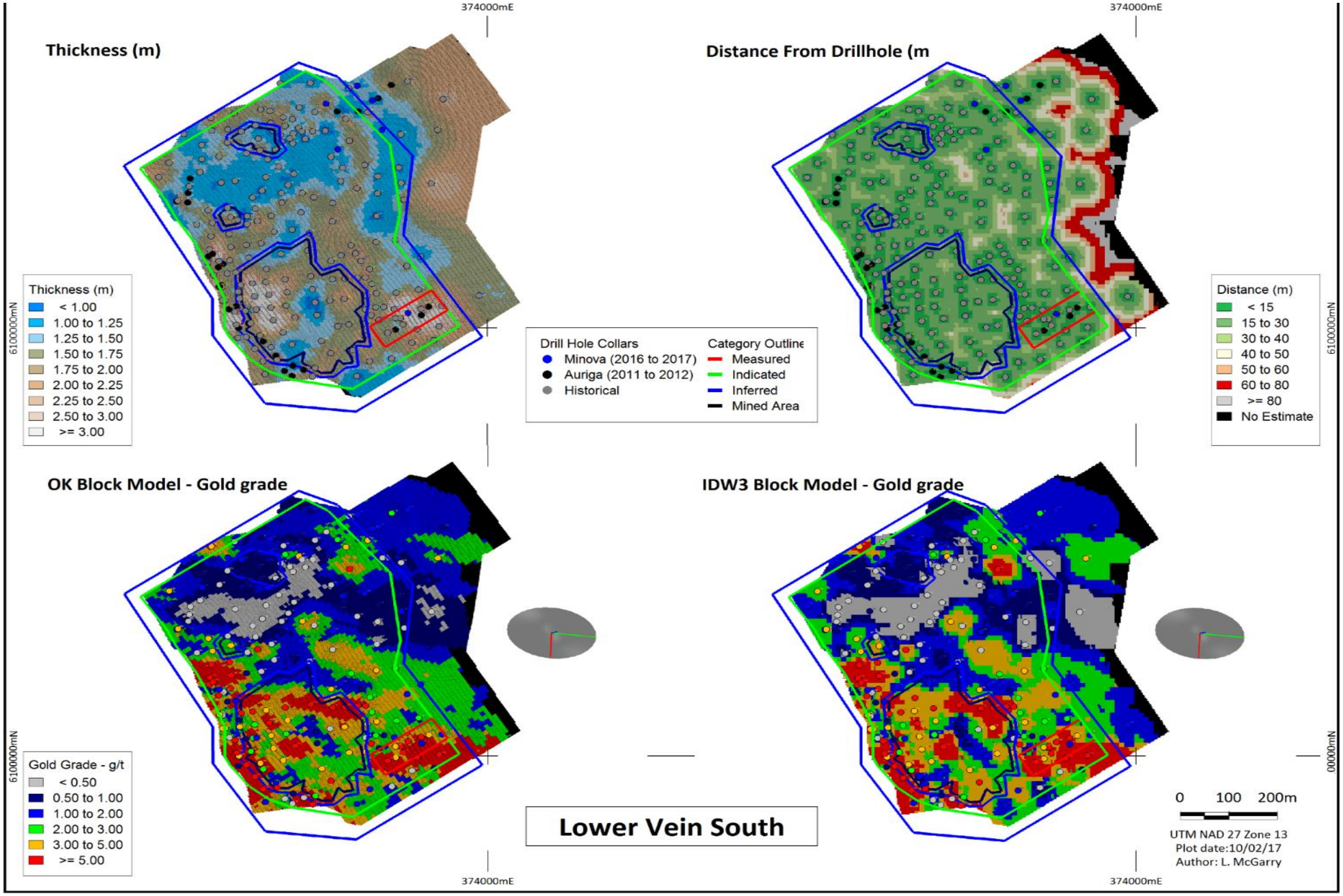


Figure 14.10 Plan views of the Lower Vein South

14.2.8 BLOCK MODEL

14.2.8.1 Block Model Construction

A block model was created to encompass the full extent of the PL Deposit using Micromine Version 2014™ software. Block model parameters are shown in Table 14.4 and block model attributes are shown in Table 14.5.

Table 14.4: Block Model Parameters			
Dimension	**X**	**Y**	**Z**
Origin	372750.8	6100846.0	274.5
Parent Block Size	10	10	2.5
Number of Sub blocks	4	4	4
Rotation Around Axis	55	25	0
Pivot Point Coordinate	373625	6100660	20

Table 14.5: Block Model Attributes	
Field	**Description**
EAST	Block centroid Easting
_EAST	Block dimension in Easting
NORTH	Block centroid Northing
_NORTH	Block dimension in Northing
RL	Block centroid RL
_RL	Block dimension in RL
Density	Bulk density assigned to the block used for tonnage estimates: Default density of 2.81
Domain	Domain the block is assigned to
THICK	True thickness of the modeled vein at the location of the block
IDWAuTC	Gold grade estimated by IDW3
OKAuTC	Gold grade estimated by Ordinary Kriging
Run	Interpolation run that informs the block grade
POINTS	Number of points that inform the block
Holes	Number of holes that inform the block
Avgdist	Average distance to samples that inform the block grade
Clstdist	Distance to closest sample that inform the block grade
AuFINAL	Final gold grade - Nlo and Slo domains receive grade by IDW3, all others by OK
Class	Block resource category: 1- Measured, 2 - Indicated; 3 - Inferred, 9 - Mined, 99-Unclassified

The block model used a parent cell size of 10 m E by 10 m N by 2.5 m RL with sub-celling to 2.5 m E by 2.5 m N by 0.625 m RL to maintain the resolution of the vein zones. The northing parent cell size was selected based on approximately one-third of the average drill section spacing in better drilled areas of the deposit. The model cell dimensions in other directions were selected to provide sufficient resolution to the block model in the across-strike and down-dip directions.

14.2.8.2 Assignment of Vein Thickness

For each domain, true thickness was assigned to the block model based on the distance between the modelled hanging wall and footwall surfaces derived from Leapfrog™. For a grid of nodes spaced at 10 m in the easting and northing direction, the minimum distance between modelled surfaces at each point was calculated. The calculated vein thickness was then applied to the model on a nearest neighbour basis.

The resultant block model file is populated with a modelled true thickness variable that can be used to constrain the resource. Example block models coloured by true thickness are displayed in Figure 14.9 and Figure 14.10 for the Lower Vein North and South.

14.2.9 Grade Estimation

14.2.9.1 Data Used

At the PL Deposit mineralization domain shell, contacts are hard boundaries for grade interpolation, such that gold grades in one domain cannot inform blocks in another domain.

14.2.9.2 Methodology

Both the Ordinary Kriging ("OK") method and Inverse Distance Weighting techniques are considered to be appropriate methods for estimating block grades at the PL Deposit where gold mineralization has a locally variable nature. In this scenario, the OK method and the utilization of a local mean within the search neighbourhood is preferred.

The OK interpolation utilised the variogram models contained in Table 14.3, and the search ellipse and sample constraint parameters detailed in Table 14.6.

Table 14.6: PL Deposit Estimation Parameters

Interpolation Run Number	**1**	**2**	**3**	**4**
Search Radii	1/3 range[1]	2/3 range[1]	Equal to the range in	1.5 × the range
Number of Sectors	1	1	1	1
Max no of Samples per Sector	12	12	12	12
Min Number of Drill Holes	2	2	2	2
Min Number of Samples per Hole	1	1	1	1
Max Number of Samples per Hole	5	5	5	5
Min number of Samples (Total)	4	4	4	2
Max number of Samples (Total)	12	12	12	12
Discretization	$3^{1}3^{1}3$	$3^{1}3^{1}3$	$3^{1}3^{1}3$	$3^{1}3^{1}3$

[1]Search ranges are defined by the semi-variogram models in Table 14.3

For validation purposes, an Inverse Distance Weighted interpolation was undertaken, whereby samples were weighted proportionally to the inverse of their distance from the block raised by a power of three

(IDW^3). The IDW^3 used the same search ellipse and sample constraint parameters as the OK interpolation.

Grades were interpolated in four passes. At 0.33, 0.66, 1.00, and 1.33 times the variogram ellipse range. Data used to interpolate grade into the PL Deposit block model contains locally clustered drill hole samples that may unduly influence or bias block grades. To address this issue, a restriction of 12 samples was applied that limits the maximum number of samples used to estimate block grades.

14.2.10 Model Validation

Validation of the grade estimates was completed by:

- Visual checks on screen in cross-section and plan view to ensure that block model grades honour the grade of sample composites;
- Comparison of sample and block grades;
- Generation of swath plots to compare input and output grades in a semi-local sense, by easting, northing, and elevation; and
- Investigation of the global change in support.

14.2.10.1 Visual Validation

A degree of smoothing is apparent but on the whole, block grades correlate very well with input sample grades. The distribution and tenor of grades in the composites is well honoured by the block model and is appropriate considering known levels of grade continuity and the variogram. Poorly informed veins are more smoothed but expected, considering the low number of data. Plan views of the South Lower and North Lower block models are shown in Figure 14.9 and Figure 14.10.

14.2.10.2 Comparison of Means

A check was conducted to test that the mean of the input data was close to the cell model mean.

The check compared the average gold grades at drill holes and for block model domains. The test demonstrated that the gold grades for the de-clustered mean input composites and both the OK and IDW block models are comparable as shown in Table 14.7.

Table 14.7: PL Deposit Comparison of Mean Gold Grades

Domain	N Points	Composite Mean	De-clustered Mean	OK Mean	IDW^3 Mean
SH	446.00	2.89	2.71	2.65	2.63
Mid	190.00	2.65	1.93	2.54	2.56
Nup	300.00	3.44	2.29	2.12	2.19
Nmn	289.00	2.42	2.02	2.76	2.93
Slo	425.00	3.47	2.71	2.86	2.84

Nlo	290.00	4.68	3.43	3.72	3.65

14.2.10.3 Swath Plots

Sectional validation plots compare the gold grades of composites (red line) and OK (black line), IDW^3 (grey line) that fall within 30 m easting and northing slices and 15 m elevation slices. The plot will identify slices that contain high-grade samples and low-grade blocks, or vice versa, which might indicate a problem with the estimation technique (Figure 14.11 and Figure 14.12).

For all domains, block grades estimated by OK and IDW^3 have a smoother profile relative to input samples. Where there are more samples, good agreement is seen between the trends of input composites and block grades estimated by each technique. The OK profile is slightly smoother than IDW. Both models reflect drill hole data on a local basis.

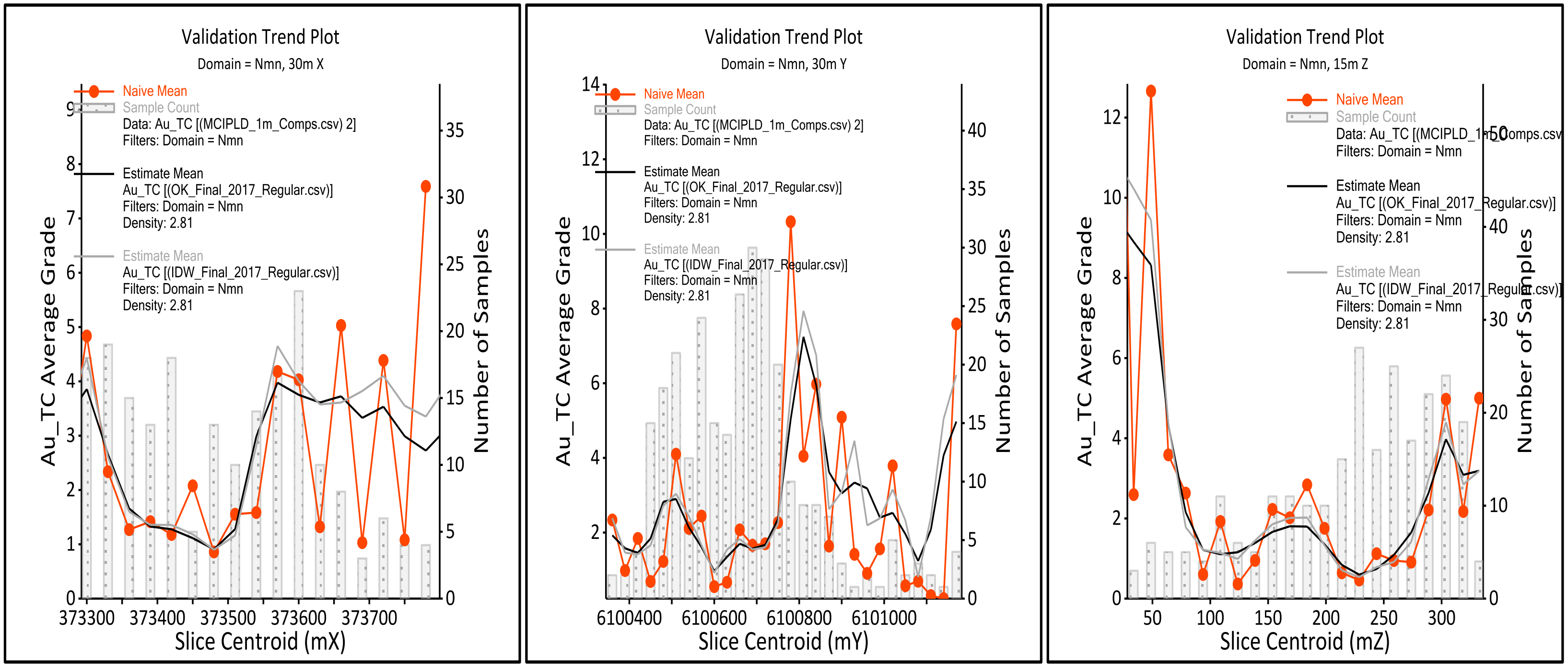


Figure 14.11 Swath plots for the Lower Vein North Domain

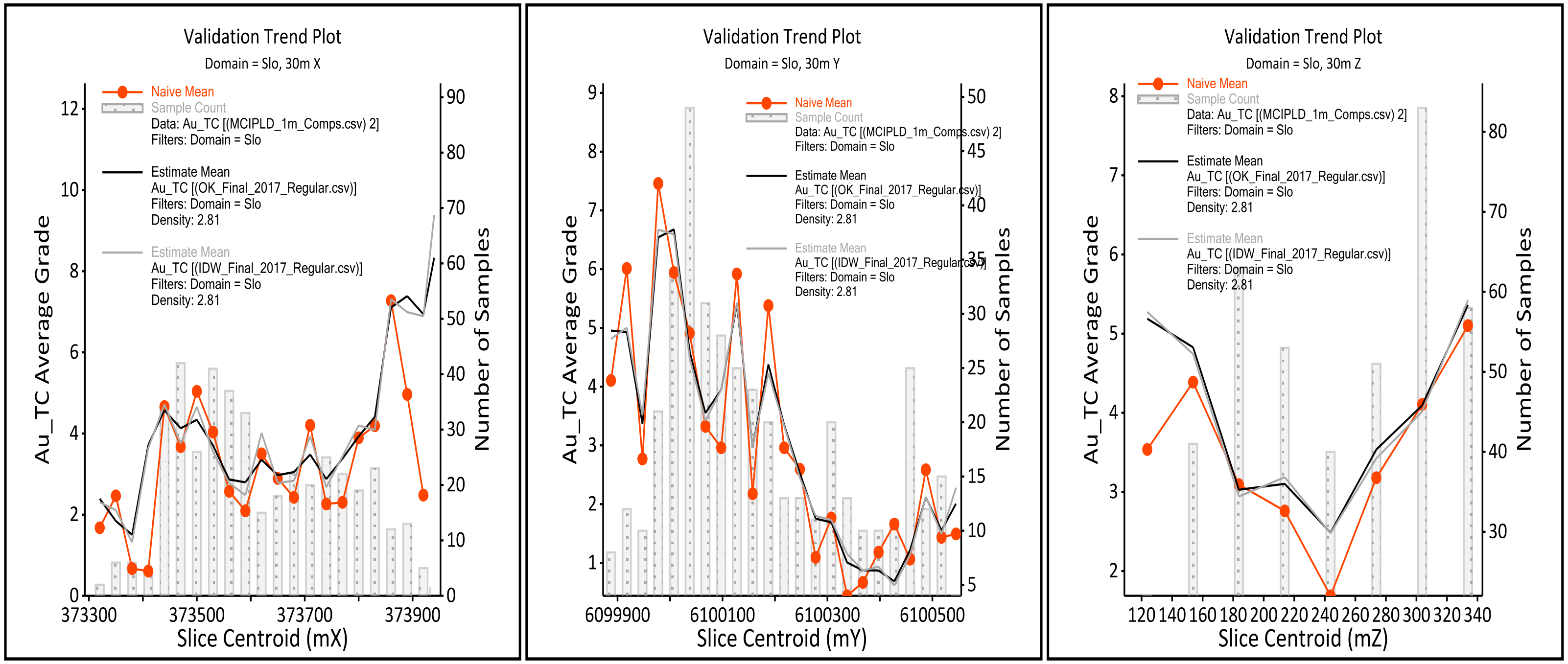


Figure 14.12 Swath plots for the Lower Vein South Domain

14.2.10.4 Global Change of Support

The Global Change of Support ("GCOS") assessment compares the estimated block model grade above a given cut-off, to the average grade of samples above the same cut-off. The average sample grade is adjusted to account for the decrease in gold grade and variability that is expected within the volume of the Smallest Mining Unit ("SMU") – *i.e.*, many meters cubed – relative the volume of a sample – *i.e.*, many centimetres cubed. The SMU grade is the theoretical target grade for block model estimates. Estimates were globally validated by comparing grade-tonnage curves for the theoretical SMU grads with OK estimates, IDW^3 estimates. Example grade tonnage curves are shown in Table 14.8.

Table 14.8: Global Change of Support for Gold Grades at a 2.5 g/t Cut-off					
Domain	**SMU**	**OK**	**% Difference**	**IDW**	**% Difference**
SH	4.62	4.23	-8.5%	5.01	8.4%
Mid	4.95	4.85	-2.1%	5.04	1.8%
Nup	5.22	5.24	0.5%	5.94	13.7%
Nmn	5.60	5.41	-3.3%	6.24	11.5%
Nlo	6.27	5.15	-17.8%	5.67	-9.6%
Slo	5.19	4.89	-5.7%	4.993	-3.8%

At a 2.5 g/t Au cut-off grade, the volume-variance correction factor between the de-clustered SMU grade (and the grade of the OK block model ranges from -11% for the Lower Vein North and 6% for the Upper Vein North. For the IDW^3 model, the correction factor ranges from -6% for the Lower Vein North and 18% for the Upper Vein North (Figure 14.13).

Except for the Lower Vein, the IDW^3 technique shows a bias toward overstating gold grades. At a 2.5 g/t Au cut-off, the OK estimation technique returns average block grades that are most similar to de-clustered SMU grades and are typically within ±10%. The Lower Vein North and South are the exception where OK results in an overly smoothed estimate and return grades 18% and 6% less than the de-clustered SMU grade.

For these domains, the IDW^3 appears to give an estimate that is within 10% of the expected grade. Based on this validation exercise, it may be appropriate to use the IDW^3 technique to estimate the grade of the Lower Vein North and South Domains.

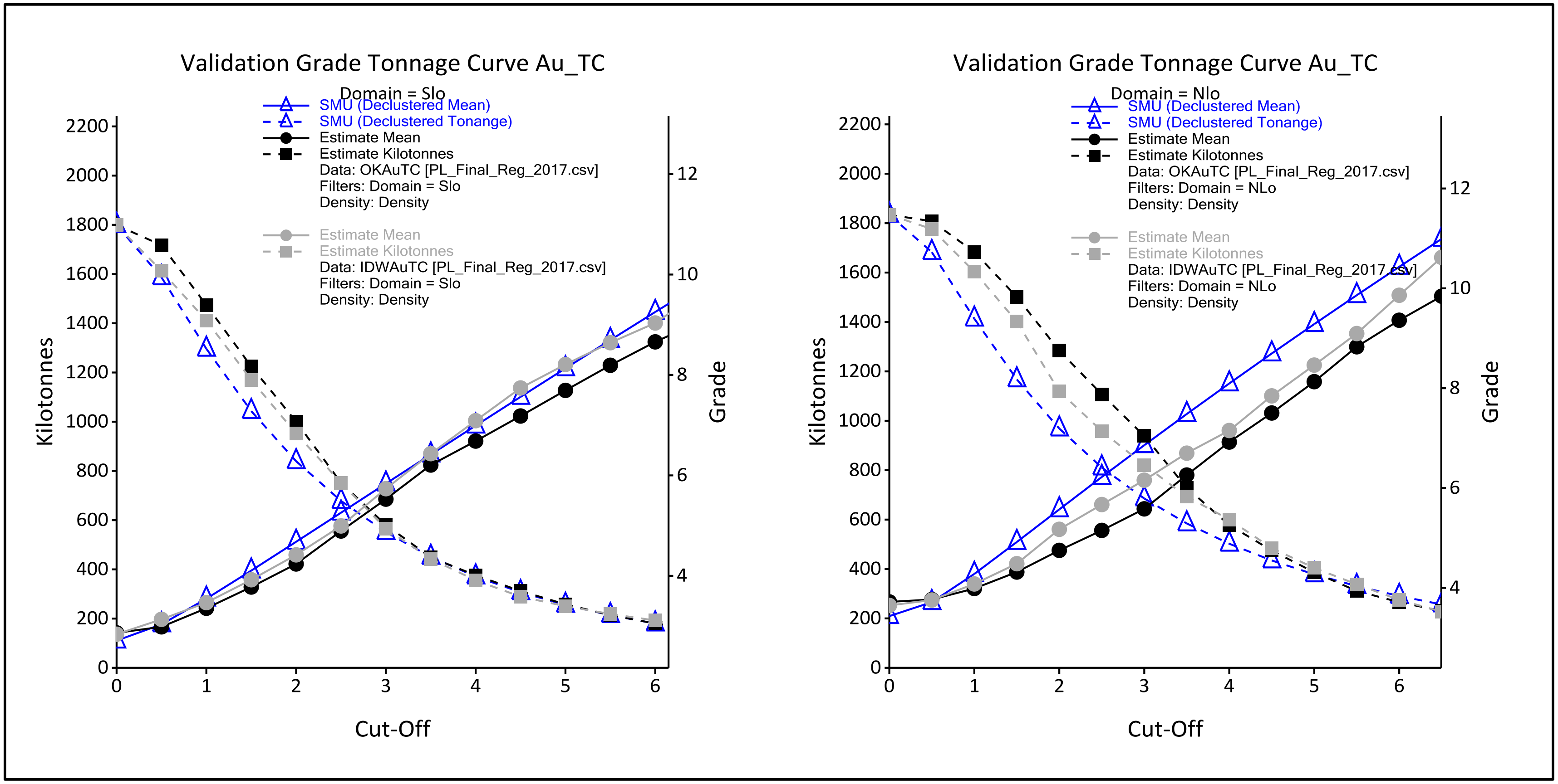


Figure 14.13 GCOS plots for the Lower Vein North and South

14.2.11 MINERAL RESOURCE CLASSIFICATION

The resource estimate is prepared in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014 where:

- **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." and,
- **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.

  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."

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 to the Indicated and Measured categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

Classification, or assigning a level of confidence to Mineral Resources, is undertaken in strict adherence to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Council, 2014). The Mineral Resource Estimate for the PL Deposit was prepared by Mr. Leon McGarry, CSA Senior Resource Geologist and Qualified Person for the reporting of Mineral Resources, as defined by NI 43-101.

14.2.11.1 Reasonable Prospects of Economic Extraction

CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014, require that resources have "reasonable prospects for economic extraction." This generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade taking into account possible extraction scenarios and processing recoveries.

- To define reasonable prospects of economic extraction the following equation was used:
    - Operating costs per ore tonne = $125/tonne
    - ($125)/[($1,250/oz./31.1035/0.80 FX × 95% Recovery] = 2.52 - Use 2.5 g/t.
    - This cut-off is in line with the cut-off grade used in the 2014 report.
- A minimum mining thickness of 1.5 m was selected.

14.2.11.2 Resource Classification Parameters

The MRE is classified in accordance with CIM Definition Standards for Mineral Resources and Mineral Reserves, adopted by the CIM Council on May 10, 2014. Resource classification parameters are based on the validity and robustness of input data and the QP's judgment with respect to the proximity of resource blocks to sample locations.

The following is taken into account when classifying resources at the Project:

- For the 2012 and 2017 drilling campaigns at PL, high core recoveries provide confidence that core samples and the assay values derived from them are representative of the material drilled and suitable for inclusion in resource estimation studies.
- Lithology domain and gold grade continuity are well established where drill density is greater than 30 m × 30 m; however, there remain portions of the deposit where sample density is insufficient to establish continuity beyond an Inferred level, specifically:
    - At depth, eastward of the western edge of Ragged Lake;
    - At depth, to the north and east of the EW limb of Ragged Lake; and
    - The northern edge of the PL Deposit, particularly the northern extension of the Lower North Vein.

- Unique down-the-hole survey, *i.e.*, azimuth and dip values that are different to the collar values, are not available for the majority of historical drilling. Auriga and Minnova drill holes allow modeling of vein intercepts with greater spatial accuracy.
- The estimation and modeling technique is considered robust.

Resource classification was undertaken using classification boundary strings assigned to the block model in a cookie cutter fashion. Strings were snapped to a grid and define a region of blocks that, on average, met the following criteria:

- **Measured Resources** are defined by Run 1 blocks that are within 30 m of Auriga or Minnova drill holes, within 30 m to the nearest hole, and with an average distance of 30 m from the block to informing samples. Measured Resources are only defined in relatively continuous portions of the deposit with higher geological confidence in three domains: the Upper Vein, Lower Vein North, and Lower Vein South.
- **Indicated Resources** are defined by Run 2 blocks that are within well-drilled areas of the deposit, where blocks are generally within 40 m of the nearest drill hole and have a reasonable geological continuity above a depth of 350 m below surface.
- **Inferred Resources** are defined by Runs 2 and 3 blocks within 60 m of a drill hole. Blocks within 10 m of underground workings are also classified as Inferred.

14.2.11.3 Mining Depletion

The deposit has been partially exploited using room and pillar mining (Pioneer, 1988-89). A three-dimensional model of underground workings in .dxf format was provided by Minnova. A polygon is used to outline the extent of underground workings in plan view. For the mined vein that intersects the underground workings, blocks that fall within the polygon outline are subtracted from the model using a 'cookie cutter' approach. A 10 m margin of Inferred classification blocks surround underground workings.

The mined area polygon outline is shown for the Lower Vein South Domain in Figure 14.10 along with a classification boundary defining a 10 m margin of Inferred blocks around the underground workings.

14.2.12 PL Deposit Mineral Resource Reporting

Resources are reported in adherence to National Instrument 43-101 Standards of Disclosure for Mineral Projects (Canadian Securities Administrators, 2011), and to the CIM Definition Standards on Minerals Resources and Reserves (CIM Council, 2014). The MRE is summarised by resource category in Table 14.9 and by resource domain in Table 14.10. The Mineral Resource has been reported above a 2.5 g/t Au cut-off grade and has an effective date of October 31, 2017.

Table 14.9: PL Deposit Mineral Resource Estimate as of October 31, 2017				
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 PL Deposit:

10. *The volume of the historical mined areas was depleted from the resource estimate.*
11. *Capping values range from 30 to 45 g/t Au and affected 16 samples*
12. *Bulk densities of 2.81 t/m^3 were used for tonnage calculations at the PL Deposit.*
13. *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%.*
14. *Tonnes and ounces have been rounded to reflect the relative accuracy of the mineral resource estimate; therefore numbers may not sum precisely.*
15. *1 troy ounce equals 31.10348 grams.*
16. *Mineral Resource tonnes quoted are not diluted.*
17. *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.*
18. *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.*

Table 14.10: PL Deposit Mineral Resource Estimate Reported by Domain as of October 31, 2017			
Domain	**Tonnes (kt)**	**Au Grade (g/t)**	**Contained Au (oz.)**
Measured			
Lower North	159	8.79	45,000
Sherridon	118	5.56	21,100
Upper North	86	8.45	23,300
Lower South	63	6.73	13,500
Measured Total	**425**	**7.53**	**102,900**
Indicated			
Lower South	262	5.30	44,700
Lower North	234	5.23	39,300
Sherridon	165	4.44	23,500
Middle	112	5.72	20,500
Upper North	105	4.33	14,600
Main	86	5.93	16,300
LL Center	30	8.43	8,000
Northern	21	4.82	3,300
South Upper	19	5.20	3,100
LL North	14	7.34	3,200
Center Mid	10	8.96	3,000
Indicated Total	**1,056**	**5.29**	**179,600**
Inferred			
Sherridon	560	4.39	79,000
Lower North	404	5.13	66,700
Main	264	6.26	53,100
Middle	250	5.13	41,300
Lower South	107	5.51	19,000
LL North	63	5.65	11,500
Northern	62	3.90	7,800
Upper North	43	5.27	7,300
LL Center	43	6.55	9,000
Lower Central	39	4.47	5,600
LL South	7	3.60	900
Center Mid	2	5.96	500
Inferred Total	**1,846**	**5.08**	**301,700**

14.2.13 Factors that May Affect the Mineral Resource

CSA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that could potentially affect this Mineral Resource estimate. The Mineral Resources may be affected by a future engineering feasibility study assessments of mining, processing, environmental, permitting, taxation, socio-economic, and other factors.

Additional technical factors which may affect the Mineral Resource estimates include:

- Gold price and valuation assumptions;
- Changes to the technical inputs used to estimate gold content (*e.g.*, bulk density estimation and grade model methodology);
- Geological interpretation (revision of vein models and the modeling of internal waste domains *e.g.*, dikes and structural offsets, such as faults and shear zones);
- Changes to geotechnical and mining assumptions, including the minimum mining thickness; or the application of alternative mining methods, such as open pit mining; and
- Changes to process plant recovery estimates, if the metallurgical recovery in certain domains is lesser or greater than currently assumed.

14.2.14 COMPARISON WITH PREVIOUS MINERAL RESOURCE ESTIMATES

Overall, the 2017 categorised model is one million tonnes smaller than the previous model detailed in Burga, et al., 2014. The smaller size results in fewer overall gold ounces. At a 2.5 g/t Au reporting cut-off, the change in resource model is mostly limited to the deeper, Inferred portions of the model.

For Inferred Resources: The 2014 MRE reported 2.16 Mt at 6.02 g/t Au for 419,000 ounces gold. The current 2017 MRE reports 1.85 Mt at 5.08 g/t Au for 301,700 gold.

The change in Inferred Resources is attributed to:

- A smaller model that is limited in extent to within approximately 60 m of a drill hole pierce point. The 2014 model includes blocks that were up to 130 m from a pierce point; and
- The inclusion of lower grade constraining intervals within the vein model to prevent the extrapolation of high-grade intervals over large distances

For Measured and Indicated Resources: The 2014 MRE reported 1.41 Mt at a grade of 6.07 g/t Au for 275,000 ounces gold. The current 2017 MRE reports 1.48 Mt at 5.93 g/t Au for 282,500 ounces gold. A change of +5% for tonnes, -2% for grade, and + 3% for gold ounces.

In a broad sense, the extent of Indicated and Inferred Resources is comparable to the 2014 MRE. However, there has been an increase in the Indicated footprint in the northern portion of the deposit where there is new drilling, but a significant decrease in Indicated Resources in the SLo Domain where blocks are depleted around historical workings.

There has been an increase in the proportion of Measured Resources. The previous, 2014 classification strategy was based on values generated during interpolation, where Measured category blocks were previously limited to those blocks informed in the first estimation run.

New drilling in the northern portion of the deposit has increased Measured category resources there. The extent of Measured Resources in the Sherridon Vein remains broadly the same. Scattered Measured

Resources in the South Lower domains are removed due to a reliance on historical drill data for those blocks and an uncertainty regarding the location of historical workings.

Category changes are generally coincident with an increased drill density, and consideration of underground workings. The new classification strategy does not represent a significant departure from previous approaches.

14.2.15 Targets for Resource Development

Targets for resource development within the immediate vicinity of the MRE model are listed below.

- At depth, eastward of the western edge of Ragged Lake:
  - Step out drilling from the thick Middle Vein intercepts in hole PUF-370 (from 348.50 m, 5 m at 4 g/t) and PUF-366 (from 397.00, 3 m at 8.48 g/t Au) will seek to expand the resource model and upgrade resource classification; and
  - Step out drilling to the east and northeast from the deep Sherridon Vein intercepts in hole PUF-360, PUF-346, and PUF-356 will seek to expand the vein model. Infill drilling between these holes will seek to upgrade resource classification in this portion of the deposit.

- To the north and east of the EW limb of Ragged Lake:
  - At depth, infill drilling within the Inferred portion of the Northern Lower domain, focusing on drill holes PUF-205, PUF-216, and PUF-226S, will seek to upgrade resources. Step out drilling to the west of these holes will seek to expand the resource model; and
  - Step out drilling northward from holes PUF-212 and PUF-267 will seek to expand the resource model northwards.

- To the south, in the Southern Lower Domain, step out drilling to the east of a high-grade intercept in hole PUF-297 (3 m at 25.33 g/t Au) where the deposit is open. Drilling will seek to expand the resource model in this area. Around this hole, infill drilling will target classification upgrades and expanding the extent of Indicated Resources southward.

To resolve postulated, steeply dipping fault offsets, infill and resource development drilling should include angled holes and the collection of orientated core measurements.

14.3 NOKOMIS MINERAL RESOURCE ESTIMATE

14.3.1 Introduction

All mineral resource estimation work reported herein was carried out by Mr. Leon McGarry, P.Geo., a Qualified Person under NI 43-101, using information and data supplied by Minnova. The effective date of the Nokomis MRE is April 15, 2014.

The Nokomis MRE is not considered in this Feasibility study and currently represents a small proportion of the Mineral Resource attributable to the Project. The following presents a summary of the 2014 MRE study, which was completed in 2014 using Micromine Version 2014™ software. The reader is referred to Burga, et al., 2014 for more details.

14.3.2 DATABASE

The Author relied on the following exploration data provided by Minnova (delivered to the Author on April 24, 2012) in the form of a data compilation CD containing a Gemcom™/Microsoft Access™ database file 'GD_Nokomis_UTM.mdb'.

- 19 Minnova diamond drill holes completed in 2012 (A4-01 to A4-19); and,
- 125 historic diamond drill holes completed between 1958 and 2005.

All drilling and sampling data for the Nokomis Mineral Resource model was compiled by the Author into a database comprised of 13,228 m of drilling from 144 drill hole records. Industry standard validation checks were completed on the database. No significant validation errors were identified, and the supplied database was deemed suitable for mineral resource estimation.

14.3.3 DOMAIN MODELING

The 2014 MRE Resource is hosted in the lithology unit logged as the 'Upper Host' (coded as '3' and 'MZ'). Interpretations were made in cross sections oriented perpendicular to the trend of the mineralization and spaced at 50, 25, or 12.5 m along the strike of the deposit. Strings are extended to half the drill hole spacing where the ore body is closed off by an unmineralized section. Where the deposit is not closed off by drilling, strings are extended approximately 50 m along strike and down dip. String outlines are joined up across strike to create three-dimensional wireframes representing the Upper Host Domain. Overburden and topographic surfaces are developed from drill hole logging and collar elevations, respectively. Surfaces are extended to cover the extent of the Upper Host Domain wireframes. The resulting domains were used as hard boundaries during estimation, and for rock coding, statistical analysis, and compositing limits.

14.3.4 EXPLORATORY DATA ANALYSIS

14.3.4.1 Gold Assays

The raw gold assay file was flagged so that each assay value was assigned to the mineralized domain it represented. Univariate statistical analysis was undertaken on assays within each domain.

Prior to interpolation, top-cut analysis was performed on raw assay data. A capping threshold of 50 g/t Au was identified. The capping value was applied to three assay values prior to estimation (Table 14.11).

Table 14.11: Nokomis Assay and Composite Summary Statistics			
	Domained Assays	Top Cut Assays (50 g/t Au)	Composited Assays (0.5 m)
Minimum	0.001	0.001	0.002
Maximum	451.6	50	50
Number of Points	586	586	732
Mean	4.10	3.27	2.80
Variance	411.91	46.53	32.56
Standard Development	20.30	6.82	5.71
CV	4.95	1.66	2.04

To generate representative length-weighted composites and honour lithological boundaries, a file was created with assay and lithology interval divisions. Assays and partial assays that fell within the Host Domain wireframes were coded by domain. To ensure equal sample support, domained assays were regularised to 0.5 m intervals, the dominant assay interval length at the Nokomis Deposit using length weighed averaging of gold grades. A nominal grade of 0.00 g/t was assigned to a small number of un-sampled assay intervals that fell within domain wireframes. Composites less than 0.3 m in length were discarded so as to not introduce a short sample bias into the estimation process.

Summary statistics for 586 composite samples from the Upper Host Domain is presented in Table 14.11. Also shown are raw assay and top cut assay summary statistics.

At Nokomis, experimental semi-variograms were constructed using 0.5 m regularised composites. The spatial continuity of the grade of composites in the Upper Host Domain is characterized by a 20% relative nugget effect, generally well-defined anisotropy with best continuity along the NNE-SSW strike direction. Experimental semi-variograms define the following ellipse dimensions (Table 14.12).

Table 14.12: Modeled Semi-Variogram Parameters for PL Deposit Gold Grade Interpolation										
Zone	Ellipise Rotation			Nugget Value		Model	Partial Sill	Range (m)		
	Z	Y	x					Major	Semi-Major	Minor
UH	40	35	10	0.70	20%	1. Spher	4.27	70	30	12

14.3.4.2 Bulk Density

The specific gravity measurements of seventeen samples collected by Minnova range from 2.62-3.36 g/cm^3, with a length weighted average of 2.89 g/cm^3, the value used for Mineral Resource estimation.

14.3.5 Block Model

A rotated empty block model was created to cover the extents of resource domain wireframes at Nokomis. A block size of 2 m × 2 m × 2 m was selected to reflect the generally narrow widths of the mineralized zones. Block model parameters are presented in Table 14.13. Model blocks are rotated by

30 degrees to honour the trend of the host zone. The block model file includes separate fields for estimated grade, rock code, density, and classification attributes.

Table 14.13: Nokomis Upper Host Block Model Definition					
Direction	**Model Origin (m)**	**Model Limit (m)**	**Model Extent (m)**	**Block Size (m)**	**Number of Blocks**
Easting	379,800	380,400	440	2	220
Northing	6,103,700	6,104,300	740	2	370
Elevation	0.00	500	304	2	152

14.3.6 Nokomis Mineral Resource Estimation

Block model gold grades were estimated using domained, top-cut, and composited drill samples. For the Upper Host Domain, interpolation by OK was undertaken in sequential runs of increasing search volume and varying minimum sample requirements until all blocks within each domain received an interpolated grade or were assigned a null value.

- Run 1 is equal to half the range in main directions and uses a minimum of 3 drill holes and 6 samples;
- Run 2 is equal to the range in main directions and uses a minimum of 2 drill holes and 4 samples; and
- Run 3 is equal to the range in main directions and uses a minimum of 1 drill hole and 2 samples.

Once grade interpolation was complete, the block model was validated by way of visual inspection of successive section lines and by swath plots at 50 m interval slices in the easting direction.

14.3.7 Nokomis Mineral Resource Reporting

Classification, or assigning a level of confidence to Mineral Resources, has been undertaken in adherence to the CIM Definition Standards for Mineral Resources and Mineral Reserves. This resource estimate was prepared in 2014 by Mr. Leon McGarry, who at that time was a Project Geologist and Qualified Person (QP) working at ACA Howe. Mr. McGarry visited the Nokomis Property between January 20-21, 2014 to review the geology, collect verification samples, and confirm the location of drill collars. The Author is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that may materially affect the Nokomis Mineral Resource estimates.

The Author considers that portions of the Nokomis Deposit have a reasonable prospect for open pit extraction. The resource is constrained to within the degree of confidence in the reported resources. Resources are classified based on the validity and robustness of input data and the proximity of resource blocks to sample locations.

Indicated Resources are identified where the average sample distance (recorded as a block parameter during grade estimation) was less than, or equal to, approximately 33-66 m. At least nine samples from three holes (minimum three samples from each hole) were required for a block to be assigned to this category. A polygon was manually created to identify 'Indicated' category resource. The polygon is created to encompass blocks that are informed by at least two drill holes within a search ellipse equal to the variogram ranges. The polygon does not include blocks that are within 10 m of the interpreted Pegmatite Dyke Domain or overburden surface. The extent of Indicated blocks is shown in Figure 14.14 and Figure 14.15.

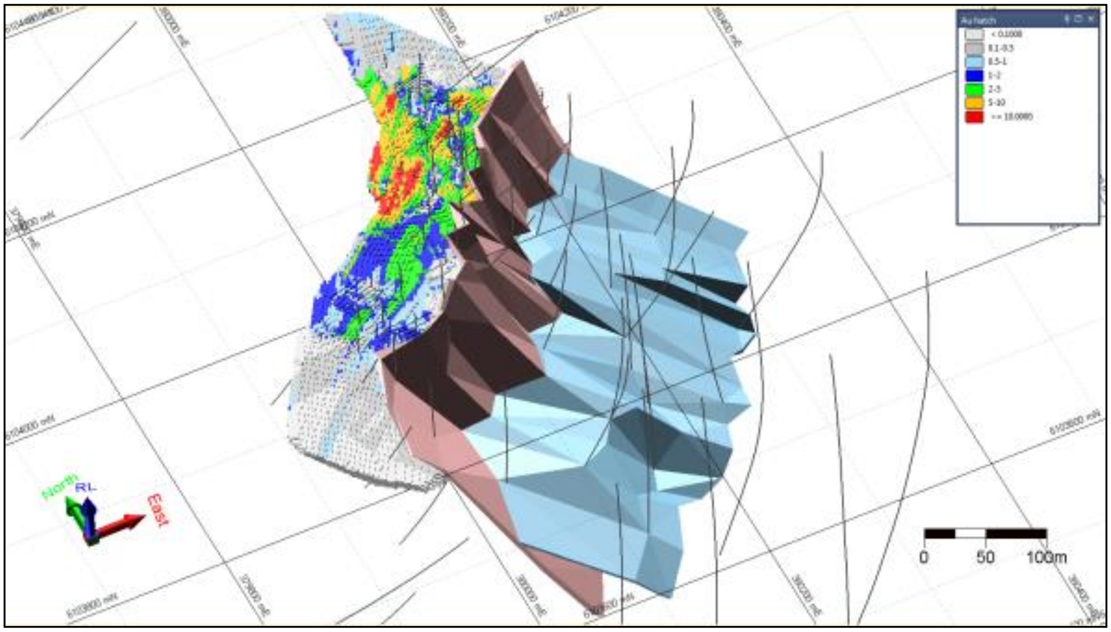


Figure 14.14 Nokomis Upper Host Domain resource blocks colored by gold grade (g/t Au)

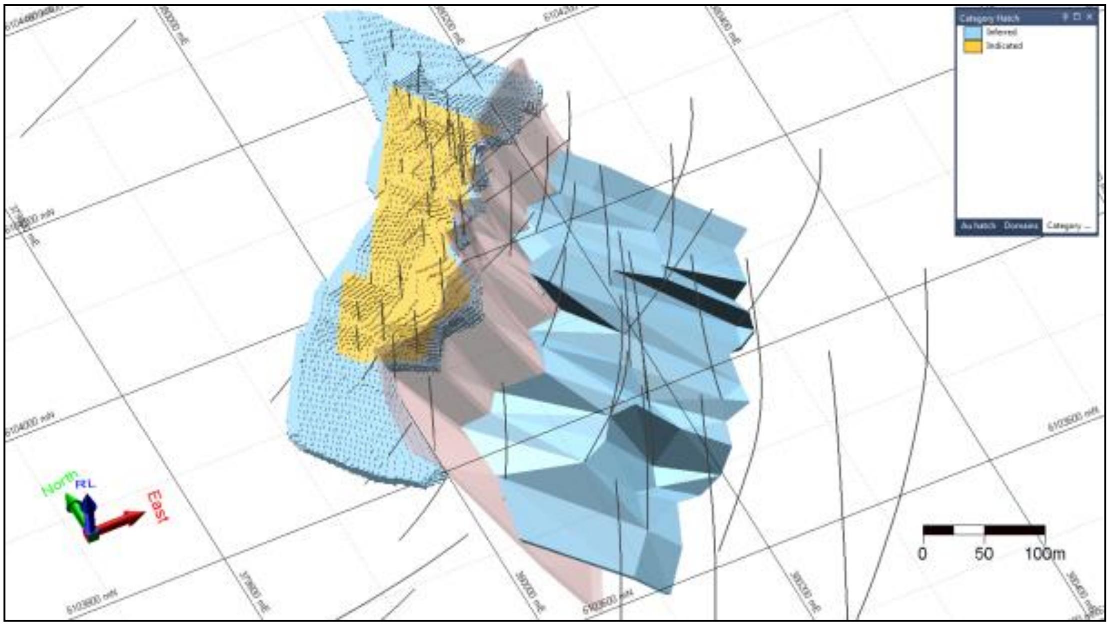


Figure 14.15 Nokomis Upper Host Domain resource blocks colored by resource category

No Measured Resources are identified at the Nokomis Property.

Table 14.14 presents a summary of total Inferred Resources attributable to Minnova's Nokomis Property. The effective date for the Nokomis Mineral Resource estimate is April 17, 2014.

Table 14.14: Nokomis Mineral Resource Estimate Statement

Class	Reporting Cut-off (g/t Au)	Tonnes	Grade (g/t Au)	Contained Au (oz.)
Indicated	0.6	371,000	3.41	41,000
Inferred	0.6	247,000	2.41	19,000

Notes:

3. A block reporting cut-off value of 0.6 g/t Au was applied to all resource blocks.
4. Tonnes and ounces have been rounded to reflect the relative accuracy of the Mineral Resource estimate; therefore, numbers may not sum precisely.
5. Mineral Resources were calculated with commercial mining software. Drill holes traces showing lithology and gold grade were reviewed in plan and cross section. Wireframe constrained block model grade interpolation was undertaken using Ordinary Kriging (OK).
6. The resource estimate was prepared by Mr. Leon McGarry, B.Sc. P.Geo, Howe Project Geologist.
7. A default average specific gravity (SG) value of 2.89 has been used.
8. No dilution factor has been applied to Mineral Resource tonnes.
9. No Measured Resources or Mineral Reserves of any category are identified.
10. 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. 11. 1 troy ounce equals 31.10348 grams.

15 MINERAL RESERVE ESTIMATES

Both the open pit and underground Mineral Reserves have been developed using best practices in accordance with CIM guidelines and National Instrument 43-101 reporting. The effective date of the Mineral Reserve estimate is October 31, 2017.

The Mineral Reserves were derived from the Mineral Resource block model that was presented in Section 14. The Mineral Reserves are the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining loses and the addition of waste dilution. The Mineral Reserves form the basis for the mine plan presented in Section 16.

15.1 MINING SEQUENCE

The PL Deposit will be primarily mined by underground mining techniques for the life-of-mine. During Years 2 to 5, small open pits on surface will supplement underground ore. The Mineral Reserves are estimated for the two (2) methods and presented below.

Details of the selection method for open pit and underground operations are given in Section 16 of this Report.

15.2 UNDERGROUND MINING RESERVES

The underground mine will be accessed via a mine portal and main ramp that will connect mine levels on the footwall side of the orebody. The sub-levels will be spaced at every 35 m vertical elevation. The deposit will be mined using an up dip stoping method utilising Alimak and slusher units for the mining of the stopes due to the approximate 30 degree dip of the ore zone with a mining rate of approximately 590 tpd.

15.2.1 CUT-OFF GRADE

The underground cut-off grade for stopes is based on operating costs for mining, processing, tailings, surface, and G&A costs, which total $184.85 per tonne, as derived in the sections that follow. The underground cut-off grade of 4.0 grams Au per tonne is calculated as follows:

Cut-off Grade = Total Mine Operating Cost / [Gold Price US$ × US:CAD FX × Gold Payability × TCRC Cost × Process Recovery) / 31.1035]
= 184.85 / [1250 × 1.25 × 99.75% × $5/oz. × 90% / 31.1035]
= 3.96 Use 4.0 grams Au/tonne

15.2.2 MINE DILUTION AND ORE LOSS

Based on the proposed mining method, a minimum mining height (ore thickness) of 1.5 m has been determined, which allows for breaking of the stope rock and slushing of the broken ore out of the stope.

This minimum mining height has been applied to the resource calculations and included in the reported Measured and Indicated Resources for reserves determination.

Additional mining dilution was added based on the mining height of the stope. Different dilution factors were applied for stopes of the minimum 1.5 m height, stopes with a mining height between 1.5-2.4 m, and stopes with heights greater than 2.4 m. The 2.4 m height increment is based on this being the height of the raise that the Alimak will develop in the stope and the raise may contain all waste or ore or a combination of ore and waste, which will cause different overall stope dilution to occur. The different dilution factors are determined as follows.

15.2.2.1 Stopes of Minimum Mining Height – 1.5 m

Dilution in the stope is the volume of the raise greater than the minimum mining height added at zero grade to the total ore tonnes contained in a stope at the minimum mining height.

- Raise 3 m wide by 2.4 m high Stope 20 m wide
- Raise Height Dilution = 2.4 - 1.5 = 0.9 m
- Total Stope Dilution = 0.9 × 3/20 × 1.5 × 100 = 9%

15.2.2.2 Stopes of Height – 1.5-2.4 m

Dilution in the stope is the volume of the raise greater than the ore height plus a waste overbreak allowance of 0.3 m in the hanging wall above the ore contact (over the width and length of the stope) both added at zero grade to the total undiluted ore tonnes contained in a stope. This dilution averages 15%.

15.2.2.3 Stopes of Height Greater than 2.4 m

Dilution in the stope is 0.3 m waste overbreak allowance in the hanging wall above the ore contact (over the width and length of the stope) at zero grade included in the undiluted tonnes of ore in a stope. This dilution averages 8%.

The overall weighted average dilution of reserves is approximately 13%.

15.2.3 Underground Reserves Statement

Based on the underground mining cut-off and mine plan presented in Section 16, the total underground reserves are 0.95 million tonnes at a grade of 6.99 g/t Au. A breakdown of the underground reserves categories is presented in Table 15.1.

Table 15.1: PL Mine Project Underground Reserve Statement		
Category (g Au/t)	**Tonnes**	**Grade**
Proven	367,458	7.77
Probable	585,269	6.52
Total Reserves	**952,727**	**7.00**

15.3 OPEN PIT MINING RESERVES

The open pit cut-off grade is based on surface mine operating costs, processing, tailings, surface, and G&A costs, which total $73.17 per tonne, as derived in the sections that follow. The open pit cut-off grade of 1.6 g/t Au is calculated as follows:

Cut-off Grade = Total Mine Operating Cost / [Gold Price $US × US:CAD FX × Gold Payability × TCRC Cost × Process Recovery) / 31.1035]
= 73.17 / [1250 × 1.25 × 99.75% × $5/oz. × 90% / 31.1035]
= 1.60 Use 1.6 g/t Au

15.3.1 OPEN PIT OPTIMIZATION

Parameter	Value
Au Price	US$1,250/oz.
Au Recovery	90%
Mineralized Material Mining Cost	$3.75/tonne mined
Waste Rock Mining Cost	$3.00/tonne mined
Process Cost	$24/tonne milled
G&A Cost	$6/tonne milled
Pit Slopes	50°

15.3.2 DILUTION

Dilution of 20% at zero grade has been included in the open pit reserves.

15.3.3 OPEN PIT RESERVES STATEMENT

Based on the open pit cut-off grade, optimization, and mine plan presented in Section 16, the total open pit reserves are 0.3 million tonnes at a grade of 4.35 g/t Au. A breakdown of the open pit reserves is presented in Table 15.2.

Table 15.2: PL Mine Project Open Pit Reserves Statement		
Category	**Tonnes**	**Grade (g/t Au)**
Proven	87,145	3.99

Probable	226,566	4.49
Total Reserves	**313,711**	**4.35**

15.4 COMBINED UNDERGROUND AND OPEN PIT RESERVES

The total underground and open pit reserves are 1,266,438 at a grade of 6.34 g/t Au.

16 MINING METHODS

16.1 INTRODUCTION

The PL Deposit of the Maverick Project will initially be mined by underground mining techniques as the environmental permits for this type of mining are already in effect and valid. As the PL Deposit extends to the near surface, a number of small open pits will also be developed, but environmental permitting of these operations is required. Open pit mining at lower costs, will supplement PL's underground-sourced processing plant feed.

The open pit permitting could require up to one year. After the underground mine has started mining, production from the open pits will be considered extensions of the existing operation and not subject to a federal government environmental review. The permitting and subsequent development period for the open pits has been allowed for in the overall Project mine production schedule.

Underground production will provide processing plant feed for the first 2 years. A combination of open pit and underground production will provide feed for the processing plant, while the open pits are in operation. Following completion of open pit mining, the processing plant feed will again originate exclusively from underground mine production.

16.2 EXISTING INFRASTRUCTURE

The previous mining operation in 1987 through 1989 developed a portal ramp to an approximate vertical depth of 130 m and a number of mining levels and stopes. These workings are presently flooded to the surface.

16.3 EXISTING MINE OPENINGS

The use of the old mine workings to develop stopes and begin mining from the old mine was assessed. The time and costs to dewater and rehabilitate the old workings and provide newly developed access to potential new mining areas is estimated to be approximately similar to the proposed plan. The rehabilitation option risk is higher because of the unknowns associated with rehabilitating the old mine workings. To ensure mining commenced in areas where no mining appears to have taken place before, would require access to the 110 Level. To reach this level near the bottom of the old mine would require:

- All old mine openings to be dewatered;
- The complete length of the existing ramp to be rehabilitated;
- Old stopes near the ramp (if exist) be stabilized, if required;
- A portion of the 110 Level to be rehabilitated to access areas where potential stopes are located; and
- Additional new lateral development to access mining areas, as required.

The reserves and mineable resources accessed would be of lower quantity and grade than the proposed new mining areas. To move this ore to the surface from the lower level would be more costly and would also require extra capital and operating costs for an extra 30-tonne haul truck to meet daily mine production.

This option was eliminated because of the uncertainties, higher risk, and potential unforeseen schedule and costs overruns.

A new ramp will be established from the dewatered top section of the existing ramp to access new mining areas to the north of the old mine. The proposed plan of a new ramp in the north of the deposit allows time to examine the old mining areas and determine conditions as well as properly locate the ore remnants or new stoping blocks.

Prior to new mine development commencing, the existing ramp would be dewatered for approximately 150 m in length. A hole will be drilled in the cover of the vent raise just north of the portal and a submersible pump would be lowered down into the water. A trailing flexible rubber pipe would be connected to the pump and discharged either directly to the TMF, if all permitting is in place, or to a tank for treatment. If the TMF is not fully permitted, treatment is required, due to the heavy metal contents that have built up in the mine water. This treated water will then be discharged to the TMF. As the water is pumped out, rehab crews would move down the ramp and proceed with the rehabilitation of the heading and installation of new mine services (compressed air line, service water line, dewatering line, electrical, blasting, and communications cabling) carried out in the dewatered section of the ramp.

16.4 SURFACE MINE DESIGN

The underground mine design will be impacted by the potential portion of the PL Deposit mineable by open pit(s). Open pit mining will not commence until Year 2 to allow time for permitting of on-site open pit mining. The first step in mine design was to determine the potential top of the underground mining blocks. The PL pits were limited to a bottom elevation of 30-40 m depth (300 m true elevation). Underground stopes located below the open pits would be mined and backfilled prior to mining of the open pits and a small pillar left in the floor of the final pit above the backfilled open stopes. Using the geology block model, open pit optimization was run on the PL Deposit using the parameters shown in Table 16.1.

Table 16.1: PL Optimization Parameters	
Parameter	**Value**
Gold Price	US$1,250 per ounce
Exchange Rate	US$1:C$1.06
Stripping, Overburden, and Muskeg	$9.20 per tonne
SG:	
Overburden	2.0
Rock	2.81
Mining Cost	
Mineralized Rock	$5.50 per tonne
Waste Rock	$4.50 per tonne
Processing Cost	$22 per tonne
Milling Rate	750 tpd
Milling Recovery	90%
Hauling Mill Feed	$2 per tonne
General and Administration (G&A)	$10.34 per tonne
Pit Slope	50°
Dilution – Planned	1.5 m Minimum Modelling Width
Dilution – Non-Planned	20%
Mining Losses	10%
Reclamation Cost, Ore, and Waste	$0.50 per tonne
Selling Costs	$26 per ounce
Elevation (Depth) Limit	300 m (40-50 m)

The open pit optimization results, Measured and Indicated Resources, are as presented in Section 15, tonnes at a grade of grams Au per tonne. The open pit results were used to develop the open pit mine plan and production schedule.

16.5 UNDERGROUND MINE DESIGN

The known mineralization at the PL Deposit consists of five parallel gold-bearing veins that strike N30°W and dip at approximately 30 degrees to the northeast. The zones are designated from top to bottom, as the Sherridon, Upper, Main, Lower, and Lower Two Zones. The mineralized veins are tens of centimetres thick and would be mined in units. Mining heights would be from the minimum mining height of 1.5 m to upwards of approximately 2-3 m.

The old portal, existing ramp, and the new ramp extension continuing from the existing ramp will provide the main access to the mine. As the ramp progresses downwards vertically, mining levels would be established at required elevation intervals. All men, equipment, and materials will be transported into and from the mine via this main ramp. All ore and waste (as required only) will be transported in underground diesel powered haul trucks operating in the underground drifts and the main ramp.

On each level, the mining areas would be accessed from the main ramp by a 4.0 m high by 5.5 m wide access drift driven in the ore zone and parallel to the strike of the ore zones.). The wider access drift would allow for material slushed from the stopes to sit on the hanging wall side of the drift while still allowing equipment access along the footwall side of the drift. The proposed mining method is up dip panel stoping utilizing Alimaks. Stoping will take place in panels, which are nominally 20 m wide (along strike) and extend lengthwise up dip over vertical intervals of approximately 35 m (producing stopes of 60-70 m length).

A pilot raise for drilling in the middle of each stope will also facilitate sufficient amounts of hanging wall ground support to prevent caving of the hanging wall and resulting ore losses. Each pilot raise will be driven 10 m beyond the up dip access drive to allow for easier Alimak set-ups for the next lift of mining. Stopes are mined the full height of the ore between levels, before backfilling. This results in optimum productivities, lower mining costs, and the minimum number of operating stopes to meet daily production targets. Mined out areas will be backfilled with primarily cemented and uncemented hydraulic backfill.

Underground development, including excavation of ramps, accesses, and haulage drifts, will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos, load-haul-dump (LHD) units, 30-40-tonne haul trucks, and scissor lifts with work platforms. Mining will utilise diesel powered, track and rubber tired mobile equipment including a single boom extension rods drill, small ANFO loading units, LHDs, and haul trucks.

Open pit mining by a contractor will utilise relatively smaller diesel operated blast hole drills, hydraulic excavators, haul trucks, and support services equipment.

Underground mining of the mineralized zones will be at a proposed rate of approximately 590 tpd or 216,000 tpy. After the first year of production, processing plant feed will be supplemented by open pit production of approximately 400 tpd with yearly production of from approximately 68,000-111,000 tpy, over 4 years. After open pit mining is completed, the mine feed will be supplied solely from the underground mine, at 590 tpd for the remainder of the mine life.

16.6 GEOTECHNICAL CONSIDERATIONS

Geotechnical design is based on previous feasibility study work and practices during the past operation and where known, has been modified or improved to present standards and practices for this type of mining in the expected ground conditions. No new geotechnical information was generated or laboratory testing performed for this present study.

16.6.1 EXISTING OPENINGS

Current development would be check-scaled to remove smaller, loose rock and ground support installed in all areas. Ground support will employ resin grouted rebar on a 1.2 m by 1.2 m pattern and welded wire mesh screen (1.2 m by 2.4 m sheet with 5.6 mm wire thickness, 100 mm × 100 mm apertures) on the backs and walls to within 1.5 m of the floor on the walls. Screen sheets will be installed with 0.2 m overlap.

16.6.2 New Lateral Development

Lateral development will be supported with 1.8 m long resin grouted rebar on a 1.2 m by 1.2 m pattern and welded wire mesh screen (1.2 m by 2.4 m sheet with 5.6 mm wire thickness, 100 mm × 100 mm apertures) on the backs and walls to within 1.5 m of the floor on the walls. Screen sheets will be installed with 0.2 m overlap.

16.6.3 Stoping

The pilot raise will be supported with 1.8 m long resin grouted rebar on a 1.2 m by 1.2 m pattern and welded wire mesh screen (1.2 m by 2.4 m sheet with 5.6 mm wire thickness, 100 mm × 100 mm apertures) on the backs and 1.2 m rebar and screen on the walls. Screen sheets will be installed with a 0.2 m overlap.

The length of stopes have been established using an allowable hydraulic radius (open stope area divided by perimeter) that depends on the rock quality and using an empirical design method. If the stopes were to remain open after mining, then sill pillars and rib pillars would be required to prevent the collapse of the hanging wall, but significant ore would be left unmined. To minimize pillars and prevent the possibility of caving, stopes will be backfilled.

The following geotechnical design criterion has been used for the stopes at the PL Deposit. The ore thickness ranges from 1.0-3.0 m high with an average of approximately 2 m. Stope widths are planned at 20 m along strike and stope lengths up dip of approximately 70 m (based on the level spacing of 35 m vertically and orebody dip of approximately 30 degrees). Stope lengths have been established using an allowable hydraulic radius.

To ensure back stability while mining of a stope, the pilot raise would also have fully grouted (cement) 8.5 m twin-strand bulbed, 15.2 mm cable bolts on a 2.5 m pattern installed into the back of the stope in a narrow fan pattern. Cables should be tensioned and installed with plates.

More geotechnical drilling will be conducted at the PL Deposit to improve rock quality data along strike and at depth and aid in optimizing stope geometry and support requirements.

16.7 MINE ACCESS AND LEVEL DEVELOPMENT

16.7.1 **Main** Access Ramp

The existing portal and 100 m section of the existing ramp will be used to access new mining areas. A new ramp will be extended laterally and in vertical extent to access the new areas to be mined. The ramp is presently located in the footwall of the orebody and new ramp development will continue in the footwall with the ramp direction plunging at a similar dip to the orebody (30 degrees). The ramp will have dimensions of 4.5 m wide by 4.5 m high (to accommodate travel of the largest mobile mining equipment) and all new ramping will have a gradient of 15%. Figure 16.1 presents the existing and proposed development for the mine as well as the potential mining blocks.

After a 100 m section of the existing ramp has been dewatered and rehabilitated, new ramping will initially be developed down to the 75 m vertical elevation at a grade of -15%. From the 75 m level, the ramp will be further extended to depth.

This ramp system will facilitate entry and exit for men, equipment, and materials to the mine and for the transport, by underground haul trucks, of ore and waste from the mine.

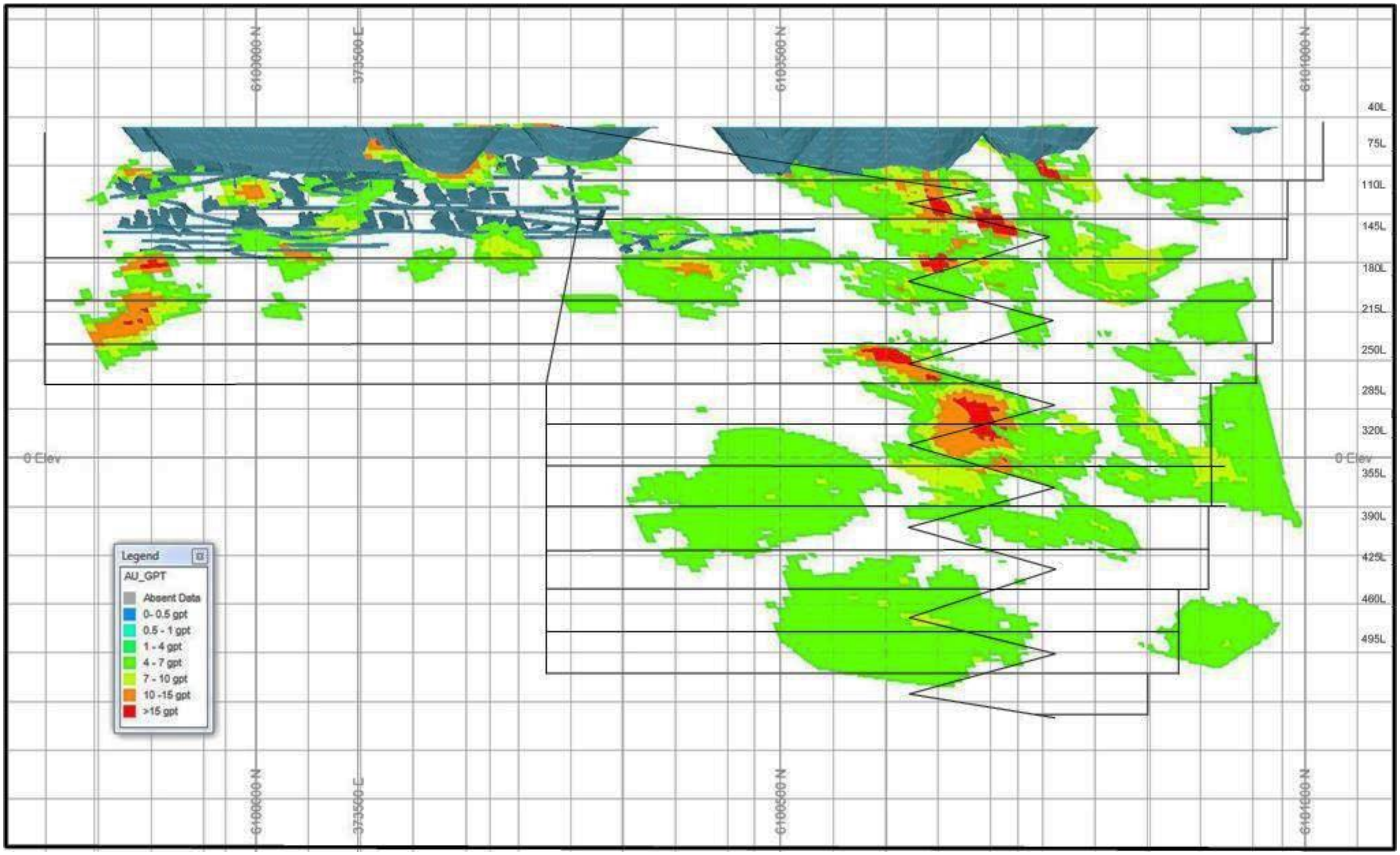


Figure 16.1 Long section showing proposed ramp and level development

16.7.2 LEVEL DEVELOPMENT

The level access drifts will be developed in ore at 5.5 m wide by 4.0 m high at a +3% gradient from the ramp.

New development headings will be drilled off using 44 mm diameter drill holes, with 4 cut holes reamed. Slashes will be drilled off to the required width and height using 44 mm diameter drill holes as well. Headings will be blasted using a combination of ANFO, stick emulsion for the lifters, and perimeter blasting products for the wall and back holes and nonel caps, initiated by electric caps. Ground support will consist of 1.8 m long resin grouted rebar, installed on a 1.2 m by 1.2 m pattern and welded wire mesh screen, on the backs and walls to within 1.5 m of the floor. The ground support will be installed using scissor lift units. All services will be installed from scissor lifts.

Services installed in the ramp and level access drives will be 152 mm airline, 102 mm service waterline, 102 mm discharge water line, 102 mm backfill line, 500 MCM power cable, 600 and 120 volt cable, 48-fibre fibre optic data and communication cable, and a central blasting line. Figure 16.2 presents an opening cross section including the services configuration.

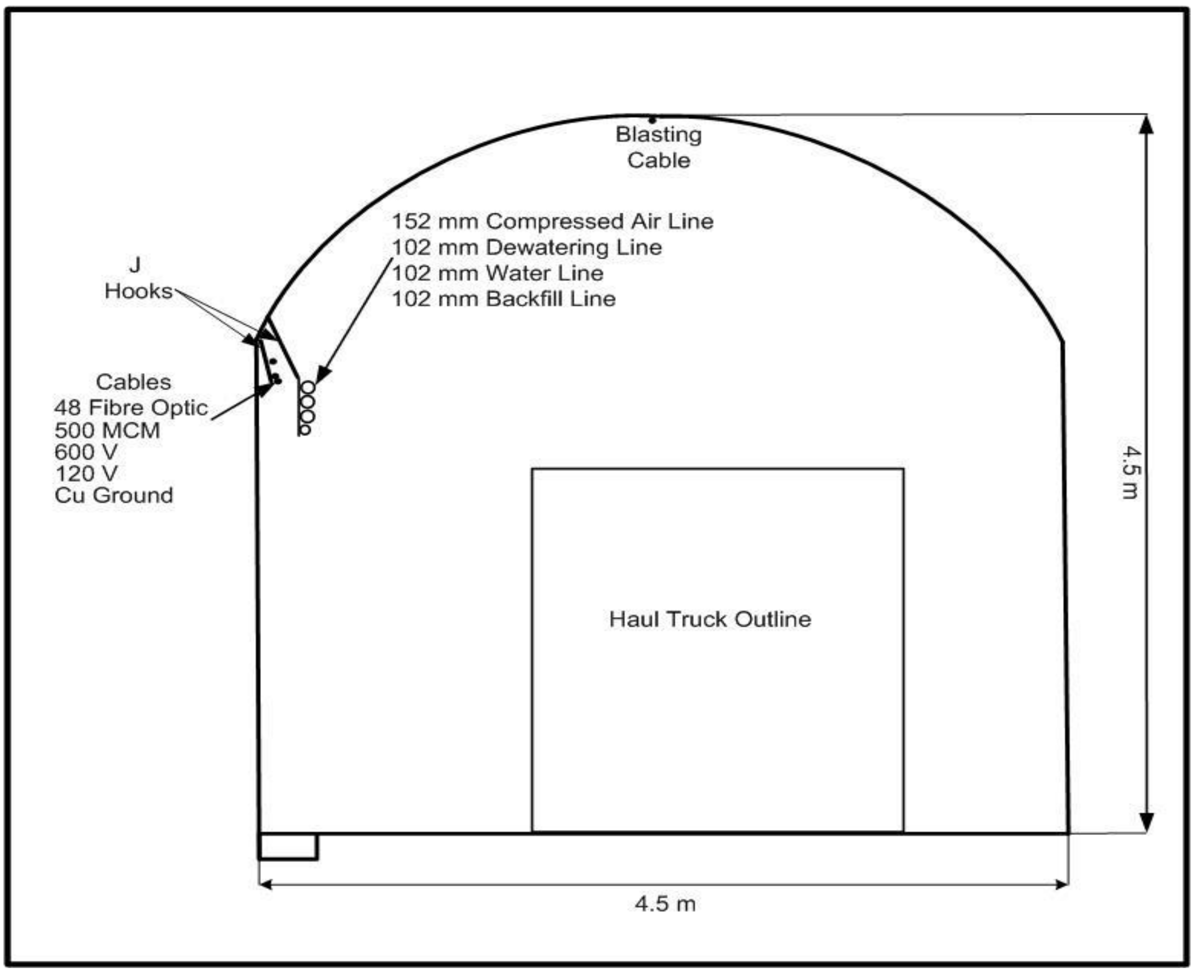


Figure 16.2 Ramp and level access cross section configuration

Services installed in the access drift will be 102 mm airline, 102 mm service waterline, 102 mm discharge water line, 102 mm backfill line, 600 and 120 volt cable, 12-fibre fibre optic data and communication cable, and a central blasting line. Figure 16.3 presents a drift cross section including the services configuration.

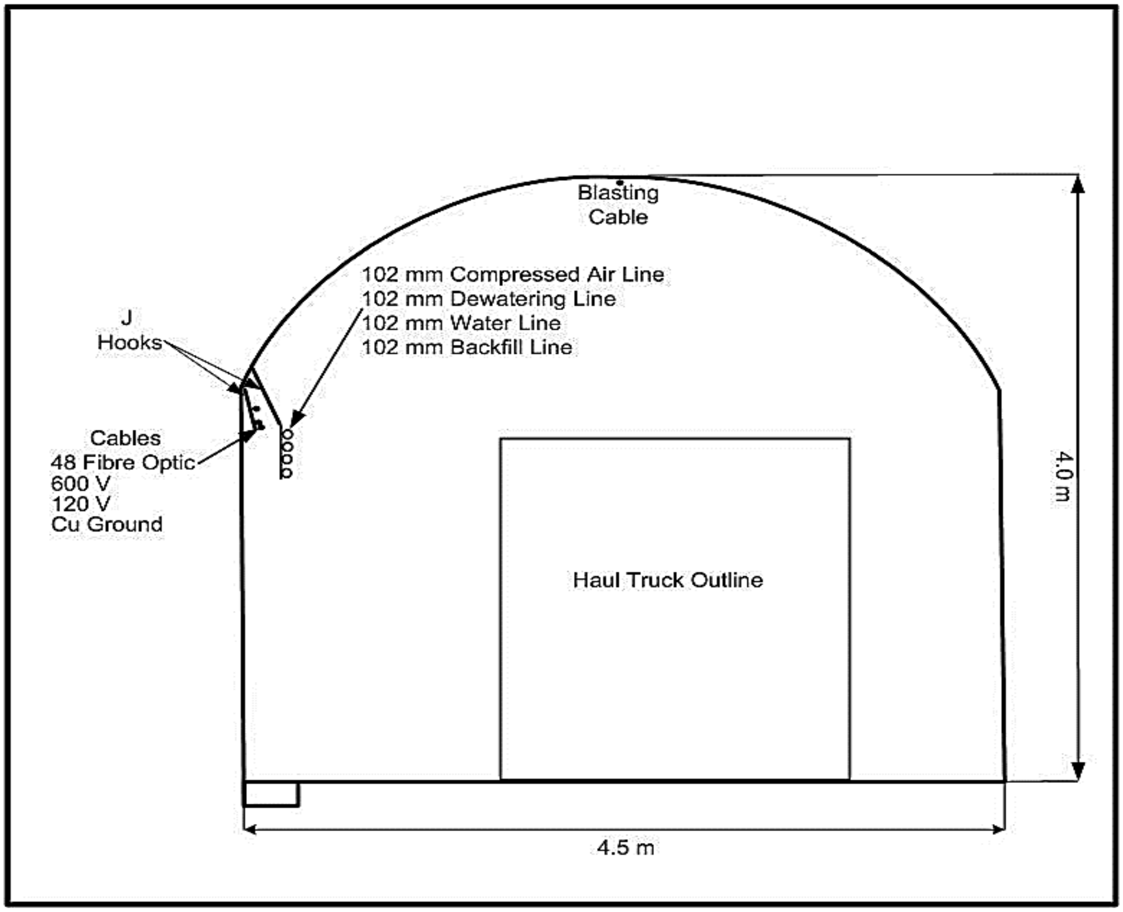


Figure 16.3 Level cross section configuration

16.8 ROCK HANDLING

From up dip panel stoping, ore will be transported by LHD units to be directly loaded into an ore haul truck (via end loading) or placed in a nearby ore remuck bay. The underground haul trucks will haul ore directly to the surface via the levels and ramp.

On the surface, the underground haul trucks haul the processing plant feed ore to a stockpile near the portal. A front-end loader reclaims the ore from the stockpile and loads it into a 40-tonne surface end dump truck, which hauls the ore to the crusher dump at the processing plant. The crusher dump is equipped with a grizzly and rock breaker to size underground ore to -0.6 m before crushing.

Waste rock will be placed in mined out stopes or trucked to a surface waste rock stockpile.

16.9 UNDERGROUND SERVICES AND INFRASTRUCTURE

Underground infrastructure will include:

- Breakdown maintenance shop;
- Fuel stations;
- Explosives and detonator magazines;
- Refuge stations;
- Main dewatering sumps;
- Main storage areas;
- Latrines;
- Electrical substations; and
- Mine wide wireless communication and control system.

Mine surface support facilities located in the area of the portal will include a surface ventilation fan set-up, backfill plant, maintenance shop, explosives magazines, mine rescue station, power substation, compressor station, small warehousing facility, laydown yard, and a water storage pond.

16.9.1 ELECTRICAL DISTRIBUTION

Primary electrical power for the mine would be provided from the main surface substation connected to the outside powerline.

The powerline would be connected to a surface substation located near to the mine portal. Power from the main substation would feed the main underground power line, a 500 mcm cable, installed in the main access ramp from the surface. This powerline would feed portable substations located on levels central to the working areas. Portable power centres would supply loads on the nearby levels and transform power down to 4,160V and 600V, as required.

On the surface, the substation would also provide 4,160V feeds to drive ventilation fans and other power requirements for the underground mine surface facilities. The system would utilize a switch room/MCC panel near the ramp portal.

The main underground mine electrical feed will consist of a 4,160V, armoured three (3) conductors, and 5 kV, 500 MCM teck cable installed in the ramp. A grounding conductor will also be hung in the ramp in conjunction with the 4,160 cable. Equipment underground will be powered by 750 kva portable substations located in electrical substation openings. The substations will step power down to 600V for mining equipment and 120V for smaller electrical powered equipment.

Table 16.2 presents the connected load list for underground and estimated electrical power consumption during peak mine development and production periods.

Table 16.2: Electrical Power Consumption during Peak Mine Development and Production Periods

Unit	Quantity	Load Factor (%)	Operating Hours Per Day (Hr.)	Consumption per Unit (kW)	Total Installed Power (kW)	Total Consumption Per Month (kwh)
Development Jumbo	2	80%	16	180	360	138,240
Diamond Drills	1.5	70%	18	90	135	51,030
Slusher	5	80%	16	23	113	43,200
Booster Ventilation Fans						
20 hp	2	75%	20	15	30	13,500
40 hp	1	75%	20	30	30	13,500
50 hp	2	75%	18	38	75	30,375
75 hp	4	75%	18	56	225	91,125
100 hp	2	75%	18	75	150	60,750
150 hp	2	75%	20	113	225	101,250
Main Ventilations Fans						
Main Intake Fan	1	100%	24	300	300	216,000
Main Exhaust Fan	2	70%	24	150	300	151,200
Main Portal Exhaust Fan	2	80%	24	100	200	115,200
Pumps						
Main Dewatering Water Pump	1	80%	12	38	38	10,944
Miscellaneous Pumps		80%	6	45	45	6,480
Compressors						
Compressor 1	1	90%	24	200	200	129,600
Compressor 2	1	60%	16	200	200	57,600
Compressor 3		10%	16	200	0	0
Lighting & Misc. Uses		100%	24	15	15	10,800
TOTAL POWER CONSUMPTION						**1,240,794**

16.9.2 Compressed Air

Compressed air would be supplied by two (2) compressors in enclosures located in the warehouse, backfill, and a compressor building near the ramp portal. They would provide approximately 238 cm per minute at a minimum pressure 8.3 bar (120 psi) to the underground mine. Each compressor would operate at half capacity to ensure one compressor could provide mine requirements when the second compressor is being repaired or maintained.

The compressors would supply the main compressed air pipeline located in the main access ramp from the surface.

Compressed air consumption is presented in Table 16.3.

Table 16.3: Compressed Air Consumption

Unit	Quantity	Utilization	Service Water Consumption			Compresssed Air	
			Per Unit (l/min)	Total (l/min)	Total Yearly (cu.m.)	Per Unit (cu.m./min	Total (cu.m./min.
Development Jumbo	2	70%	250	500	136,080		
Long Hole Drill	2	80%	60	120	37,325	20	40
Diamond Drills	2	80%	140	280	87,091		
ANFO Loader	2					15	30
Shotcrete Machine	1	100%	8	8	3,240	24	24
Stopers	4	60%	60	240	55,987	6	24
Alimak Jackleg Longtoms	6	50%	60	360	69,984	10	60
Air Tools (1 lot)	1					40	40
Misc. Water Usage (1 lot)	1	100%	20	20	7,776		
Water Sprays - Mucking (1 lot)	1	100%	40	40	15,552	20	20
Gland Water, etc. (1 lot)	1	100%	10	10	3,888		
TOTAL CONSUMPTION					**416,923**		**238**

16.9.3 Service Water

The underground mine would require approximately 417,000 m^3 of service water per year for use in drilling, dust suppression, etc. This water will be supplied from a water storage pond on the surface, which will store water recycled from the underground mine. All service water requirements will be met by water pumped out of the mine and sent to the surface water storage pond.

Water would be sent underground in a pipeline located in the trackless access ramp from the surface. This will feed the main distribution lines on the levels, which would send water to the stope access crosscuts. Water pressures and volumes would be controlled by installing water stations, at appropriate vertical intervals within the mine, which would house a transfer station and holding tanks.

16.9.4 Mine Communications and Control Systems

An 802.11 (Wi-Fi) voice and data transmission network will connect the mine and the surface operations. The system is comprised of access points (transmits data to and from clients -computers, tags, PLCs, etc.) installed in the mine drifts which facilitate communication between clients and transfers data to a database server and control system on the surface. Wired telephones will be located at key infrastructure locations, such as the refuge stations. Key personnel (such as mobile mechanics, crew leaders, and shift supervisors) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will be supplied with handheld mobile telephones, suitable for use underground, for contacting over the 802.11 network.

16.9.5 Mine Dewatering

The average underground dewatering requirement during production will be approximately 1,720 m^3 per day. This quantity is based on the mine water balance as follows:

- 1,160 m^3 per day will be produced by underground production mining and development activities;
- 60 m^3 per day will decant from hydraulically backfilled stopes; and
- 500 m^3 per day net groundwater inflow to the mine.

During the pre-production and initial year of production periods, water will be pumped directly from level sumps to the surface because mining will be near the surface.

The long-term mine dewatering system will include water collection sumps located on each level. The sumps would be located near the point where the ramp and level access crosscuts intersect and would be designed to prevent water entering the ramp from the levels. Overflow drill holes from the sumps would send water to the main water collection sumps, for settling, recirculation, and/or discharge from the mine. Main collection sumps would be located on the 215 and 425 Levels. Each main sump would be comprised of 2 dirty water and 1 clear water sumps. Dirty water sumps would be subdivided by removable timber baffle walls into 3 compartments to aid in settling of solids. The dirty water sumps would be used one set at a time, and slimes removed from the non-operational sump with LHDs. Water would overflow from the dirty water sumps into a clear water sump.

Each clear water sump, similar in size to the dirty water sumps, would be utilized to treat and store clear water prior to recirculation within the mine or discharge. Water would be pumped to a surface holding pond for underground process water or discharged to the water treatment facility on the surface.

16.9.6 Breakdown Maintenance Shop

A small breakdown shop will be set up during the pre-production period in a section of the lateral development dewatered to start the new ramp. This shop will be used until a permanent breakdown shop is located lower in the mine. The mobile equipment breakdown maintenance shops would be used to perform all breakdown maintenance on mobile mining equipment. Major equipment preventative maintenance work and other major repairs would be performed in a surface shop located near the portal.

The permanent breakdown shop would be constructed near the 150 Level, off the ramp. The shop would consist of a main shop area for one large piece of equipment or a couple of smaller units. The facility configuration would consist of an access drift leading to the main shop area, a welding area, wash bay area, parts storage warehouse, electrical room, lunchroom, and supervisor's office.

The main shop area would be equipped with an overhead bridge crane. The electrical room, meeting room, and office would be isolated by steel hinged doors. The lunchroom would be equipped with wooden benches and tables and the office would be equipped with computer workstations connected to the mine information management system.

16.9.7 Fuel Stations

Portable self-contained fueling and lubrication stations will be located on levels where mining equipment is parked. The units have built in isolation doors and fire suppression.

SatStat fuel station bladders will be filled at the surface tank farm and transported to the underground fueling station on a flat-bed utility vehicle. The SatStat bladder will be set into the stationary SatStat fueling station from which fuel will be dispensed by equipment operators. Each bladder has a capacity of 1,000 litres. The station will be equipped with heat-sensitive fire suppression from Ansul. A second SatStat station storing oils and lubricants will be located near the fuel station. Several of these fueling and lubrication stations will be placed on different levels of the mine (Figure 16.4).

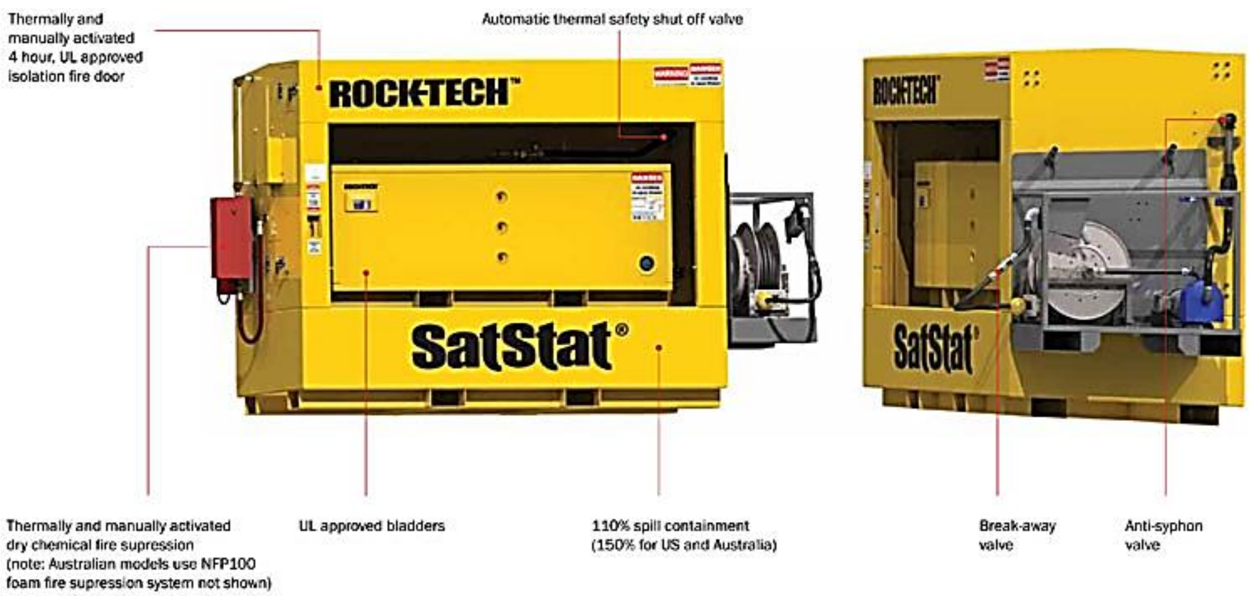


Figure 16.4 SatStat fuel station

16.9.8 Refuge Station

Main refuge stations would be located on the 75, 180, 285, 390, and 495 Levels.

Refuge stations would be fitted with a double door entry system in concrete walls at one end. The facility would include wooden benches and tables, hand washing station, and other equipment and supplies, as well as a supervisor's desk and other associated furniture. The refuge stations would also be equipped with safety and rescue equipment. Compressed air and water lines would be connected from the mine's supply system to lines inside the refuge station. The facility would be fitted with an electric heater unit and be vented through intake and exhaust ventilation ducts to the outside.

16.9.9 Explosives Storage

All blasting would utilize ANFO explosives. ANFO would be delivered in bulk bags, to the explosives magazines. Other stick explosives would be stored in this magazine as well.

Explosives magazines would be located on every third level. The explosive magazine floor would be gravel and the magazine entrance would include a concrete wall with doors to allow access for mobile equipment and people traffic. Both sides of the magazine would be fitted with wooden shelving on which bulk explosives bags can be placed. This magazine would require a fire suppression system. A flashing red light would be mounted by the entrance to indicate its location.

16.9.10 Detonator Magazine

Detonator magazines would be located near the explosives magazines. The magazines would be equipped with a gravel floor and suitable wooden shelving to allow stacking of detonator boxes on each side. The entrance would be blocked with timber posts and screen, with a man door in the wall. A flashing red light would be mounted by the entrance to indicate its location.

16.9.11 Materials Storage Areas

Storage areas, specially constructed for the purpose for storing mining consumables including pipe and fittings, ground support materials, ventilation supplies, etc., would be developed on every third level. The storage areas would include shelving and low wooden racking to safely store articles. Materials and parts would be palletised or placed in specially designed containers (for bulk materials and parts) for sending underground via the ramp. Service vehicles would transport the bulk materials to the storage areas. Materials would be distributed from the storage areas to work place storage areas by service vehicles.

16.9.12 Washrooms

Portable toilet units, equipped with a mine toilet and small sink, would be located on appropriate working levels and near the refuge stations.

16.10 SURFACE SUPPORT FACILITIES

Surface support facilities would include a main maintenance shop, backfill plant, explosives magazines, laydown yard, mine rescue station, water storage pond, power substation, and compressor station.

A small maintenance shop facility would be provided to perform major equipment repairs and rebuilds. A description of the shop facility is contained in the infrastructure section of this report. The warehouse for mine items only would be a combination of pallet (large or bulk items) and shelved (smaller items) storage.

The explosives storage area for the mine would be located 500 m from the mining and other facilities. The magazines would be housed in metal shipping containers and located so they can be observed by security located at the services site. The magazines would not be in direct line of sight of the mine or other facilities to protect mine personnel, equipment, and facilities.

A laydown yard would be constructed near the ramp portal to store materials and equipment required for the underground mine. This laydown yard would have raised timber stands on which to place large materials, such as screen, pipe, etc., as well as gravel graded areas for storing equipment and materials. A storage building would store equipment requiring protection from the elements.

A fully equipped mine rescue station is required on the property. The mine rescue station would be equipped with all necessary equipment, including self-contained breathing apparatus, flame lamps, gas testing equipment, rescue equipment, etc., and supplies and chemicals required to operate the station. There would be enough equipment to, in an emergency, have three 5-person mine rescue teams operating or on standby at any one time.

All underground mine water would be sent to a water storage pond and reused or discharged.

16.11 MINING METHOD

The proposed mining method is up dip stope mining with stopes developed and mined at the inclination of the ore zones (see Figure 16.5 and can be referred to during the method description that follows). Stopes would be 23 m wide by the height of the ore zone or 1.5 m height, whichever is greater. Stopes would be developed between levels spaced at 35 m vertical intervals. Stopes would be mined from the lower level to the upper level and broken ore removed from the stope by pulling it down the slope with a scraper slusher. At the bottom of the stope, a front-end loader (LHD) picks up the ore and end loads it into underground haul trucks for transport to the surface. A description of the stope development and mining methodology follows.

A LHD mucking drift (4.0 m high × 5.5 m wide) will be developed along the strike length of the stope. An inclined pilot drift at the dip of the ore zone would be developed perpendicular to the LHD mucking drift in the middle of the stope and from the bottom to the top of the stope. A long hole drill will drill holes up to 10 m in length into the walls of the inclined pilot drift. The long holes would be drilled into both walls of the inclined pilot drift at a downward angle toward the mucking drift. The long holes would be loaded with explosives and blasted several rows at a time on either side of the inclined pilot drift. The blasted ore would be moved by a slusher from the blasted horizontal bench to the LHD mucking drift.

More details on each cycle in stope development and mining follows.

The proposed mining method provides a proven and safe series of development and production cycles to mine an inclined stope of approximately 20 m wide and 60-70 m length up dip. Stope heights would be from 1.5 m to the height of ore (maximum height approximately 3-4 m).

The stope development phase creates the LHD mucking drift in ore along the strike width of the stope. This would be developed using 2 boom electric hydraulic jumbos for drilling development rounds and LHDs and trucks to move blasted rock away from the advancing face.

- The stope pilot raise would be developed by Alimak at the centre of the stope and up dip with dimensions of 3 m by 2.4 m height (Figure 16.5). The Alimak is used to move men and gear to the blasted face area. The Alimak is stopped at the end of the rail where ground support ends. A work staging incorporated into the Alimak platform is dropped into place, to provide a level working floor, from the Alimak. The steps in developing the pilot drift are as follows (refer to Figure 16.6 and Figure 16.7):
- The Alimak is moved to the face of the pilot drift. The round in the face of the pilot drift is drilled using jackleg drills operating from the Alimak platform (dropped into place at the face).
- The round drill holes are loaded with explosives and detonators from the Alimak platform. Explosives used are ANFO and stick powder.

- The Alimak is moved to the bottom of the pilot drift and into the protected Alimak nest and the round blasted.

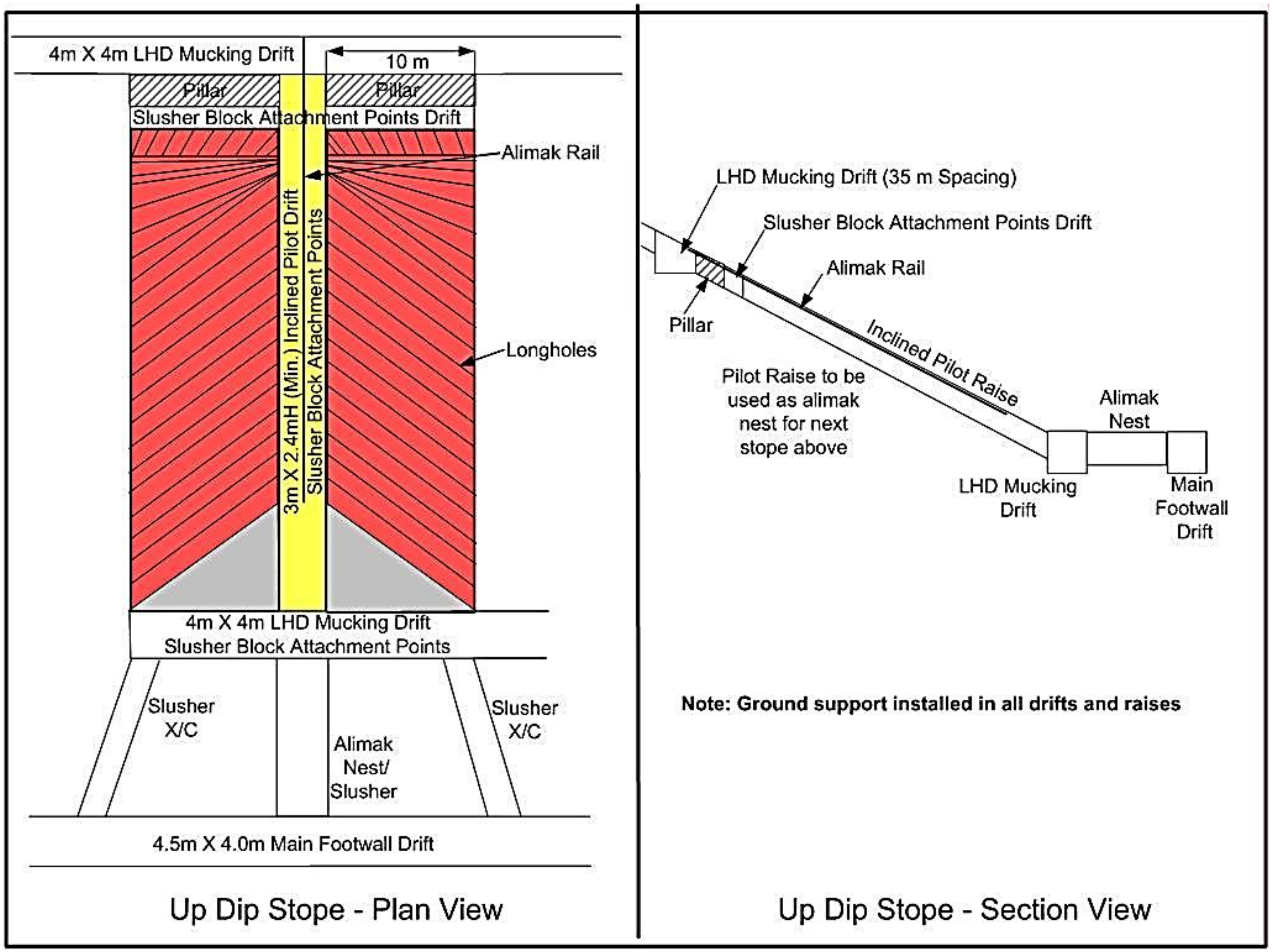


Figure 16.5 Up dip mining method general arrangement

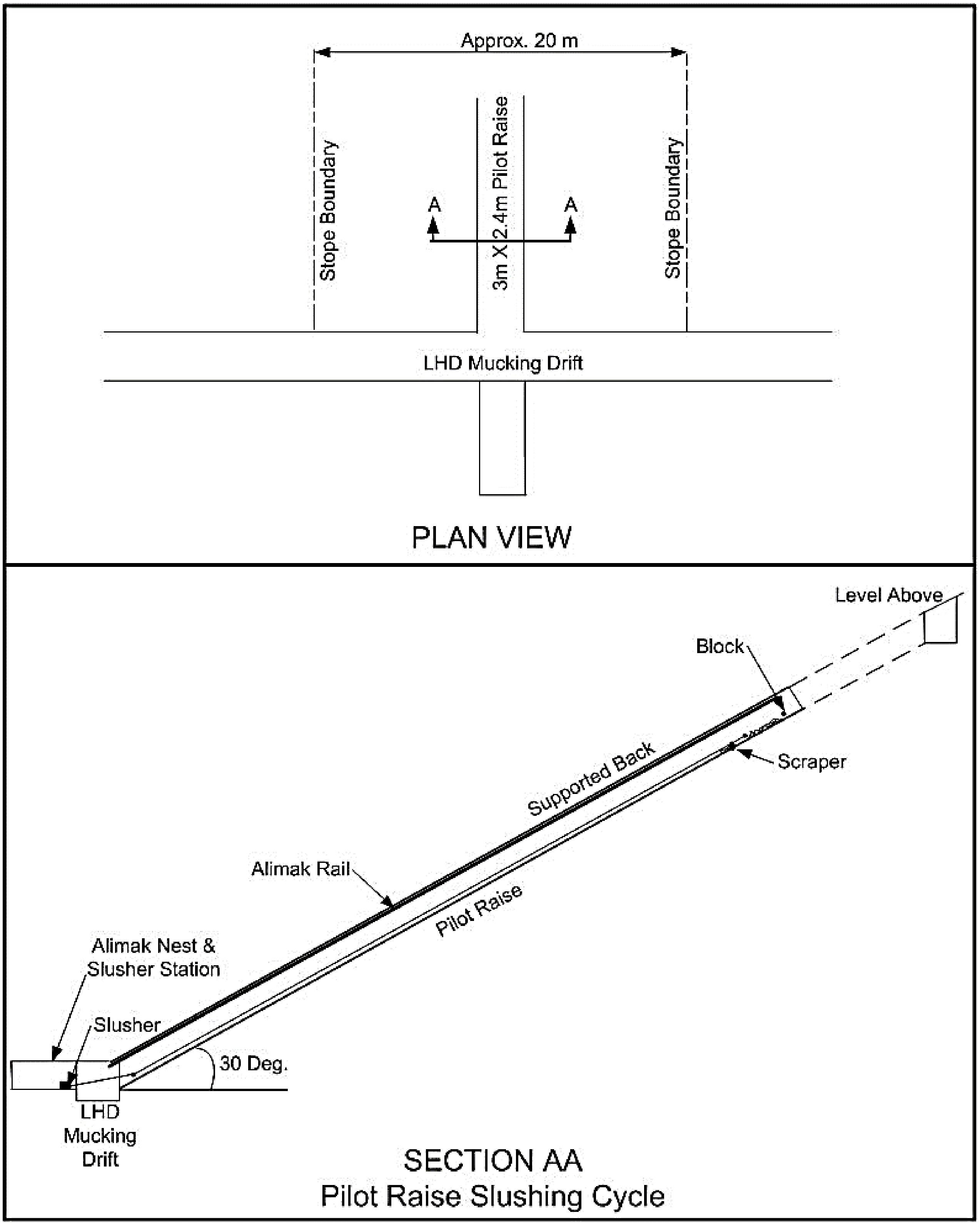


Figure 16.6 Pilot raise development

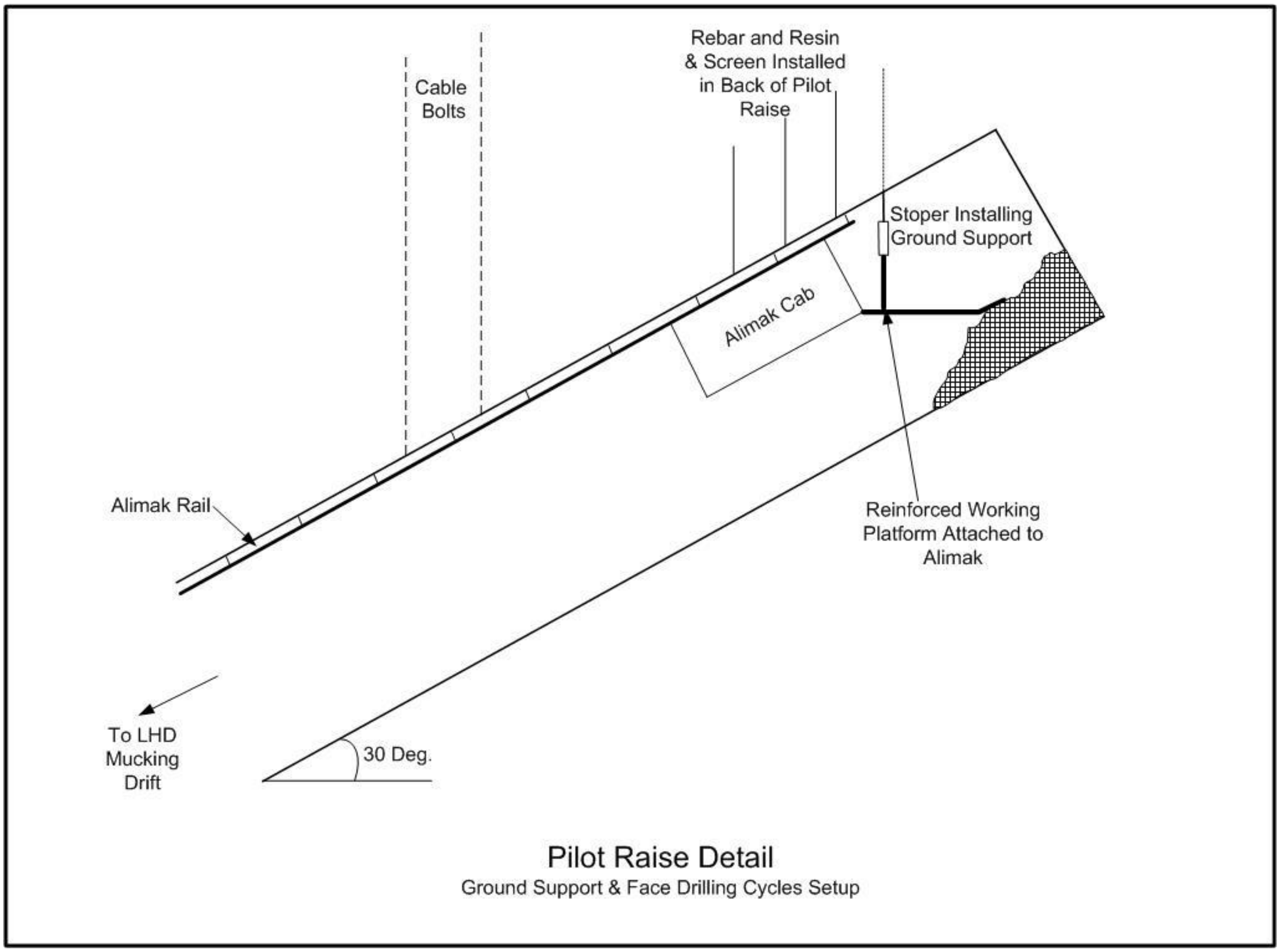


Figure 16.7 Pilot raise development detail

- Blasted rock is removed from the raise using a slusher and scraper prior to the Alimak traveling in the raise. The rock would be scraped to the bottom of the pilot drift and loaded by an LHD into underground haul trucks for removal. The slusher operator would have lights and a camera to see what he is doing.
- When rock removal is completed, the Alimak is moved to the face.
- The heading is scaled and ground support installed. Ground support will consist of fibreglass rebar with resin and welded wire mesh screen (back and top half of walls) installed from the working platform. Screen is not installed on the lower half of the wall to prevent damage to and interference with the scraper mucking.
- After ground support has been installed, the next raise round is drilled off with jacklegs operating on the working platform.

After a pilot raise is completed, cable bolts will be installed in the hanging wall of the stope, in a fan pattern, from the pilot raise. The cable bolts will generally be bulged twin strand 7-10 m long and grouted in place. The pilot raise will break through to the next level and three (3) rounds will be developed into the hanging wall for the next stope up dip.

Once the raise reaches an elevation of approximately 7 m below the LHD mucking drift on the level above the stope, a sub-drift perpendicular to the raise and along strike across the total width of the stope would be developed. This drift would allow for attachment of slushing blocks in different positions during slushing out of the stope, once blasting is completed up to this drift. As with the pilot, raise ground support will be installed in this drift to allow miners to work in the drift. The raise would be completed (after the drift) to the level above.

Drilling and blasting of the two side wings of the stope would be the next cycle (see Figure 16.5). For this phase, the Alimak would travel from the level above the stope to the drilling or blasting positions. Drilling of the long holes would progress from the top of the stope downwards to prevent damage or fines build up in the collar of the holes. The Alimak would have a long hole drill setup attached to the unit and moved into position at each drill hole ring. The working platform would be dropped into place and the long hole drill secured in position. A combination of horizontal and down dip and up dip angled holes would be drilled to blast the stope to its full height (1.5 m minimum height or the height of the ore zone) and width. Extension style drill rods would be used to drill the desired hole lengths. The blast hole pattern can be adjusted to conform to changes in ore geometry, including rolls in the orebody on dip, to minimize dilution. Drill holes would be loaded with ANFO and detonators from the Alimak and blasted, mining a series of slices up the length of a stope. The blasted rock from the ring blasting would be moved by slusher and scraper to the LHD mucking drift at the bottom of the stope. The walls of the stope during blasting would be angled to facilitate slusher mucking of the broken ore. The slusher blocks would be moved along the wall of the LHD mucking drift, along the pilot raise and ultimately in the sub-drift at the top of a stope to allow the slusher to reach the broken ore.

Once a stope is mined out, it will be backfilled with cemented (stopes on one or both sides still to be mined) or uncemented hydraulic tailings backfill.

In the proposed mining method, the Alimak is a means to move people and gear efficiently to workplaces and the orebody dip angle is not relevant for the Alimak, as it can operate at any angle (by design) as long as the rail is securely fastened to the back of the pilot raise. The Alimak also provides an opportunity to more easily put in place working platforms and allow for long hole drill setups.

16.11.1 OPENING SIZES

The pilot raises would be a minimum of 3 m wide and 2.4 m height or the height of the ore zone, whichever is greater. Minimum raise heights may be lowered marginally, as operational experience allows.

Stope access crosscuts will be developed in waste from the ramp to the footwall ore contact and then a level access drive or ore haulage drift developed along the strike in the ore zone. The stope access crosscuts will be developed 4.0 m wide by 4.0 m high at a +3% gradient from the ramp and the level access drifts will be developed 5.5 m wide by 4.0 m high.

The lateral development headings will be drilled off using 57 mm diameter drill holes with four (4) cut holes reamed. Headings will be blasted using a combination of ANFO, stick emulsion for the lifters, perimeter blasting products for the wall, and back holes and nonel caps, initiated by electric caps. Ground support will consist of 1.8 m long resin grouted rebar, installed on a 1.2 m by 1.2 m pattern and welded wire mesh screen, on the backs and walls to within 1.5 m of the floor. The ground support will be installed using bolter-screener scissor lift units, equipped with a drilling boom. All services will be installed from scissor lifts.

Services installed in the stope access crosscuts will be 102 mm airline, 51 mm service waterline, 102 mm discharge water line, 102 mm backfill line, 600V cable, and a central blasting line.

The pilot raise will be drilled with handheld pneumatic drills (jackleg and stopers) to drill 32 mm diameter holes for ground support and the raise rounds. The raise rounds drill pattern will typically be a 600 mm × 600 mm pattern with an extra nine (9) cut holes, four (4) of which are reamed. The slash holes will be drilled on a 900 mm by 900 mm pattern. Perimeter holes may be drilled on a tighter pattern, typically on a 450 mm spacing depending on ground conditions.

16.11.2 DILUTION

Based on the proposed mining method, a minimum mining height (ore thickness) of 1.5 m has been determined, which allows for breaking of the stope rock and slushing of the broken ore out of the stope. This minimum mining height has been applied to the resource calculations and included in the reported Measured and Indicated Resources for reserves determination.

Additional dilution in stopes was added, based on the minimum mining width 1.5 m, stopes for a mining height of 1.5-2.4 m and stopes with heights greater than 2.4 m. The 2.4 m height increment is based on this being the height of the raise that the Alimak will develop in the stope and the raise may contain all waste or ore or a combination of ore and waste, which will cause different overall stope dilution to occur. The different dilution factors are determined as follows.

16.11.2.1 Stopes of Minimum Mining Height – 1.5 m

Dilution in the stope is the volume of the raise greater than the minimum mining height added at zero grade to the total ore tonnes contained in a stope at the minimum mining height.

- Raise 3 m wide by 2.4 m high; stope 20 m wide;
- Raise Height Dilution = 2.4 - 1.5 = 0.9 m; and
- Total Stope Dilution = 0.9 × 3/20 × 1.5 × 100 = 9%.

16.11.2.2 Stopes of Height – 1.5-2.4 m

Dilution in the stope is the volume of the raise greater than the ore height plus a waste overbreak allowance of 0.3 m in the hanging wall above the ore contact (over the width and length of the stope) both added at zero grade to the total undiluted ore tonnes contained in a stope. This dilution averages 15%.

16.11.2.3 Stopes of Height Greater than 2.4 m

Dilution in the stope is 0.3 m waste overbreak allowance in the hanging wall above the ore contact (over the width and length of the stope) at zero grade included in the undiluted tonnes of ore in a stope. This dilution averages 8%.

The overall weighted average dilution of reserves is approximately 13%.

16.12 MINING OPERATIONS

16.12.1 DRILLING

Long holes in the stopes will be drilled off using a long hole drill boom incorporated into the Alimak with pressurized stabilizer arms, which can be extended to the back and floor and walls of the pilot raise to maintain drill setups. The drill would use 1.5 m long extension steel and button bits to drill the long holes of 64 mm diameter. The same drilling unit would be used to drill cable bolt holes.

16.12.2 BLASTING

Up dip stoping will be blasted with ANFO in the non-perimeter holes and low impact emulsion in the back and floor perimeter holes to minimise overbreak. All explosives will be initiated using electric initiation systems connected to a central blasting system.

16.12.3 GROUND SUPPORT

The footwall drifts, LHD mucking drifts, and stope access crosscuts will be supported with resin grouted rebar and welded wire mesh screen. All ground support will be installed using scissor lift units in trackless development headings.

In the up dip stopes, resin grouted fibreglass rebar and screen will be installed from the Alimak working platform and staging on the muckpile, as appropriate. Cable bolts will be installed in the hanging wall of the stopes from the pilot raise working off the Alimak working platform. Cable bolts installed in the hanging wall of the stope, in a fan pattern from the pilot raise, will generally be bulged twin strand 7-10 m long and grouted in place. Drilling of holes will utilize the long hole drill incorporated into the Alimak. Cables will be inserted manually from the Alimak working platform and grouted using a small grouting pump. Cable support will be designed for the ground conditions and to maximise the span of the stopes that can safely be mined.

16.12.4 MUCKING

Mucking of up dip stopes will be performed by 2 drum 50 HP electric slushers. The slushers will be equipped with a 1.2 m box style scraper, with a capacity of 90-tonnes per hour (slushing a maximum of 70 m distance). Ore will be scraped to LHD mucking drift at the bottom of each stope. A 4 m^3 bucket LHD will pick up the ore and end load 30-tonne trucks for ore haulage to surface.

16.12.5 MINING EQUIPMENT

The mine development group will require two 2 boom electric/hydraulic jumbos, two 5 m^3 bucket LHDs, one 4 m^3 LHD, one 30-tonne truck, 2 scissor lift trucks, one ANFO loading unit, handheld drills, and 1 light utility vehicle. Table 16.4 presents the mine equipment fleet.

Table 16.4: Mine Equipment Fleet

Equipment	Units	Development	Production	Services	Maintenance	Staff	Total
Electric/Hydraulic 2 Boom Jumbo	each	1					1
5 cu.m. LHD	each	2	1				3
Haulage trucks 12 tonnes	each	0					0
Haulage Trucks 30 tonnes	each	1	1				2
Haulage Trucks 40 tonnes	each						0
Scissor-Lift Truck	each	1	1	1			3
Scissor Screener Bolter	each		1				1
ANFO Loader	each		1				1
Longhole Drill Rig	each		1				1
Cable Bolt Unit	each		1				1
Utility Boom Truck	each			1			1
Front End Loader	each			1			1
Light Service Vehicle	each	1	2		1	3	7
Man Carrier	each						0
Grader	each			1			1

The mine production group, including stope preparation, will require 6 double drive Alimak's, 10 electric slusher and scraper units, 3 long hole drill units for the Alimaks, 3 cable inserting units and grout pumps, three 4 m^3 LHDs, two and later up to four 30-tonne trucks, 1 scissor lift, and 2 light utility vehicles.

The mine services group will require 1 scissor lift truck, 1 utility boom truck, 2 personnel carriers, 1 grader, and a 1.2 m^3 LHD.

The maintenance group will require 1 utility boom truck and 2 light utility vehicles. Warehousing will require 1 front-end loader. The mine staff, engineering, and geology will require 2 light utility vehicles.

Underground operations and maintenance personnel will be transported to their working places in personnel carriers. During the shift, workers will travel around the mine in light utility vehicles, such as Toyota Landcruiser™ or Hilux™ vehicles, equipped with bench seats in the box for people to sit on. Service vehicles for materials and parts will consist of flat bed or pickup trucks with a box, which can hold palletized, containerized, or individual items. Mine staff, engineering, and geology personnel will travel in light utility vehicles.

16.13 MINE BACKFILLING

All stopes will be backfilled with hydraulic backfill, cemented and uncemented, to fill the voids, and prevent caving. The hydraulic backfill will consist of classified mill tailings. Hydraulic backfill will be delivered at approximately 70% solids by weight and at a rate of 390 tpd.

16.13.1 UNDERGROUND DISTRIBUTION SYSTEM

Hydraulic backfill would be delivered to the top of the stopes by the hydraulic backfill pipelines. The backfill would be pumped from the backfill plant located by the mine portal, to the portal underground, in a 102 mm Schedule 80 steel pipe line. The main distribution line would be installed in the ramp and bore holes between levels. The main line and bore holes would feed backfill lines on the mining levels.

Distribution piping and bore hole break out points would be located on each level where the piping or bore hole intersects a level. The break out points will be located near the main ramp on each level and approximately ⅔ of the distance along the overall strike length of the zones to be mined. The vertical piping or bore hole at a level break out point will be equipped with a 102 mm high pressure flex hose. The flex hose would be connected to the level backfill piping (for backfilling a stope on the level) or to the next leg of the vertical piping or bore hole carrying backfill to a lower level.

Hydraulic fill fences, constructed at the stope entrances, would consist of a muck pile covered in shotcrete and drainage tubes through the walls. Backfill would be delivered to the top of the stope by a HDPE pipe hanging at the back of the stope. Water pressure would be monitored and filling would be interrupted, if pressure reached a safety limit.

16.13.2 BACKFILLING PAST MINING VOID AREAS

When mining extends into the dewatered old workings areas, old stopes would be backfilled to ensure stability of old and new workings. To backfill the past, mined stopes would require fill fences be constructed in key areas to protect existing drifts.

16.14 VENTILATION

The ventilation system is designed to adequately dilute the exhaust gases produced by diesel equipment. The required air volume was calculated as 0.05 cm per second (100 cubic feet per minute) per brake horsepower of diesel equipment, as per Canadian standards for Tier 3 diesel engines. Where Tier 4 diesel engines are available with equipment, Manitoba permits a reduced ventilation volume of 0.025 cm per second (50 cfm) to be allowed for this equipment. The horsepower rating of the underground equipment was determined and utilisation factors were applied to estimate the total amount of air required (see Table 16.5). A model of the PL Mine main ventilation circuit was created in VNETPC from the mine design and standard frictional resistance values. Headings that would only be supplied with air via auxiliary ventilation were not included in the model.

Table 16.5: Equipment Ventilation Requirements Calculation

Unit	Quantity	Engine HP	Engine KW	Total Installed HP	Utilization	Total Ventilation CFM
Development 2 Boom E/H Jumbo (Tier 4)	2	150	113	300	25%	3,750
Anfo Loader Truck	1	150	113	150	25%	3,750
4.6 cu. m. LHD (Tier 4)	4	293	220	1,172	80%	46,880
30 t Haul Truck (Tier 4)	4	413	310	1,652	100%	82,600
Other Ground Support Drill	1	150	113	150	25%	3,750
Scissor Lift	4	149	112	596	20%	11,920
Grader	1	173	130	173	100%	17,333
Service/Boom Truck	1	149	112	149	100%	14,900
Mechanics Truck	1	149	112	149	40%	5,960
Personnel Vehicles	5	50	38	250	50%	12,500
TOTAL				4,741		203,343

The mining operation to support the mining equipment fleet would require ventilation air volumes of approximately 94-104 cm per second (200,000-220,000 cfm). The ventilation system would consist of a push-pull system utilizing ventilation raises and the main access ramp.

Two 4 m by 4 m ventilation raises would be developed from the surface to the bottom of the mine in legs and be located at either end of the levels. An additional ventilation raise of the same size would be developed at approximately the northern extremity of the old mine workings from the surface to the bottom of the mine. One raise would be an intake raise and the other two exhaust raises. High pressure fans would be located on the surface on top of the exhaust raises and low pressure fans on top of the intake raise.

Air would flow from the intake ventilation raise along a level, be picked up by auxiliary ventilation fans, and pushed into stope accesses. From there, air would flow in the LHD mucking drift and up the pilot raise in the centre of the stope to the main footwall drift on the level above the stope. Air would travel in the main footwall drift to the exhaust raise and to the surface in the raise. Approximately one-half of the fresh air sent underground would be split off and enter the ramp from the levels and flow up the ramp to the surface.

If required, low pressure fans would be connected to the ramp near the portal to assist air exhaust to the surface.

Ventilation doors and fan installations will be constructed at the portal to accommodate passage of men and equipment to and from the mine. The intake raise will also have a manway installed in it to provide a second means of egress.

The intake raise will be equipped with high volume, low head fans and a mine air heating unit fired by propane. The intake fans would be two 182 cm, 1,000 RPM and 150 HP at approximately 1 kPa operating pressure and supply 140 m^3/s (210,000 cfm) of fresh air.

The exhaust fans installation will be located on the surface at the top of the 2 exhaust ventilation raises (south and north exhaust raises) and at the portal. The 2 exhaust raise ventilation fans will be 152 cm, 1,500 RPM, 150 HP fans with a capacity of approximately 40 m^3/s (85,000 cfm) with maximum operating head pressure capacity of approximately 2-2.5 kPa. The portal fan will be two 100 HP fans moving 50 m^3/s (100,000 cfm) of air, at approximately 2-2.5 kPa operating pressure. Fans will be variable speed to facilitate adjusting of air volume delivery to the working areas, as required.

Fresh air delivery to the stopes will be controlled using auxiliary ventilation fans and ducting. Ventilation regulators, doors, and bulkheads will also be used to control the airflow in the mine.

The ramp development will use 150 HP fans. Other lateral development will use a combination of 100 HP and 150 HP fans depending on the heading length. Development headings are sized to accommodate large ducting (122 mm), to reduce head losses.

Auxiliary ventilation delivery to the stopes will typically use 75-100 HP fans, with 91 mm (36 inch) flexible ducting (Figure 16.8).

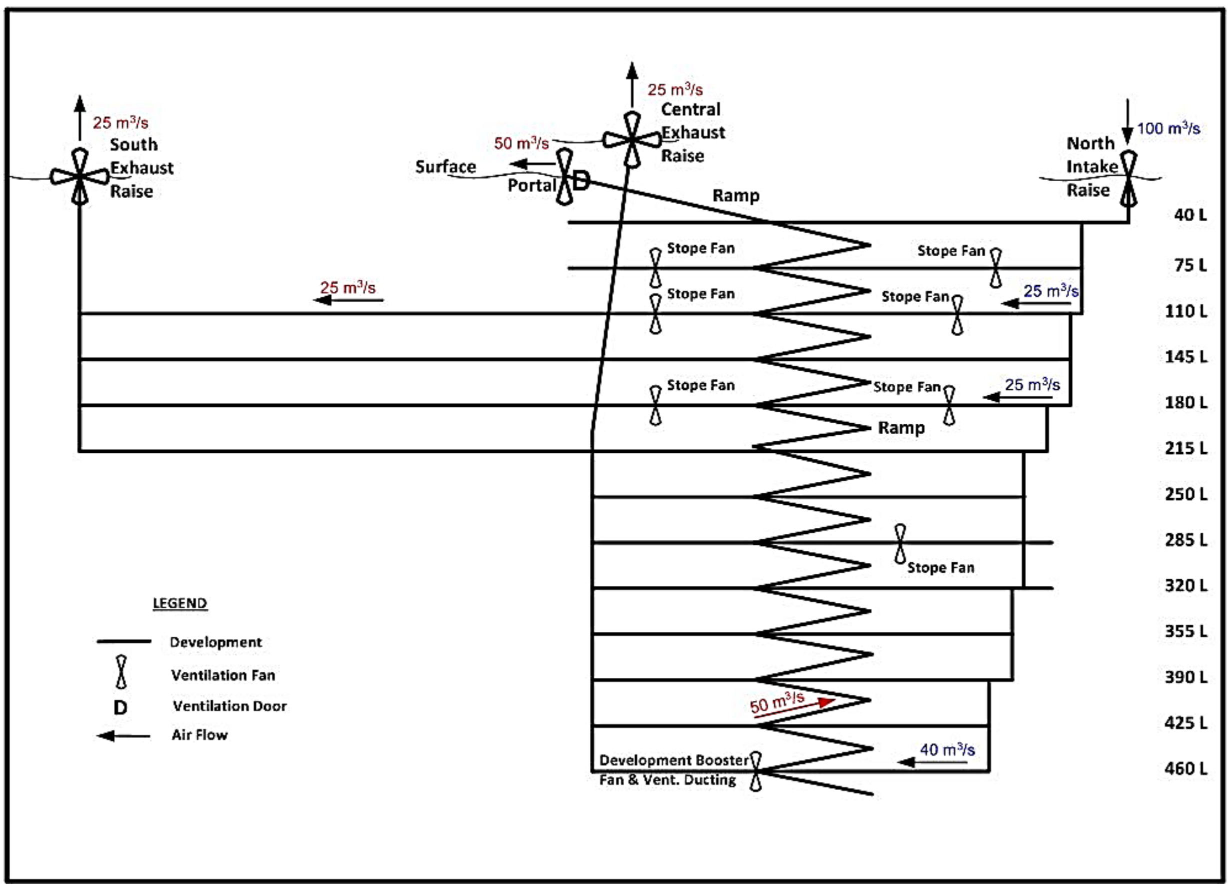


Figure 16.8 Mine ventilation schematic

16.15 OPEN PIT MINING

The open pit optimization and design work resulted in 5 small open pits to be developed to approximately 40 m depth (see Figure 16.9).

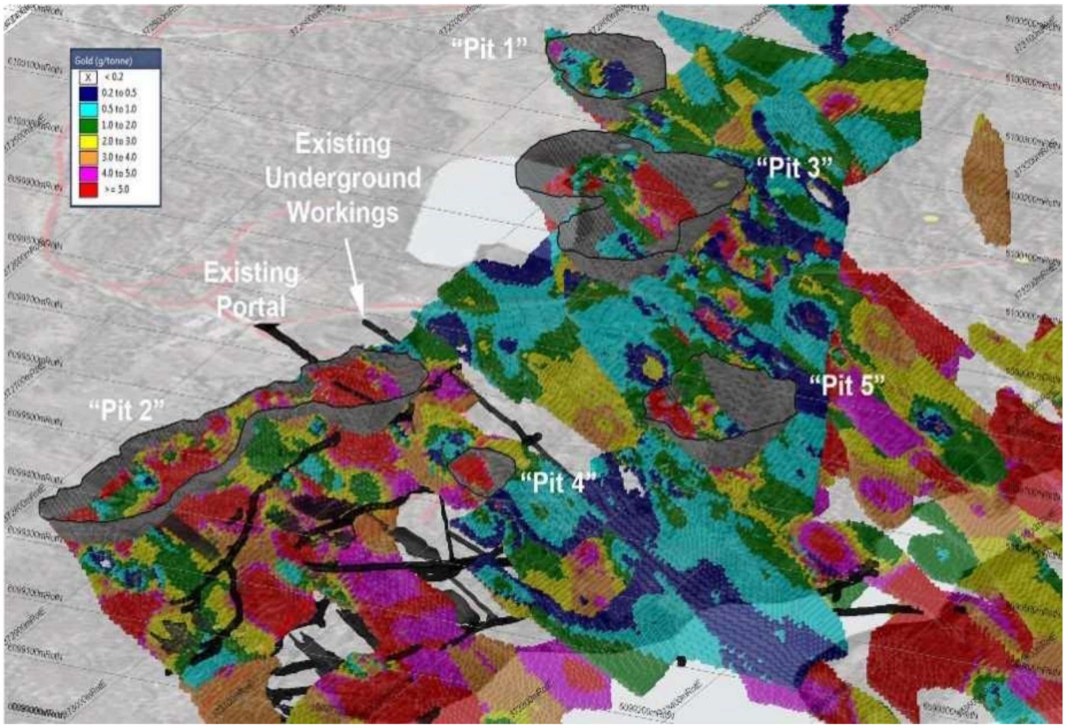


Figure 16.9 PL Mine open pits configuration

The open pits would be mined by a contractor.

The size of the open pits indicate that relatively small mining equipment would be used. The open pits would have overall pit slopes of 50 degrees. The open pit mining would utilize 5 m high benches with safety berms of 3-5 m wide included for every three (3) benches.

Drills will perform single pass drilling with 76 mm and 102 mm drill holes used in ore and waste, respectively. Ore will be drilled on a 3 m spacing by 3.5 m burden and waste on a 4 m spacing by 4.5 m burden.

The 40-tonne rock trucks will be loaded with 2-3 cm bucket excavators and ore hauled between 1.5-2 km to the processing plant.

Support equipment will include an explosives loading truck, front-end loader, D6-D8 dozer, grader, water truck, mechanics truck, and pickup trucks.

The contractor will provide all open pit personnel and supervision. PL Mine will provide survey and geology services.

16.16 DEVELOPMENT AND PRODUCTION SCHEDULES

Mine production will be 590 tpd or 216,000 tpy. Development is scheduled to meet stope mining requirements, on each yearly basis.

16.16.1 PRODUCTIVITIES

A number of independent and different methods for scheduling stopes and backfilling were used and manpower, equipment, and stope cycle components times were verified by more than one method in all cases. Core Mining Group Ltd. (sister company to AMPL) operation's people reviewed the estimates and adjusted them or confirmed them. Dependencies were checked or equipment and crews were configured to provide maximum flexibility and minimise dependencies.

Development crews in waste headings will generally have multiple headings available for advancing at any time. For development scheduling, each crew is scheduled to advance 1.5 rounds (4.6 m length) per day, of 4.5 m by 4.5 m or 4.5 m by 4m headings, for a total of 2,100 m of advance per year (not including safety bays, slashing, cut outs, etc.).

With ore development, stoping, and backfilling, the following parameters were used in determining stope requirements:

- Each up dip stope pilot raise produces approximately 50 tonnes per round, 1.5 rounds per day. There will be 2-3 pilot raises being developed at any time for a total of 100-150 tpd.
- Each up dip stope can blast and muck the equivalent of 250-300 tpd, requiring 2 of 3 available stopes mining at a time.
- To meet daily production will require 3 stopes in mining; 2-3 stopes in pilot raise development, 2 stopes in access development, 1 stope being backfilled, and 1 being readied for backfilling. Therefore, a total of 9-10 stopes at any time is required.
- Development has been scheduled so it is well ahead of the mining requirements and mining takes place on more than one level simultaneously.

16.17 UNDERGROUND MINE DEVELOPMENT SCHEDULE

The mine development schedule includes rehabilitation of existing ramp and lateral development and new waste and ore development. The development schedule ensures development is in place approximately one year before ore zone stope development and mining is required.

The development metres are based on preliminary level plans generated from the block model with lateral development centre lines applied to the plans to access all the stoping areas scheduled in the ore production schedule. Ramping and raising connect the different levels with quantities determined, accordingly. A 20% additional development factor was applied to all metres to account for safety bays, small storage areas, and other cut outs required.

16.18 MINE PRODUCTION SCHEDULE

The mine production schedule is based on mining 600 tpd of reserves, for 360 days per year. Table 16.6 presents the development schedule for life-of-mine.

Table 16.6: Life-of-Mine Underground Development Schedule

Heading	Quantity	Units	Year											Total
			-1	1	2	3	4	5	6	7	8	9	10	
Existing Development Rehabilitation														
Ramp	1,000	metres	200		800									**1,000**
Levels	400	metres			200	200								**400**
Lateral Development - Waste														
Spiral Ramp Surface to 145 Level	1,160	metres	580	580										**1,160**
Spiral Ramp 145 to 530	1,680	metres		250	300	600	530							**1,680**
40 Level	940	metres	100	300	460	80								**940**
75 Level	730	metres	100	300	230	100								**730**
110 Level	730	metres			200	300	230							**730**
145 Level	1,110	metres		200	910									**1,110**
180 Level	1,130	metres				1,000	130							**1,130**
215 Level	900	metres				100	800							**900**
250 Level	800	metres				260	540							**800**
285 Level	200	metres					200							**200**
320 Level	200	metres					200							**200**
355 Level	200	metres					200							**200**
Intake Ventilation Connection Drifts	300	metres		60	60	90	90							**300**
Exhaust Ventilation Connection Drifts	300	metres		60	60	90	90							**300**
Pre-Production Development - Ore														
75 Level	200	metres	200											**200**
40 Level	100	metres	100											**100**
Raise Development														
Intake Raise 1 - 145 to Surface	150	metres	70	80										**150**
Intake Raise 1 - 495 to 145	210	metres			74	74	62							**210**
Exhaust Raise 1 - 145 to Surface	150	metres		36	36	78								**150**
Exhaust Raise 1 - 495 to 145	210	metres			74	74	62							**210**
Boreholes														
Backfill Boreholes	360	metres		90	90	90	90							**360**
Drainholes	360	metres	70	20	90	90	90							**360**
Total Existing Development Rehab	**1,400**	**metres**	**200**	**0**	**1,000**	**200**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,400**
Total Lateral Development	**10,680**	**metres**	**1,080**	**1,750**	**2,220**	**2,620**	**3,010**	**0**	**0**	**0**	**0**	**0**	**0**	**10,680**
Total Raise Development	**720**	**metres**	**70**	**116**	**184**	**226**	**124**	**0**	**0**	**0**	**0**	**0**	**0**	**720**
Total Boreholes	**720**	**metres**	**70**	**110**	**180**	**180**	**180**	**0**	**0**	**0**	**0**	**0**	**0**	**720**

The production schedule is derived from scheduling all the stopes, which meet the 4 g/t Au cut-off grade. Year 1 initial stoping started on the 75 and 110 Levels working up to the 40 Level, as this minimises pre-production development, while accessing more than one year of production at grades significantly above the mine average grade. In subsequent years, stoping blocks between 2 or 3 levels were scheduled generally moving from the top down. Pillars in between the blocks would be removed at the end of each block life. Table 16.7 presents the summary production schedule.

Table 16.7: Summary Mine Production Schedule

Mining	Year 1		Year 2		Year 3		Year 4		Year 5		Total	
	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade	Tonnes	Grade
U/G Level												
40	97,740	7.54	105,967	6.85	15,962	4.73					219,669	7.00
75	118,260	8.36	20,964	5.80	29,022	4.87			8,846	4.25	177,092	7.28
110			20,936	10.63	29,643	9.26	13,968	5.05	11,650	4.48	76,196	8.13
145			68,133	8.19	33,125	6.30	36,946	5.40	22,581	4.16	160,784	6.60
180					67,015	6.70	89,367	5.67	28,195	4.44	184,577	5.86
215					29,592	10.17	17,394	5.82	8,052	4.49	55,038	7.97
250					11,641	13.06	13,783	7.53	1,914	4.05	27,339	9.64
285							11,354	10.67	873	4.46	12,228	10.22
320							5,516	4.72	5,635	4.24	11,151	4.47
355							27,672	7.25	981	4.03	28,653	7.14
Total U/G Mining	**216,000**	**7.99**	**216,000**	**7.54**	**216,000**	**7.42**	**216,000**	**6.16**	**88,727**	**4.34**	**952,727**	**7.00**
Open Pit												
1 - Ore									38,441	3.44	38,441	3.44
1 - Waste			0		0		0		228,338		228,338	
2 - Ore			59,516	4.32	20,000	4.96	20,000	4.96	18,167	4.96	117,683	4.64
2 - Waste			404,710		136,000		136,000		123,533		800,243	
3 - Ore			4,046	3.36	20,000	4.24	26,176	4.24	45,205	4.24	95,428	4.20
3 - Waste			43,903		217,000		284,010		490,476		1,035,389	
4 - Ore									6,384	9.28	6,384	9.28
4 - Waste			0		0		0		44,148		44,148	
5 - Ore			4,046	4.08	27,609	4.08	21,433	4.08	2,688	3.60	55,776	4.06
5 - Waste			37,830		258,144		200,399		25,133		521,506	
Total O/P Ore Mining			**67,609**	**4.25**	**67,609**	**4.39**	**67,609**	**4.40**	**110,885**	**4.36**	**313,711**	**4.35**
Total O/P Waste Mining			**486,444**		**611,144**		**620,408**		**911,628**		**2,629,624**	
Total Mine Production	**216,000**	**7.99**	**283,608**	**6.75**	**283,609**	**6.69**	**283,609**	**5.74**	**199,612**	**4.35**	**1,266,438**	**6.34**

16.19 MINE SURFACE INFRASTRUCTURE

Surface facilities will generally be centred near the portal or processing plant.

Surface support facilities will include explosives magazines, mine supervision, geology, engineering, and administration offices and mine change house, power substation, warehouse and laydown yard, and water collection ponds.

16.19.1 EXPLOSIVES MAGAZINES

The explosives magazine would be located 500 m from any facility, including the mine portal. The actual magazines would be provided and permitted by the explosives supplier.

The area would be cleared and a gravel base laid. The shipping containers used to store the explosives and detonators would be raised off the ground to assist in the transfer of explosives from the delivery trucks to the magazines. The area would be fenced around its entire perimeter with a locked gate access. The area would be provided with lighting. Outside the fencing, a berm of several metres height would be constructed to contain any potential explosions in the magazines.

16.19.2 BACKFILL PLANT

The hydraulic backfill plant would be capable of delivering the total daily tailings production underground. The following criteria are used in the plant capacity design:

- Yearly Mining Rate 216,000 tonnes
- Mine Operating Days 360 days
- Daily Mining Rate 600 tonnes
- Backfilling Capacity 600 tpd
- Backfilling Placement Time 18 hours per day
- Backfilling Rate Per Hour Approximately 30 tonnes dry tailings per hour
- Backfill Mixtures 70% solids: approximately 4% cement by weight and uncemented

16.19.2.1 General Flowsheet Description

Partially dewatered tailings, using the thickener located at the processing plant, will be pumped to the backfill plant at the portal. The tailings would be placed in a 4 m diameter by 4 m high agitated backfill preparation tank equipped with a mixer, in batches by volume of tailings slurry. When primary stopes using cemented backfill are being backfilled, cement will be added to the backfill preparation tank. Cement would be added to the backfill preparation tank from a cement silo of approximately 116 cm (2.2 m diameter by 8 m height) by a screw conveyor. The backfill would be mixed for several minutes to ensure even mixing of cement within the tailings slurry. Water would be added to ensure the final cemented backfill mixture contained approximately 70% solids by weight. In the case of uncemented backfill, water would be added to the mixing tank, if required, to ensure a 70% by solids backfill mixture. The prepared backfill mixture would be pumped or flow into the main backfill lines directly from the plant.

16.19.2.2 Plant Design and Equipment

The equipment to be installed in the backfill plant is listed in Table 16.8. The plant would be automated and instrumentation would control the process. The plant would be equipped with a central control room where a control room operator would monitor the operation. The control room operator would also maintain plant housekeeping.

Table 16.8: Backfill Plant Equipment List	
Equipment	**Quantity**
Thickener	1 each
Flocculent System	1 each
Thickener U/F Pump	2 each
Backfill Mixing Steel Tank	1 each
Backfill Tank Mixer Arm and Motor	1 each
Process Water Steel Tank	1 each
Process Water Tank Pump	2 each
Steel Cement Silo c/w Dust Collector	1 each
Cement Screw Conveyor	1 each
Cement Weigh Hopper	1 each
Flush Air Receiver	1 each
Backfill Positive Displacement Pump	1 each
High Pressure Water Pump	1 each
Air Dryer	1 each
Instrument Air Receiver	1 each
Cleanup Pumps	2 each
2-tonne Overhead crane	1 each
Gland Water Pump	2 each
Actuation Valves	1 lot
Miscellaneous Small Equipment	1 lot
Instrumentation	1 lot

16.19.3 Other Facilities

A warehouse, backfill, and compressor building will be constructed from the shipping containers stacked two high for walls and a roof placed on top of the container walls. The building will have the dimensions of 10 m by 36 m. The containers will provide space for warehouse storage of smaller items. Larger items will be stored at the main warehouse near the processing plant. Items would be stored on a combination of pallet (large or bulk items) and shelved (smaller items) storage systems. Valuable items would be placed in a locked storage area.

A mine laydown yard will be constructed near the portal to store materials and equipment required for the underground mine. This laydown yard would have raised timber stands on which to place large material, such as screen, pipe, etc., as well as gravel graded areas for storing equipment and material. A cold storage building will store equipment requiring protection from the elements but will not require heating. Smaller supplies and components requiring heated storage would be placed in the warehouse at the mill.

All underground mine water would be sent to the water treatment facility and reused or discharged. All mine process water will be obtained by gravity clarification of mine effluent water in a 3-stage settling pond system. It is assumed that this system is of sufficient capacity to produce clear enough overflow that can be used for the underground equipment and to a lesser degree, the hydraulic backfill plant. Additional steps, such as the use of flocculants, might have to be considered should the clarity of the recycled mine water not be suitable for use in meeting the site process water demands.

Mine process water will be transferred from the settling ponds to the underground mine workings by a surface pump house feeding the main water distribution piping system through a water line located in a dedicated mine service raise.

Bottled water will be used to meet all of the potable water demand at the mine.

A fully equipped mine rescue station is required on the property and will be incorporated into the shipping containers near to the warehouse, compressors, and backfill building. The mine rescue station will be equipped with all the necessary equipment, including self-contained breathing apparatus, flame lamps, gas testing equipment, rescue equipment, etc., and supplies and chemicals required to operate the station. There will be enough equipment to, in an emergency, have two 5-person mine rescue teams operating or on standby at any one time.

The mine will be technically supported by the geology and engineering departments. The geology department will be responsible for mapping and interpretation, sampling of production drill holes, grade control, and ore reserve estimations. There will be a separate exploration group to undertake the exploration work on the property and to prove up new mineral resources for potential mining. The engineering department will be responsible for mine planning, production scheduling, surveying, geotechnical design, collecting, and reporting performance statistics for the mine and any other technical requirements that support the operation.

16.20 GRADE CONTROL

Underground grade control is critical as the ore in stopes will be delineated and grade tracked by grade control samples and not clear visible contacts. An on-site laboratory will ensure quick sample assay turnaround for the mine as well as the processing plant. The underground mine grade control programme will include:

- Geological chip sampling;
- Channel sampling of development and mining drift faces;
- Drill cuttings sampling from the blast hole drilling; and
- Grab sampling of LHD bucket loads and remuck bays.

16.21 UNDERGROUND MINE MANPOWER

Mine manpower, included in the operating costs, totals 45 people during pre-production to an average steady state of approximately 100 personnel at full production. Staff comprise 16 people of the total production period mine manpower complement and hourly employees make up the remainder. Table 16.9 shows the average manpower complements for the mine.

Table 16.9: Development and Stoping Manpower Complement

Position	Total Complement
Stoping	
Driller	4
Slusher Operator	2
Ground Support/Blaster	4
Helper	2
Longhole Driller	4
Blaster	2
Blaster Helper	2
LHD Operator	4
Haul Truck Operators	8
Lateral Development	18
Total Mine Development & Stoping Manpower	**50**

Item	Total Complement
Mine Superintendent	1
Mine Supervisor	4
Mine Trainer/H&S Coordinator	1
Chief Engineer	1
Mine Engineer	1
Mine Planning Technician	1
Ventilation/Surveyor Technician	1
Chief Geologist	1
Mine Geologist	1
Geological Technicians	4
Total Mine Staff	**16**

Support Services Manpower Costs ($)

Position	Total Complement	Annual Compensation ($)
Serviceman	4	$108,000
Grader Operator	1	$108,000
Construction/Services/Backfill Leader	2	$114,750
Construction /Services/Backfill Helper	2	$108,000
Lamproom/Dryman	2	$81,000
General Labourer	4	$81,000
Total Mine Support Services Manpower	**15**	

Underground Maintenance Department Manpower Costs ($).

Position	Total Complement	Annual Compensation ($)
Leadhand Mechanic	2	$121,500
Mobile Mechanic	4	$121,500
Mechanic	4	$121,500
Mechanics Helper	4	$108,000
Electrician	2	$121,500
Electrician Helper	2	$101,250
Stationary Mechanic	1	$121,500
Total Mine Maintenance Department Manpower	**19**	

17 RECOVERY METHODS

The past producing mine included a conventional processing plant comprising crushing, grinding, gravity concentration (jig) flotation, Merrill Crowe, leaching, and refining for gold recovery to doré bars. The crushing circuit and processing plant buildings and a majority of the old equipment exist though the plant has been vandalized mainly with all large cabling being removed.

As part of AMPL's review of the process flowsheet and processing plant, because of the noted (during mill site assessment) and reported (monthly reports) past problems with mill recovery and throughput, it was determined that the plant design proposed in the Kilborn Feasibility Study of 1986 and the actual plant built, differed in a number of aspects. The biggest change was that the grinding mills appear to have been bought not for purpose but probably for availability and cost, as both mills are the same and adapted to their differing roles. During AMPL's assessment, the actual grinding mills selected have effectively reduced the processing plant capacity to approximately 750-800 tpd. To achieve the rate of 1,000 tpd when open pit mining is anticipated would require the installation of an additional ball mill, which could potentially be accommodated relatively simply into the existing plant.

17.1 FLOWSHEET DESCRIPTION

The proposed process flowsheet is presented in Figure 17.1.

Ore from the underground mine will be delivered to the crusher dump hopper. A primary Jaw crusher will reduce the size of the run-of-mine ore to a nominal 120 mm. A cone crusher operating in closed circuit with a double deck vibrating screen will further reduce the size of the ore to 13 mm. A fine ore bin with a 1,500-tonnes live capacity will be used for ore storage prior to rod milling.

Ore from the fine ore bin will feed the comminution circuit, which consist of a primary rod mill and a secondary ball mill. The ore will be ground to a final product size of P80 of 100 microns. A Knelson concentrator will be incorporated in the grinding circuit to recover coarse free gold, which will be treated using an in-line-leach reactor. The leachate from the ILR circuit will be directed to the Merrill-Crowe circuit while the tailings will be sent back to the grinding circuit.

The rod and ball mills will operate in closed circuit with cyclones. The overflow from which will feed the flotation circuit will consist of roughers, scavengers, and cleaner cells. The final flotation concentrate will feed cyclones operating in a closed circuit with a re-grind mill. Cyclones overflow will feed a thickener. Thickened pulp will constitute the feed to the cyanide circuit. Scavenger flotation tailings will be used either to produce backfill or be pumped directly to the tailing impoundment area. The thickened flotation concentrate will be subjected to cyanidation utilizing four leach tanks operating in series. The residual leached tails will be sent to two drum filters also operating in series in order to wash the pulp and remove gold in solution. The final re-pulped filter cake will be pumped along with the flotation tailings to the tailing impoundment area.

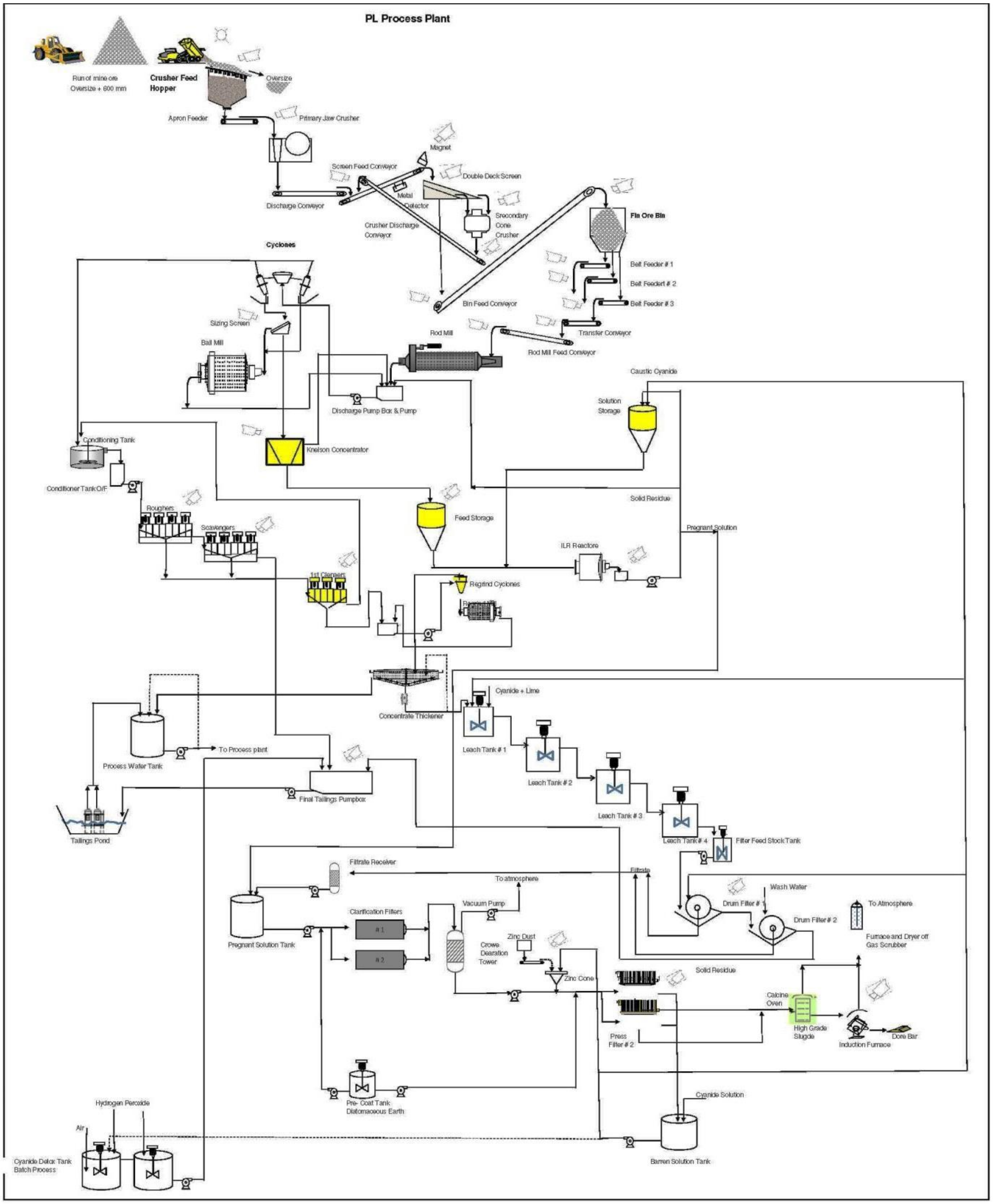


Figure 17.1 Processing plant flowsheet

The Merrill-Crowe process will be used to recover gold from the pregnant solution generated from the leaching process. Precipitate from the Merrill-Crowe process will be recovered using press filters and sent to the refinery for smelting using an induction furnace in order to produce doré bars. A portion of the barren solution will be treated with hydrogen peroxide in order to destroy residual cyanide in solution and sent to the final tailings pump box for disposal to the TMF.

17.2 PROCESS DESCRIPTION

17.2.1 CRUSHING

Run-of-mine ore will be dumped onto a stationary grizzly with the undersize feeding a 100-tonne capacity dump hopper. Ore will be withdrawn from the hopper using a variable speed apron feeder, which will feed a primary jaw crusher. Crushed ore will be conveyed to a single, double deck vibrating screen situated directly over the short head cone crusher. The screen undersize from the bottom deck will fall directly onto the fine ore bin conveyor and be transported to the fine ore fine while the coarse oversize from both decks will discharge through a chute to feed the short head cone crusher. The cone crusher will operate at a closed side setting of 10 mm. The crusher discharge along with the jaw crusher discharge will be combined to feed the vibrating screen.

17.2.2 GRINDING AND GRAVITY CIRCUITS

Ore will be withdrawn from the 1,500-tonne fine ore bin by means of three (3) variable speed belt feeders, which discharge onto the rod mill will feed conveyor. The mill feed conveyor will be equipped with a weigh scale and will be ratio controlled for water addition to the rod mill in order to achieve optimum density within the mill.

Rod mill and ball mill discharge will operate in a closed circuit with a cyclopack consisting of three operating and one standby cyclones. Feed to the cyclones will be via a variable speed pump in order to maintain constant flow to the cyclones. Two of the three operating cyclones underflows will feed the ball mill while one cyclone underflow will be used to feed a sizing screen for the Knelson concentrator. Cyclone overflows will be directed to the flotation circuit.

Oversize material from the Knelson screen will flow into the ball mill feed chute along with the cyclone underflow while the screen undersize will feed a Knelson concentrator. The Knelson will run continuously for several hours after which it will be taken off line for several minutes in order to flush out the unit. During operation, the discharge from the unit will go back to the cyclone feed pump box. The gravity concentrate from the unit will be pumped to a holding tank, which will feed an in-line leach reactor operating in a batch process mode. Leach times will be several hours after which the leached ore will be washed in order to remove residual cyanide and sent back to the grinding circuit while the leachate solution will be sent to the Merrill-Crowe circuit.

17.2.3 FLOTATION

The flotation circuit will produce a bulk sulphide concentrate utilizing a bank of rougher, scavenger, and cleaner cells. The cyclone over flow from the grinding circuit will be first pH adjusted with lime, if required. Reagents will be added to the conditioning tanks and sent to the first bank of rougher cells. Concentrate generated from the roughers and scavengers will feed a bank of cleaner cells. Tailings from the scavenger circuit will constitute final tails and be sent to the TMF. Concentrate generated from the cleaner cells will feed a re-grind mill operating in close circuit with a series of cyclones. Cyclone underflow will flow back to the re-grind mill for additional grinding while cyclone overflow will feed a thickener prior to leaching. Cleaner tails will be sent back to the conditioning tank for further processing.

The re-ground concentrate will be dewatered and thickened to obtain an underflow density of 50-55% solid prior to leaching. Thickener overflow solution will be sent to the process water tank for re-use in the grinding circuit while thickened underflow will feed the leach circuit.

17.2.4 CYANIDATION

The leach circuit consists of four leach tanks operating in series. Cyanide and lime will be added to the first tank along with the thickened pulp. Oxygen will be added to each leach tank in order to improve leaching kinetics. Total residence time will be approximately 48 hours. Discharge from the last leach tank will feed two drum filters operating in series. The filtrate from the drum filters containing pregnant leach solution will be pumped to the pregnant solution tank. A wash of recycled barren solution will be applied to the cake, which will be subsequently re-pulped in barren solution and washed on the second drum filter using fresh water. The filtrate from both drum filters will be combined and pumped to the pregnant solution tank. The filter cake from the second drum filter will be re-pulped using fresh water and pumped along with the flotation tailings to the TMF. In order to minimize a buildup of deleterious elements in the barren solution, which may affect leaching kinetics, a portion of the barren solution will be treated using hydrogen peroxide in order to destroy residual cyanide in solution and sent to the TMF. This solution will be replaced will fresh cyanide solution.

17.2.5 MERRILL-CROWE

Pregnant solution will be clarified using two pressure clarifiers. The clarifiers will be backwashed with water when the flow through the unit decreases. Diatomaceous earth will be used as the filter pre-coat media.

The clarified solution from the leach circuit and the in-line leach reactor will flow through a Crowe vacuum tower, which will be maintained under vacuum. Air will be removed from the solution to increase the reducing power during the precious metal precipitation process. Zinc dust will be added to the deaerated pregnant solution using a belt feeder.

17.2.6 REFINING

The solution will then be pumped to the precipitation press. The precipitate will be dried by air and then discharged. The precipitate will be mixed into batches with flux and charged to the induction furnace where it will be refined into doré bars containing both gold and silver values.

17.3 PROCESSING PLANT RECOVERY

Based on the metallurgical testwork and process presented in the previous sections, the expected gold recovery is 90% with a plant operating availability exceeding 90%.

17.4 EXISTING PLANT

17.4.1 AECOM ASSESSMENT

AECOM consultants in late 2011 inspected the processing plant buildings and equipment and prepared an estimate of rehabilitation work required and costs. Any references made to instrumentation and process logic controllers (PLC) have not been included in the following description because complete replacement of these systems, with state of the art technology, is included in this present re-start plan.

17.4.1.1 Ore Receiving/Crusher Building

This building, which is a conventional steel structure with insulated exterior metal cladding and roofing, appears to be in reasonable condition with the exception that the siding was failing in a couple of spots.

The ore bin inlet grizzly was covered with metal cladding and the grizzly and the inside of this bin could not be inspected. However, the exterior of the bin, as could be seen from inside the crushing plant, did not appear to have a significant amount of corrosion.

The major equipment inside this building appears in reasonable condition. The crusher motors could be rotated freely and the crusher components did not appear to be badly worn or corroded. This equipment would require a complete maintenance inspection and servicing by a millwright (or equivalent) prior to returning to service. Anticipated activities include drive belt replacements, checking of all bearings, and replacement, if required. The cone crusher lubrication system was noted to require more significant repair.

The building sump and wet scrubber pumps appeared to be seized and would likely require extensive servicing or replacement. The smaller electrical motors on the sump and scrubber pumps were missing.

Belt conveyor idlers appeared to have been well greased at the time of closure and most idlers can still be rotated. All idlers would need to be inspected and properly cleaned and lubricated prior to returning to service. Belt conveyor pulleys will need to be cleaned and lagging replaced. Conveyor belting has been removed.

The overhead bridge crane could not be inspected closely, since it was not accessible. However, it did appear to be in good condition based on what could be observed.

The dust control ducting and equipment appears to be in reasonable condition for re-use.

Most of AECOM's observations on the 25 kV electrical supply have been superceded, as extensive vandalism has taken place since their report was completed. The AECOM descriptions are, therefore, not included here.

The electric unit heaters, including the disconnect switches within the crusher building, appeared to be in good condition. It is anticipated that they could be re-used when the plant is re-started. Cables from the disconnect to the heater unit are still in place.

Instrumentation devices within the crusher building appear to be in reasonable condition for re-use, including field junction boxes and local push button stations. The apron feeder pull cord switches were still in place and reasonable for re-use.

In general, the main feeders and control cables to the equipment were cut and removed. However, power and signal cables, from the control stations to the individual motors and instrument devices, were still in place. These will require testing at the time of re-starting to confirm cable continuity. Re-wiring from the MCC to the individual equipment will be required, since all cables from the MCC trailer have been cut and removed.

The conveyor gallery structure appears to be in good condition with no significant corrosion. Access ramps and stairs from the ground have been removed, possibly to prevent access for safety and vandalism prevention. The gravity take-ups of the conveyors were noted to be laying on the ground near the structure. Most of the idlers and pulleys we tried to rotate could be rotated and some of the motors and gear boxes could be rotated. Other motors and drives were not accessible or may have been seized. Feeder and signal cables to the conveyor drives were cut and removed. Lighting fixtures that were in place had broken lenses and the bulbs were missing. Re-wiring and lens replacement is necessary. The weigh scale unit sensor and the weightometer controller are still in good condition, but required re-wiring. Pull chord switches for all the belt conveyors were still in place and suitable for re-use.

The internal wiring inside the control panel (OP-1) is in good condition; there is no sign of defects. However, several control and signal to field instruments and junction boxes were cut and removed. Several panel instruments, such as controllers, indicators, and recorders, have broken glass windows, which require replacement. The two annunciator panels are still in place and appear to be in good condition. Most push button and indicating lights stations have broken lenses and push buttons knobs were removed. These can be easily restored by replacing the defective/damaged component(s). The control panel enclosure is still in good condition and is reasonable for re-use.

All field instrument devices, with the exception of the control valves, appear to be in good condition but again, most of these instruments were obsolete. Field control stations, push button stations, safety switches, and junction boxes are still in good condition and are reasonable for re-use.

17.4.1.2 Main Processing Plant

Similar to the crushing building, this building is a conventional framed steel building with exterior insulated metal siding and roofing.

The crushed ore storage end of the building was notably wet due to precipitation entering around and into the bin. The bin and belt feeders also showed signs of precipitation damage/corrosion. The belt conveyors and feeders in this area were noted to be more corroded than in the ore crushing due to the wetter environment. The drives and motors would require a more thorough inspection and servicing than those in the crushing building. Conveyor belting on most of the belt feeders and conveyors will need to be replaced. The dust collector (wet scrubber) may be salvaged and re-used.

The grinding mill appeared to be properly mothballed, including removal and storage of the rubber liners, rods and balls, blocking of the mills, and heavy greasing of the ring and spur gears. Significant quantities of rods were noted to be stored on the grinding area floor that could be used in the mills. The main 500 HP drive motors were noted to be clean and could be rotated by hand. A spare 500 HP motor was located on the floor near the rod mill and appeared to be well lubricated and wrapped in plastic. The gearing from the main gear reducers on the grinding mills appeared to have been removed for long-term storage. However, we did not find any such gearing in the plant areas. The air clutch on the ball mill was noted to be missing and possibly stored with the gearing from the main gear reducer. The rod mill air clutch was in place as a complete assembly. Lubrication systems for both mills were noted to be present and appeared in reasonable condition.

The hydro-cyclone units, externally, appeared to be in good condition. The one hydro-cyclone that was open was noted to be clean inside and some of the rubber lining appeared lose; requiring some repair.

The flotation machines appeared to be clean and in good condition. The flotation blowers were noted to have some dirt build up and corrosion; however, they are in reasonable condition considering their location and service conditions.

Wash drum filters were washed out, drained, and in reasonably good condition. Corrosion was noted on the drum filter surface and piping. The barren solution and reclaim water line were properly hooked up with a drum filter. Vacuum pump, filtrate receiver, and the connection to the drum filters were in reasonably good condition. Air blowing of the pipe lines should be done before being brought back in to service.

As a minimum, all pumps and motors will need a closer inspection, cleaning of the bearings, and lubrication before putting them back into operation. Some of the pumps (mainly the sump pumps) are seized and will require more extensive overhaul or replacement. Some of the smaller process pumps are extremely corroded and need replacement, especially around the process tanks. The larger slurry and solution pumps are considered usable, subject to inspection and servicing.

Water tanks are drained and in fair condition for use. Process tanks inside are showing signs of corrosion and will require patch work (*e.g.*, the pregnant solution tank). In general, all the tanks appeared to be usable. Refer to the equipment list, which provides more specific information on the tank condition, as observed.

Compressors, tank, and dryer are in reasonably good condition. Servicing is required before putting the system into operation. The tank's auto drain filters will need servicing or replacement.

The dismantled tailing pumps are in good condition. The exterior above ground insulated tailings line was removed and will need to be replaced.

The furnace has been removed. The exhaust hood from the furnace was in place and re-usable. All other components required in the gold room have been removed.

17.5 PLANT PRESENT CONDITION

In March 2017, AMPL Professional Ltd. sent a team of engineers and operators to the PL site to inspect and develop the re-start plan for the PL Mine. This multi-disciplinary team spent several weeks assessing all aspects of the processing plant and infrastructure.

The main changes in and observed conditions since the AECOM assessment are:

- All major cable wiring has been stripped from the processing plant.
- The gravity circuit is not complete and is recommended to be replaced with a Knelson concentrator, a more efficient and secure modern technology.
- The Merrill-Crowe components were inspected and complete replacement is recommended.
- One drum filter in the processing plant has been partially dismantled and is beyond repair.
- The gold room is completely empty of equipment.
- The overhead cranes in the processing plant and maintenance shop building have been operated and work. They still require overhauls before being put back into service.
- The crusher building power substation and the MCC trailer have been vandalized beyond repair.
- The processing plant MCC room components require replacement.
- The processing plant and maintenance shop/warehouse lighting for the most part is functional.

17.6 TAILINGS MANAGEMENT FACILITY

The past employed Ragged Lake Tailings Management Facility, which is presently in the process of being added to Schedule 2 of the Metal Mining Effluent Regulations (MMER), designating the area as a tailings disposal area, will be recommissioned. This tailings disposal area requires an addition, before use, to the MMER Schedule 2, as the regulations were not in effect when the original mine was in production.

The tailings area encompasses a total area of approximately 0.5 km^2 and can hold 1 million m^3 of tailings and water at its present elevation of 346.0 masl. This figure will increase to approximately 2.3 million m^3 once the required control structures and dams to raise the operating level of the tailings basin to 348.0 masl are built. To be noted, there are three additional structures to be built to control water flows from the south end and the west side of the lake. These structures are indicated in Figure 17.2. Construction costs have been estimated and included in the capital estimates for all the required structures for the purpose of this report. Figure 17.2 presents a plan view of the tailings area.

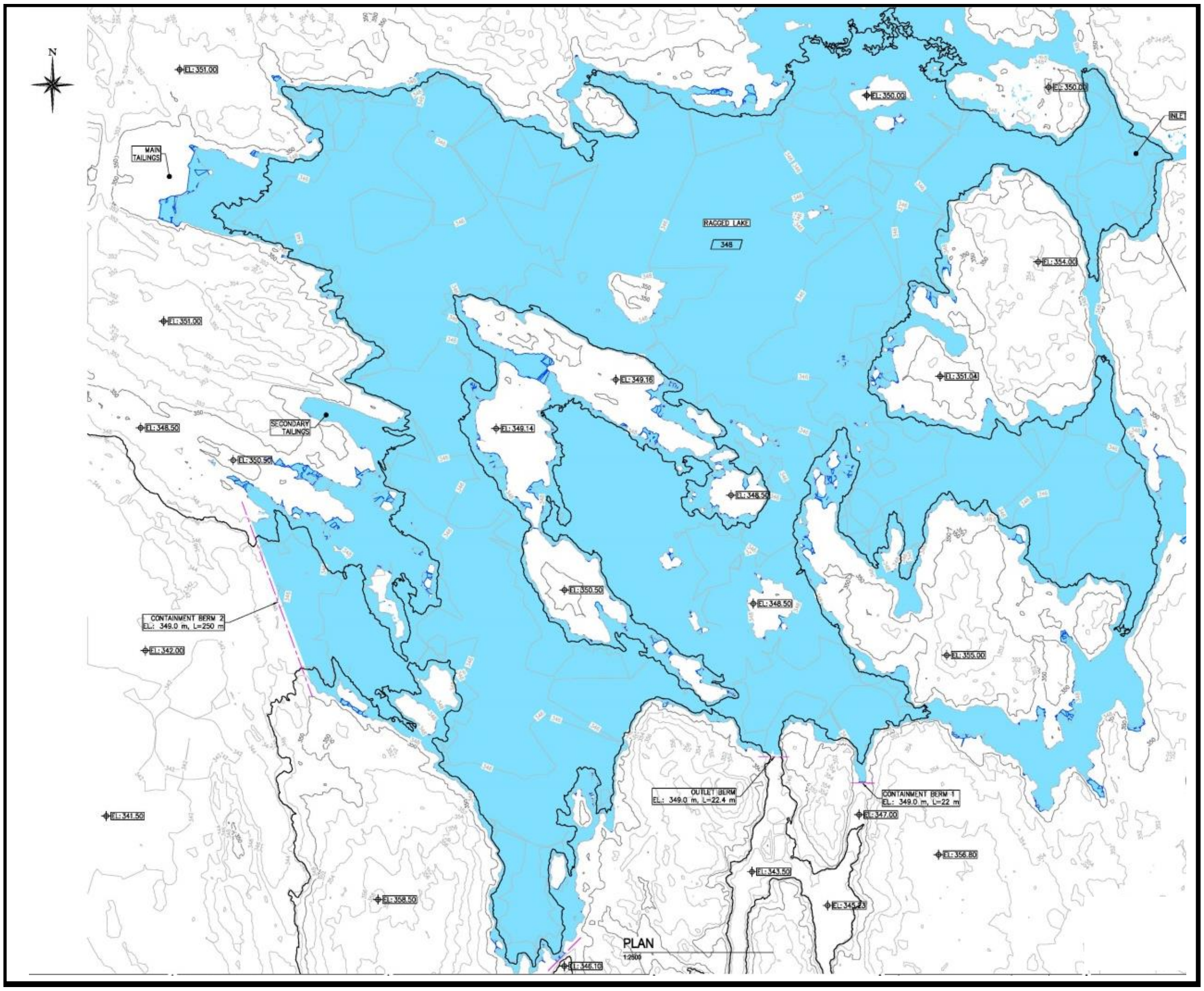


Figure 17.2 Ragged Lake TMF plan view

The area is generally shaped similar to a large flat bowl. All water drains from the north to the southwest of the area. There is presently one outlet with a 1 m high notched weir in place. The notched weir is constructed from concrete with wooden boards slotted into place to raise the level of the overflow point in the weir. The weir is presently overflowing. There are no dams in the management facility.

During the past operation, approximately 350,000 tonnes of tailings (approximately 200,000 m^3) were deposited in the TMF in the northern end, near the processing plant. The tailings are not completely covered in water but are surrounded by water.

During the past operation, a water reclaim pumping station was located approximately 0.5 km to the east from the tailings discharge point with power supplied to the pump by an overhead transmission line. The transmission line is in place though it requires some refurbishing and the addition of a transformer.

A preliminary design for the TMF was prepared by WESA.

As part of the tailings management requirement for the proposed Puffy Lake Gold Mine complex, tailings from the mining operation will be pumped directly into the Puffy Lake Mine Tailings Management Area (TMA), located south of the plant. The water storage in the TMA will be controlled by constructing an outlet structure at the outflow area of the TMA shown on in Figure 17.3.

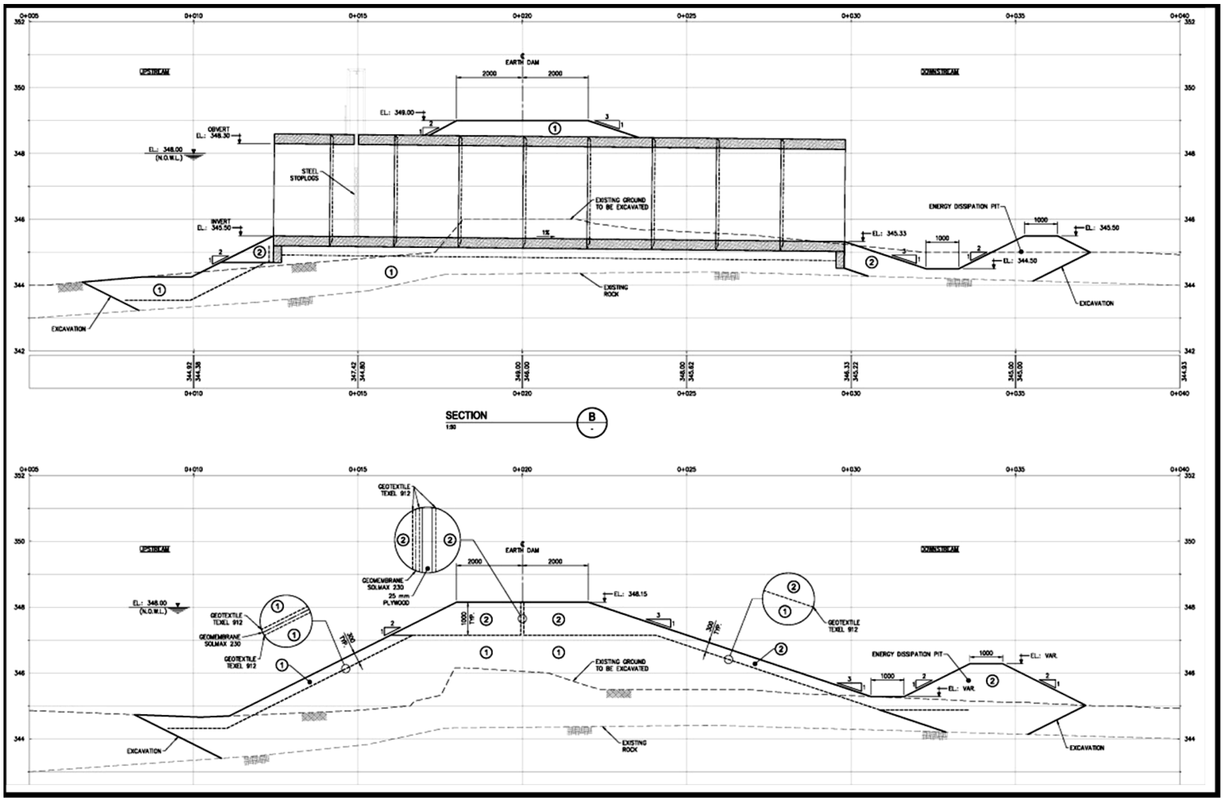


Figure 17.3 TMF dam's construction general arrangement

The work objectives included provide a hydrotechnical analysis of the TMA watershed to assess the inflow design flood and developing the outlet control structure concept.

17.6.1 DESIGN CRITERIA

The concept design of the proposed outlet structure are based on the following criteria:

- The structure must be able to control the water elevation at 348.0 m in the upstream TMA;
- The structure must be able to discharge the design flood without overtopping and/or failure;
- The structure should need no and/or very little operation except in an emergency situation;
- The structure should require very little maintenance during its design life;
- The design material used should be suitable for semi-permafrost zone foundation conditions;
- The construction of the structure should be economical and cost effective;
- Most of the construction materials used should be available at the site; and
- The design of the structure should be ecologically friendly.

17.6.2 HYDRO-TECHNICAL ASSESSMENT

17.6.2.1 Watershed Description

The TMA is located immediately south of the proposed plant and sits on the divide between Churchill and Burntwood River Basins in the west central area of Manitoba. The TMA watershed is part of the Canadian Shield and drains an area of approximately 3.2 km^2. This value has been estimated from topographic maps at a scale of 1:50,000. The TMA flows directly into Puffy Lake, approximately 1.5 km downstream through a narrow reach about 10-15 m wide with steep sides. The proposed outlet structure is located at the head of that stream. The normal operating water level of the TMA will be 348.0 m when the mine is in operation.

Land use within the watershed is divided between predominantly wooded (59.9%), marsh land (12.9%), bedrock (12.0%), and water bodies (15.1%). The main hydraulic characteristics of the TMA watershed are listed below:

- Total Area of the Watershed (TMA included): 3.17 km^2
- Average Area of TMA: 1.54 km^2
- Size of the TMA Watershed (TMA not included): 1.63 km^2
- Length of the Watershed: 2 km
- Average Slope of the Watershed: 0.55%

The following parameters were calculated based on those watershed characteristics, following the Quebec Ministry of Transportation (MTQ)[1] design guide:

- Run-off Coefficient Cd: 0.33
- Time of Concentration ("TC") of the watershed: 2.27 hours

The TMA storage curve has been developed based on LiDAR waterline elevations and is presented in Figure 17.4.

[1] Quebec design guide "Manuel de Conception des Ponceaux (MTQ)" was used as a reference material. An equivalent Manitoba design guide is not available.

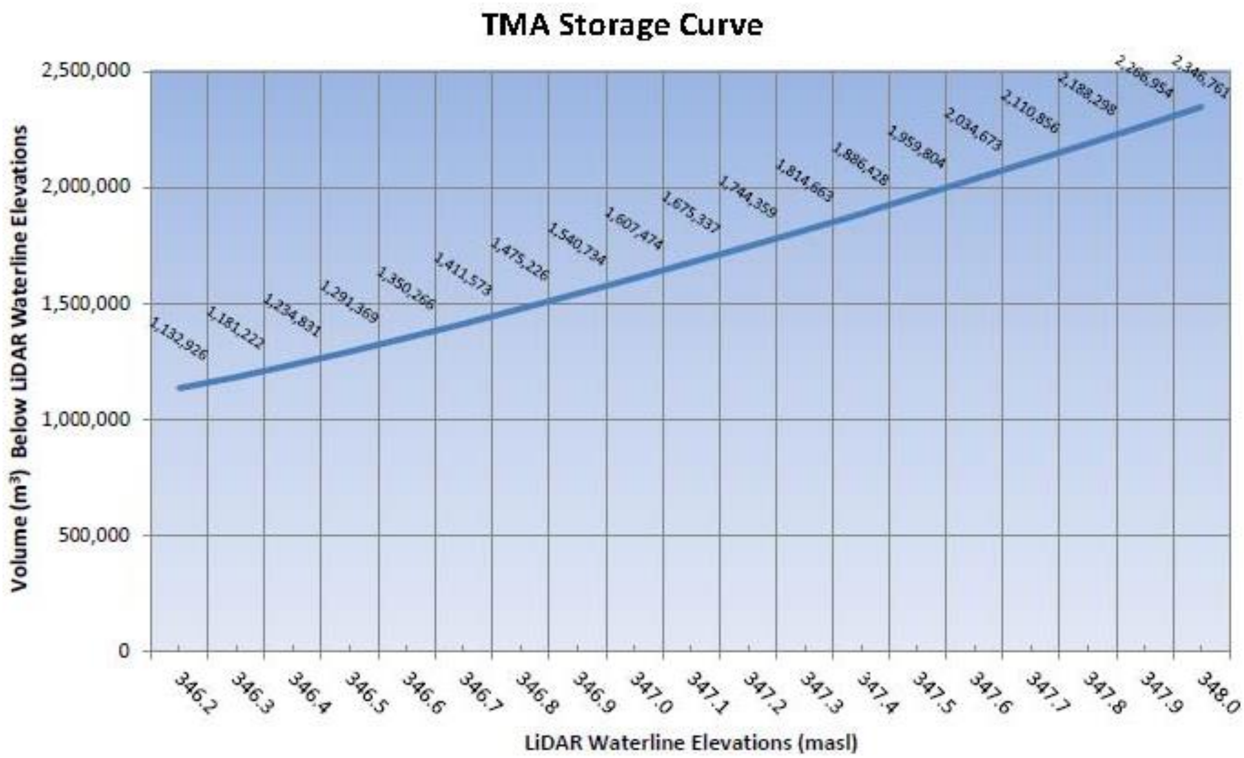


Figure 17.4 TMA storage curve

17.6.2.2 Dam Failure Risk and IDF

Based on topographic maps, at scales of 1:50,000 and aerial views from the software Google Earth™ downstream of the TMA outlet structure, the area seems mostly composed of wooded land and bedrock outcrops common to the Canadian Shield, with a large marshland directly upstream of Puffy Lake. As there are no local population areas or infrastructures at risk of being destroyed or heavily damaged in the event of a dam failure, it is reasonable to assess the level of consequences for dam failure as **MINIMUM** based on the Lakes and River Improvement Act. Consequently, a 1:100 year flood flow as the IDF is recommended for this structure design.

17.6.2.3 Determination of Flood Flows

Given the absence of a gauging station located on a watershed with an area comparable to that of the TMA, the rational method was used to determine the design flow rate.

The 100-year storm event was determined using the IDF curve from the Short Duration Rainfall Intensity-Duration-Frequency Data from Environment Canada for the Island Falls #4063560 meteorological station located 100 km northwest of the TMA. A rain intensity of 21 mm/hour corresponded to the estimated time of concentration (t_c = 2.27 hrs). The maximum flow corresponding to the 100-year storm event was determined to be 6.10 cms using the rational method. However, since the watershed includes storage due to the lake and wetlands, the maximum flow must be adjusted to allow for attenuation by lamination and, therefore, a value of 3.30 cms was calculated according to the MTQ guidelines. As shown in Table 17.1, maximum and laminated flows were estimated for additional return periods of 2, 5, 10, 25, and 50 years.

Table 17.1: Outflow for Different Return Periods		
Period (years)	**Maximum Flow (cms)**	**Laminated Flow (cms)**
2	2.87	1.55
5	4.21	2.27
10	4.91	2.65
25	5.14	2.78
50	5.78	3.12
100	6.10	3.30

The 100-year storm event is estimated to increase the volume of water in the watershed by 150,400 m^3. If this water is added to the lake at the same time, the water level elevation would rise to 348.18 masl, according to the storage curve presented in Figure 17.6 below.

17.6.3 Outlet Control Structure Concept

17.6.3.1 Description of the Control Structure Concept

The proposed structure is shown in the plan and in this section on Figure 17.2 and Figure 17.3. The control structure consists of an embankment dam composed of a steel stop log concrete culvert sluiceway section and a free overflow rock fill weir section.

The core of the dam is composed of 0-100 mm diameter granular fill with gentle slopes both upstream and downstream. The overflow surface of the weir will be covered by a 200-600 mm diameter rip-rap to protect against wave action. The dam will be covered with an impervious geomembrane that will be protected against perforation by layers of geotextile. The crest of the dam will be at elevation 349.0 masl, which corresponds to a minimum freeboard of 0.5 m.

The suggested sluiceway will consist of a pre-cast concrete box culvert 2.5 m wide and 3.5 m high. The top log elevation will be set at 348.0 masl, which is the proposed water level in the TMA (to allow for normal flows to be routed). The top of the culvert opening will be at an elevation of 348.3 masl.

The overflow rock fill weir will be 10 m wide and its crest will be at elevation 348.15 masl, which will allow flow from the 100-year storm event to be discharged. This design is favourable for cold weather conditions because the culvert will not be underwater in the winter months and will, therefore, allow for continuous flow throughout the year.

17.6.3.2 Discharge Capacity and Water Management

The discharge capacity of the box culvert and overflow weir depends on the upstream water level and the number of logs installed in the box culvert. The water level in the TMA will be maintained at an elevation of 348.0 masl with all logs in place. The dimensions of both hydraulic structures are presented in Table 17.2.

Table 17.2: Hydraulic Structure Dimensions		
Structure	**Width (m)**	**Crest Elevation (m)**
Box Culvert (with logs)	2.5	348.00
Overflow Weir	10.0	348.15

The rating curves illustrating the relationship between the discharge flow and the upstream water level, based on the number of logs in the sluice, are presented for the complete system on Figure 17.5. Each log is assumed to be 0.2 m in height. The graph shows that the laminated flow for the 100-year storm event can be discharged while maintaining a freeboard greater than 0.5 m and keeping all logs in place. Each log removed can approximately provide an extra cubic meter per second of discharge capacity.

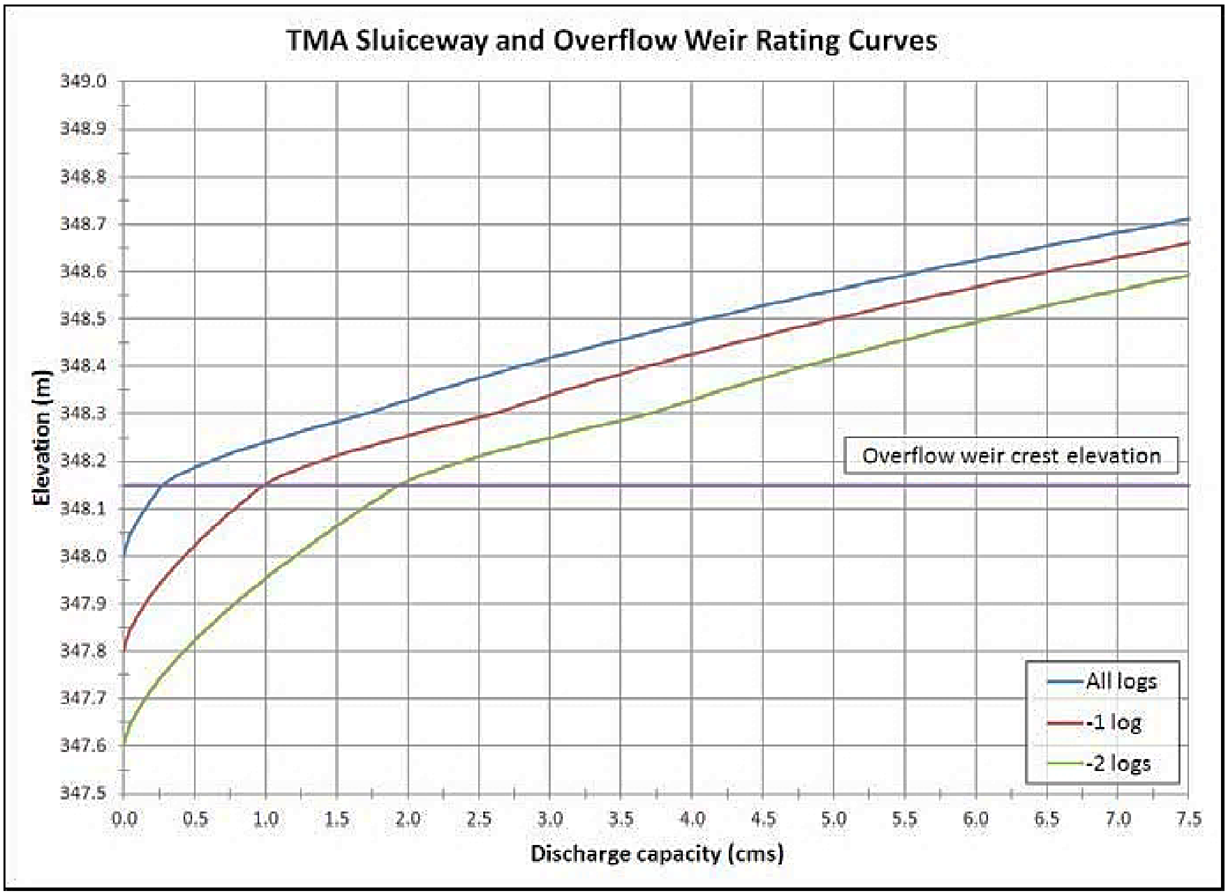


Figure 17.5 Sluiceway and overflow weir rating curves with logs

The rating curve of the complete system when no log is in place, such as in the case where the water level of the TMA would need to be lowered considerably, is presented on Figure 17.6.

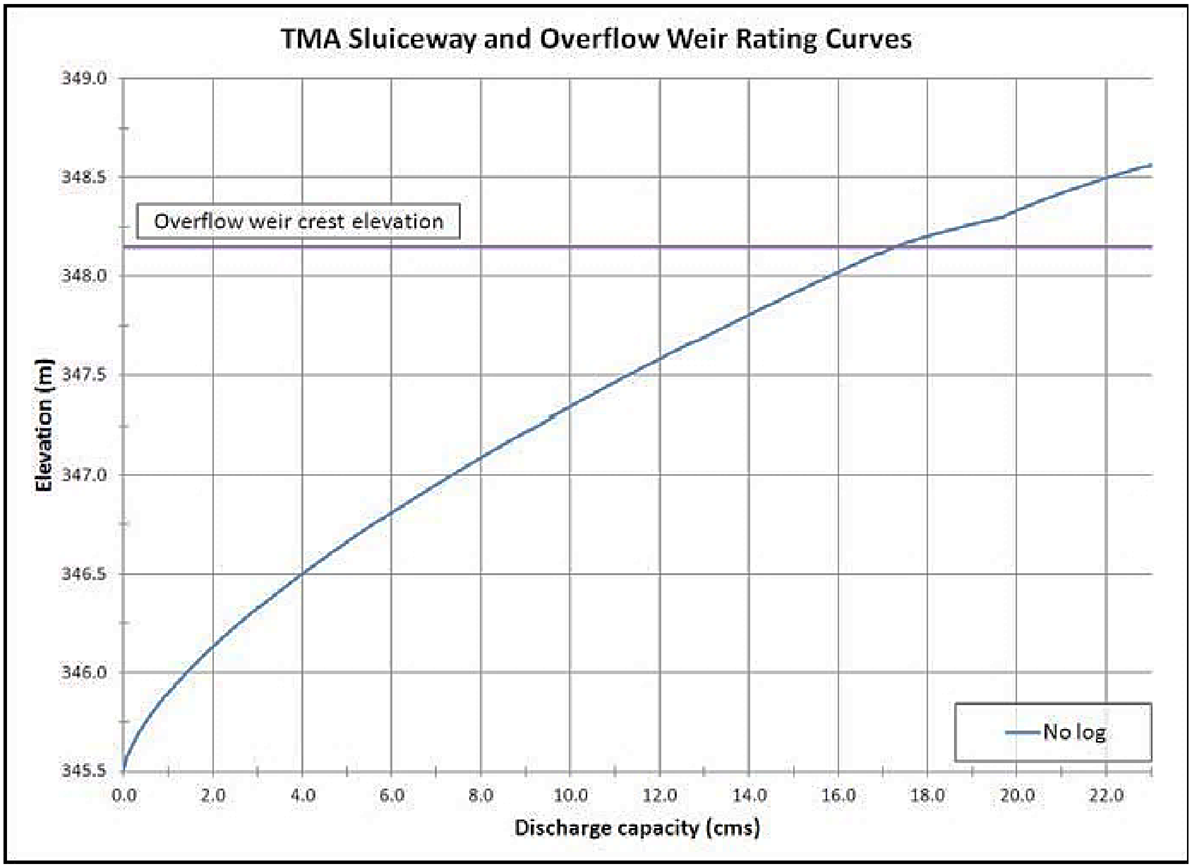


Figure 17.6 Sluiceway and overflow weir rating curve with no log

17.6.3.3 Flow Monitoring

The rating curves presented in the previous section would be used to provide flow monitoring in the outlet control structure. The approach would be as follow:

- Install a water level gauge data logger;
- Proceed with 3-4 on-site flow measurements (Doppler); and
- Calibrate the rating curve and extract flows in relation to the continuous water level readings.

17.6.3.4 Alternate Design Consideration

As a secondary design proposition, a “bottom draw” structure could be installed instead of the box culvert. This structure would consist of a steel pipe installed below the frost level beneath the dam. Its intake would be located just upstream of the dam with a concrete drop inlet and a butterfly type valve that would be installed in the middle of the crest of the dam to easily control the outgoing flow. A concrete outlet on the downstream side of the dam would also be necessary. This option appears to be less suitable for winter conditions, as the drop inlet and the butterfly valve could freeze up.

It should be noted that when a detailed survey of the site is conducted, additional freeboard dykes around the low lying areas of the TMA may be required to retain the design storage volume.

17.7 PROCESSING PLANT AND TAILINGS MANAGEMENT MANPOWER

The processing plant will be operated on 2 shifts by 8 people, as shown in Table 17.3. The processing plant staff will total 9 people, as listed in Table 17.4.

Table 17.3: Processing Plant Operators

Position	Total Complement
Lead hand/control room	4
Crushing	4
Grind/flotation	4
Leach/filtration	4
Total Processing Operations Manpower	**16**

Table 17.4: Processing Plant Staff

Position	Total Complement
Mill Superintendent	1
Metallurgist	1
Foreman	1
Assayer	2
Technicians	4
Total Processing Staff	**9**

Table 17.5: Process Plant Maintenance Manpower

Position	Total Complement
Mechanics	4
Electricians	1
Instrument Technician	1
Helpers/labourers	2
Total Maintenance	**8**

18 PROJECT INFRASTRUCTURE

18.1 EXISTING INFRASTRUCTURE

The PL Mine, as a past producing mine, has significant infrastructure in place, which will require refurbishing and upgrades as well as additions and expansions to the existing facilities.

The access road from the paved highway is the same road as from the past operation. The roadway has few functional culverts in place. Near the mine property, one water crossing has a large steel plate structure laid on top of the roadbed to prevent roadbed erosion. The majority of the road material in the existing roadbed is sand. In the spring time, the road is almost impassable, turning into mud, especially in the low sections where no drainage has been provided. The road is not ditched and is not wide enough along its entirety to facilitate 2-way traffic for tractor trailer trucks.

Amps Powerline, a local powerline construction company (of which some senior employees previously worked for Manitoba Hydro), conducted a site survey of the PL Mine powerline in March 2017. The company assessed the damages caused by the forest fires and vandalism, after the mine site was placed in care and maintenance. A number of years previous to 2017, Manitoba Hydro had begun to re-establish the powerline (paid for by Minnova) in anticipation of a potential re-start at the PL Mine. Manitoba Hydro installed new power poles to the property line; however, they did not install missing conductors or the step up transformation and metering required. There are approximately 300 poles in the 20 km line from the electrical utility supply in the town of Sherridon to the PL Mine site. About 10 km of the 25 kV overhead line route is missing the conductor and in many places, the conductor is laying on the ground. There are pole lines on the property, with some conductor in place, facilitating power supply to the mine portal area, processing plant, and tailings water reclaim.

The electrical power main substation is located adjacent to the processing plant/services building. Checks of the equipment by Siemens indicate that the transformers are in good condition and with minor refurbishment can be placed back into service. The main MCC room in the mill has been vandalized and will need to be upgraded or replace. The crusher building substation is not serviceable as it has been widely vandalized (including several bullets holes in the transformer) and both the transformer and MCC rooms will need to be replaced.

The processing plant/services building houses the maintenance shop, warehouse, and offices/dry complex separately at one end of the building.

The maintenance shop and warehouse areas, which are one-half each of the same section of the building, are each 24 m long by approximately 10 m wide by 10 m high. The maintenance shop has 3 overhead roll-up doors (which require replacing) of 4.8 m wide by 4.5 m high, one man door, and a 10-tonne overhead bridge crane (bridge is 7 m above the floor elevation). The crane is functional but needs checking and refurbishing, as required. The warehouse is equipped with a 2.4 m wide by 4 m high roll-up door and a man door next to the roll-up door. The warehouse area has no warehouse related items contained within it. There is presently the potable water tank (1.8 m diameter by 4 m high) for the mine offices/dry complex located in one back corner of the warehouse building. There is a 3 m by 3 m walled in services counter area and fencing gate (to restrict access to the main warehouse area) at the front of the warehouse. The warehouse is separated from the maintenance shop by a concrete block wall, floor to roof of the building.

The office/dry complex is adjacent to the maintenance shop and warehouse complex. The ground floor of the building comprises the dry area and what was the mine crew line up area. There is the main male dry area consisting of a clean clothes locker area, dirty clothes hanging basket area, and shower, sink, and toilet facilities. The female dry area consists of a small clean clothes locker area, dirty clothes hanging basket area, and two showers and sinks and toilets facilities. In both dry areas, lockers and baskets and associated hardware have been removed. The T-bar ceilings in most of the areas have been removed and lighting and ventilation systems substantially vandalized and removed. The dry wall in all areas is damaged (to varying levels of destruction) and in some places removed. The crew lineup area has had everything removed and is an empty shell. The floors will need to be redone in all areas. The staff dry and mechanical room are on a first floor level with the staff dry almost completely intact and only requiring refurbishing of drywall and fixtures. Lockers are in place in the clean clothes area but hanging baskets will be required in the dirty clothes section. The floor is in good shape.

The mechanical (HVAC and water heaters) room is in excellent condition and requires minor refurbishing of equipment, which all appear in good condition for recommissioning. The water heaters will need to be re-wired and re-plumbed, as both wiring and copper piping has been vandalized.

The office area walls, doors, HVAC, electrical wiring, and fire protection have been almost completely removed. The outside windows have had their glass removed and are boarded over. The washroom area has had all fixtures removed, as well as the walls and doors. There is one large walled in area where the dry wall is in good condition with only cleaning, repainting, and re-establishing of electrical and HVAC services required.

The assay laboratory building, located adjacent but separate from the office/dry complex area of the main processing plant/services building, is serviceable and a portion is presently being used as the diamond driller’s storage area. The building has a sample receiving and preparation area, wet analysis area with fume hoods, a furnace room, offices, and mechanicals room. The building’s work benches and sinks are in place and the electrical and HVAC requirements are functional. Some refurbishing and upgrades will be required. Complete re-equipping of the laboratory tools and equipment will be required.

The warehouse/mill laydown area, with some timber racks and old parts left on the racks, is adjacent to the maintenance shop area. The area has grasses and weeds covering many areas but appears to be in useable condition.

The exploration camp, consisting of 16 single rooms, a central washroom and shower area, kitchen and eating area, television room, and satellite communications room, is located by the main gate to the property and approximately 60 m from the mill building. The camp consists of connected trailers and is propane heated. Each accommodation room consists of 2 beds, 2 closets, a desk, and a chair. The kitchen has food preparation counters, a large electric cooking grill, fridges, and freezers to cook for about 20 people. The eating area next to the kitchen can comfortably seat about 15 people. The television room consists of several couches, a flat screen television and DVD player, and a large coffee table. Water is supplied to the camp from containers filled at municipal water sites and trucked to the site.

The original sewage treatment facility, consisting of a septic field fed by a disc processor, is located several hundred feet from the end of the main plant building office/dry complex end. The sewage treatment facility will need to be replaced; however, the septic field has been tested in 2017 and appears functional. The exploration camp is not connected to it, with sewage trucked to the nearest sewage treatment facility in Sherridon.

18.2 PROJECT INFRASTRUCTURE

Infrastructure required for the PL Mine will include:

- Upgraded all season 9 km gravel access road;
- Site road upgrades;
- Completion of power supply and distribution lines refurbishment;
- Recommission of the main power substation and MCC room and installation of a new crusher building substation and MCC room;
- Refurbishment of the mechanical workshop, office, dry complex, assay laboratory facilities, and other service areas and equipping of these facilities;
- Expand the exploration camp to 32 person capacity (first 6 months of pre-production period) and complete expansion of the camp to accommodate 150 people for operations period; and
- Installation of a potable and service water supply and treatment facilities.
- Installation of sewage treatment facilities for the main camp and the mill/shop/office complex.

The proposed site layout is shown in Figure 18.1 and Figure 18.2.

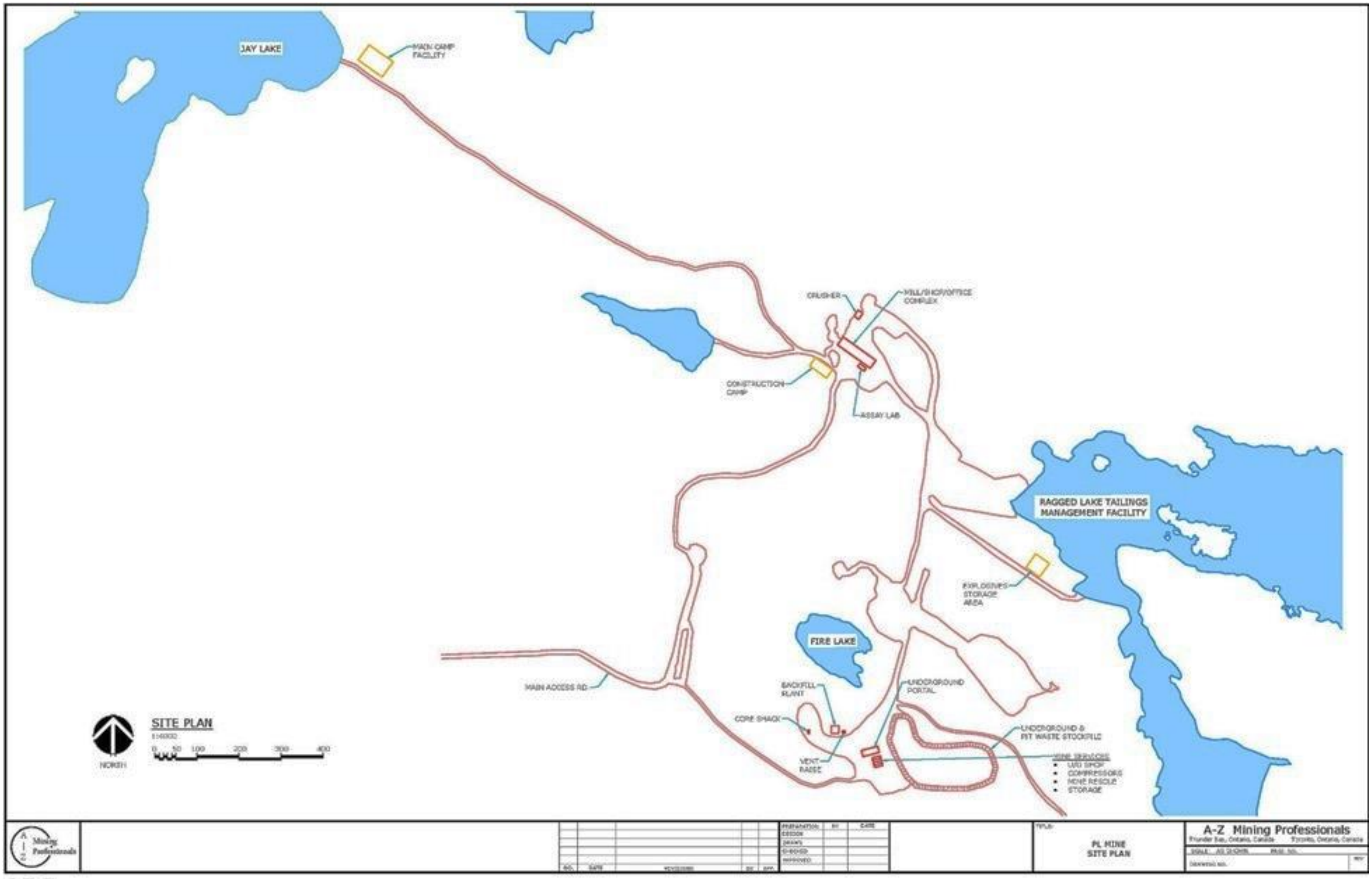


Figure 18.1 Site plan overview

A
Z
Mining Professionals

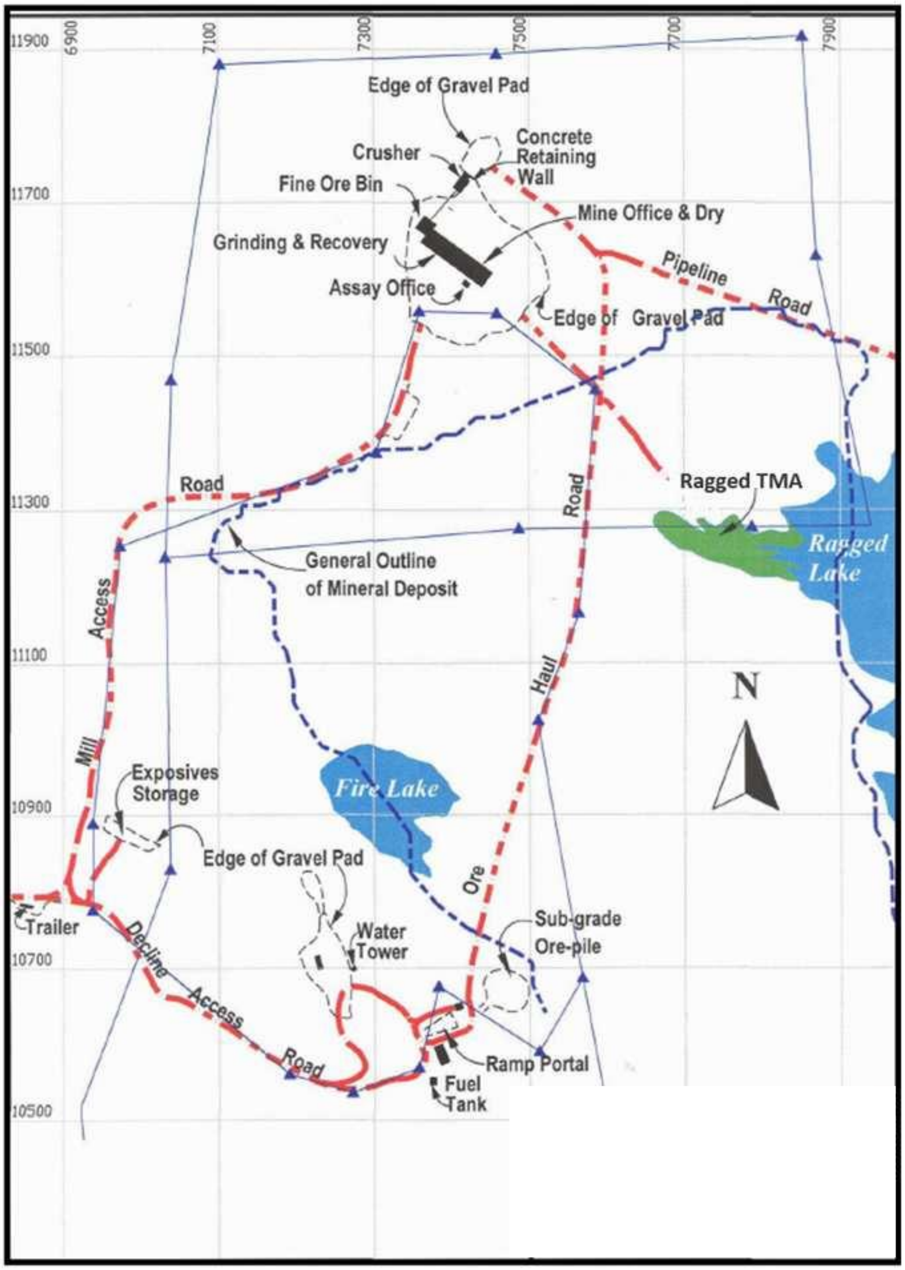


Figure 18.2 Site plan

18.3 ACCESS ROAD

The existing road will be upgraded to an all season road as follows:

- Install 8 - 0.6 m diameter corrugated metal pipe water drainage culverts;
- Install 2 - 0.9 m diameter corrugated metal pipe water drainage culverts;
- Install 2 (KM 9) - 1.6 m diameter corrugated metal pipe water drainage culverts;

- Install a coffer dam and three 20 m × 1.6 m diameter corrugated metal pipe culverts at KM 5.1;
- Build up the road sub-base with coarse crushed rock placed along the total length of present roadway alignment. Widen the roadway to allow 2-way traffic passing widths for large vehicles;
- Construct ditching, as required, for a total length of approximately 400 m along the roadway to divert the surface water;
- Adjust roadway alignment to eliminate one S bend at KM 1 and S bends at KM 8;
- Add 0.3 m of 51-102 mm crushed rock on top of the sub-grade crushed rock for the entire length of the road; and
- Top cover the roadway with 0.15 m of 19 mm gravel for the entire length of the road.

A contractor will be used to construct the road.

The coarse and crushed rock will be obtained from the small quarry already established from the past operation along the roadway. The contractor will drill and blast material, as required, and use a portable crushing plant to produce the required rock sizes for use in the road construction.

18.4 SITE ROADS

The existing on-site roads are of a similar construction to the access road and will require upgrading similar to that proposed for the access road. Drainage culverts will be installed, where required, for water crossings and course rock base and gravel top cover will be applied for the all season roads. The road to the mine will facilitate 2-way truck traffic and other roads will allow for 1-way, small vehicle traffic. Approximately 5 km of site roads require upgrading.

18.5 POWER SUPPLY AND DISTRIBUTION

The existing main powerline refurbishing will be completed from the Sherridon substation to the site's main substation as well as completing and re-establishing of the on-site power distribution lines.

18.5.1 MAIN POWER SUPPLY TO SITE

At the Sherridon substation, 12.47/24.9 kV step up transformers will be installed to power the main powerline to the mine site. On the main powerline, the 25 kV conductor will be checked with most of it expected to be placed back into service. Missing overhead wire for approximately 10 km and missing pole hardware (not installed by Manitoba Hydro) will be installed. All old polymer insulators (small brown plastic insulators) on existing poles will be replaced with modern insulators.

Manitoba Hydro will require a metering structure at the connection point adjacent to the mine's property entrance with the costs for this borne by Minnova. A customer installed and owned 25 kV gang operated switch will be installed at the metering point as well a second 25 kV gang operated switch will be installed, owned, and controlled by Manitoba Hydro, may be required by the utility.

18.5.2 ON-SITE POWER DISTRIBUTION

The 25 kV overhead power line from the Manitoba Hydro connection point near the property boundary to the site's main power substation and the branch line to the mine portal requires repairs and upgrades.

Between both lines, 11 utility poles must be replaced and the 25 kV conductor be checked and with most of it expected to be placed back into service. The overhead 300 kVa transformer bank will be rebuilt or replaced. The old polymer insulators (small brown plastic insulators) on existing poles will be replaced with modern insulators. The existing gang operated switch at the mine entrance will be removed. All 25 kV dip poles to site transformers will be replaced, as they have been cut by vandalism. Each transformer on-site will have its own new gang operated switch as an isolation point.

The main 25 kV distribution substation consists of 2 transformers, one 25 kV to 4,160V (1,500 kVa), and one 25 kV to 600 V (1,500 kVa) transformer, located near the processing plant building. Both transformers tested (by Siemens) satisfactorily, but it was noted that there is some missing cabling, wiring, and some surface rust but otherwise both transformers appear to be in good condition. The 25 kV to 600V oil is good and can be operated as normal. The 25 kV to 4,160V oil test detected acetylene at 8 ppm and should be re-sampled when the units are placed back into operation to determine the severity of the problem. This substation provides power to the mill building, including the offices/dry complex, maintenance shop, and warehouse areas, assay laboratory building, exploration/construction camp, and the sewage treatment plant.

The crusher building pad mount transformer will be replaced, due to vandalism, with a mining skid installation. The substation will consist of an outdoor oil filled 1,000 kVa/600/347 to 120/208V transformer and switch gear and include a station for incoming connections, incoming switch, and secondary breakers. This substation will power the crusher and ancillary equipment in the crusher building.

A new substation will be installed near the mine portal and connected to the 25 kV overhead branch powerline. This substation will comprise an outdoor oil filled 2,500 kVa/4,160/600 to 120/208V transformer and switch gear. This will include a station for incoming connections, incoming switch, and secondary breakers and all skid mounted. This will power the surface ventilation fans and heaters, surface shop, warehouse storage, and mine rescue building, and the main 4,160 underground power feed cable feeding all the underground electrical equipment.

The camp and pump house transformer station will comprise an outdoor oil filled 1,500 kVa/600/347 to 120/208 transformer and switch gear. This will include a station for incoming connections, incoming switch, and secondary breakers and all skid mounted. This will power the one hundred (100) person camp for the operations period and the supply water for the processing plant offices/dry complex and camp as well as the sewage treatment facilities for the camp. The transformer will be connected to the main substation via a 1.5 km overhead powerline as the camp and pump house will be located close to Jay Lake.

18.6 SURFACE MAINTENANCE SHOP AND WAREHOUSE

The existing shop and warehouse areas are located at one end of the processing plant building and will be refurbished and equipped to function. The two facilities share a common centre wall.

The maintenance shop requires limited refurbishment with replacement of the 5.0 m wide by 6.5 m high roll-up doors, repairs to the heating ducting, and refurbishing of the operational 5-tonnes overhead crane. Work benches are in place and the shop will be provided with all equipment (welding equipment, vices, hydraulic hose making equipment, etc.) and tools required by the mechanics performing maintenance functions.

The warehouse will have pallet racking and shelving storage installed to store materials on. The shelving will include storage drawers for smaller items, including screws and nuts and bolts. The pallet racks will have levels and shelving will have shelves. An area at the back of the warehouse, where the roof is lower, will be walled off with heavy metal screen, with a locked gate to provide a locked storage area. The roll-up door to the warehouse will be enlarged to 3.6 m wide by 4 m high and a new roll-up door installed. The warehouse wicket area will be re-established with a counter and lockable window.

To provide offices for the shop and warehouse (which do not presently exist), 2-20 foot shipping containers will be located against the common wall between the shop and the warehouse, on the warehouse side of the wall. Entrances to the shop offices will be knocked through the block wall facilitating access to the offices from the shop side of the building. A set of stairs will be provided to access the top of the shipping containers where less used and heavier spare parts and motors can be stored. A new potable water treatment plant will be installed in a 20 sea container at the far end of the shop with the tank being located on top of the container.

18.7 OFFICES AND DRY COMPLEX

18.7.1 GROUND FLOOR

The ground floor of the office complex part of the processing/services building will house the men’s and women’s dry complex, mine and maintenance crews lineup room/wickets and supervisor’s desks, lamp room, mine and maintenance general foremen’s offices, a mine clerk’s offices, and maintenance shop lunch room.

The main hourly and women’s clean clothes locker areas, dirty clothes baskets change area and showers, toilets, and sinks located on the ground floor of the office building complex (next to the maintenance shop and warehouse) require extensive refurbishment and re-equipping. Plumbing requires minor repairs with taps and fixtures to be replaced. Most toilets and sinks, if not cleanable, will be replaced as will all shower dispenser systems. Dry wall in many areas needs repair or replacement and all walls cleaned and painted. The T-bar ceiling in most areas needs to be reinstalled as does ventilation ducting and electrical wiring and fixtures. Lockers and baskets are required for both the men’s and women’s clean clothes areas. In all areas, the floor will be refinished with a non-slip coating.

A new lunch room for the maintenance shop will be created from the old maintenance dry area, which has had all the fixtures removed. This will require the T-bar ceiling, wiring and lights, and the ventilation system to be re-established. The dry wall requires minor repair, cleaning, and painting. Remaining plumbing can be adapted, as required, for lunchroom piping requirements. The floor will be re-surfaced with a non-slip flooring. Lunch table and chairs, cabinets, coffee machine, microwave, etc. will be provided.

Mine and maintenance general foremen's offices and mine clerk's office will be created in the stripped out area adjacent to the lunch room. Work on the ceiling, walls, and floors is similar to elsewhere and walls will be constructed creating the office spaces with doors. Plumbing in this area will be decommissioned.

The area by the back stairs to the main second floor office area will become the lamp room. Cladding and insulation on the outside wall will be replaced. Dry wall requires repair, cleaning, and repainting and the floor re-surfaced with a non-slip coating. A wall and door will be installed to create a survey storage unit under the stairs. Benches will be installed along the outer walls and light charging stations installed on the opposite new wall by the stairs. The existing doorway to the outside will be modified to provide access to a new electrical shop and to the outside, using modified sea containers abutting the outside wall by the doorway.

The line-up room will be located next to the lamp room. This area will consist of three 2.4 m wide by 2.1 m deep areas with wicket counters and short walls separating each counter area from the next. Locking drawers and storage cupboards will be provided for the supervisors under the counter. This area flooring will also consist of a non-slip coating applied directly to the concrete floor. Other work will include re-establishing lighting and ventilation and heating as well as repair, cleaning, and painting of the dry wall and replacement of two pieces of the building cladding. The main mine tag board will be located on the wall across from the wickets.

18.7.2 Second Floor Areas

The staff dry complex is almost completely intact and in generally very good condition. The walls require some repair, cleaning, and re-painting and the T-bar ceiling requires minor repairs. One sink, the toilets and shower and sink fixtures require replacing. The dirty clothes hanging baskets are missing and need to be re-installed.

The mechanical room consists of the heating and ventilation systems for the offices and shop/warehouse areas, water heaters for the dryer's, and electrical panels for lighting and plugs for the three floors of the office complex area. A new telephone exchange centre will be required. The two heating units will need to be serviced and checked. The hot water tanks have had the wiring cut to the tanks. The wires are too short to reconnect but can be moved back to one tank necessitating only new wiring for the fourth tank in line. All plumbing has been removed for the tank inlets, outlets, and pressure relief systems. The electrical panels appear to be in good shape as do the transformers. Wiring has been cut and removed, including ground wires. The panels still have all the office wiring connected but this wiring has been cut-off in the office area. This panel will need to be completely re-wired.

18.7.3 Top Floor Office Area

The top floor is accessed by two stairwells at either end of the main offices area. The dry wall along the stairwells requires repair, cleaning, and repainting. The stairways are in excellent and serviceable condition. The fire doors need the glass replaced or new doors installed. All windows in the outside walls require glass to be installed. Heat reflective glass will be installed in the outside windows.

The main offices area has had most walls removed creating one large open space. On the stairwells side (adjacent to the end wall of the shop and warehouse section) of the main offices are partial walls creating separate office space. One room has dry wall and T-bar ceiling intact while the other two only have the steel studs for walls in place. These spaces will be refurbished, as one computer server room, one office space, and a conference room. On the other side of the stairwell, near these office areas, was located the old men's and women's washrooms and a small kitchen. The plumbing is intact but everything else, including the walls, has been completely removed. This area will be reconstructed, as before and equipped.

The remaining open area of the office space, during mine construction and the initial operating period of the mine, will be converted into simple workstation spaces with desks, chairs, shelves, and storage cabinets. The ceiling will be left open to the roof and electrical wiring, light fixtures, and ventilation system piping will be exposed with electrical and ventilation drops to workstations and open areas, as required.

During the first year of mine operations, a row of offices will be constructed along the one long wall and one short wall (by the bathrooms end of the space) containing windows. Most offices will be approximately 3.5 m wide by 4 m long (from the outside walls). The general manager's office will be 5.1 m long by 4.5 m wide. All office walls will be pre-finished dry wall. Each office will have a locking door and window allowing light to pass through to the main workstation equipped 'bullpen' area. Internal windows will generally be 1.8 m wide by 1.2 m high. The secretary's office will require a smaller window, as the office is narrower. All offices will have a ceiling constructed from sound proof ceiling tiles. The large corner office by the washrooms will be divided in half and a door installed in the middle for access. The corner storage area will be cleaned out and repainted for file storage. This large corner office will be for health and safety and training. The old file storage room, near the new conference room, will be converted to a photo copy/printing room. The concrete flooring in the main office area will be refinished eventually with tiling.

18.8 PROCESSING PLANT CONTROL ROOM AND OFFICES

18.8.1 CONTROL ROOM

The central control room for the processing plant has been extensively vandalized and requires a new T-bar ceiling, unbreakable glass in the 10 window panels overlooking the plant, and glass in the 6 outside looking panels, to be installed. The entrance door will also be replaced. The dry wall requires repair, cleaning, and painting and electrical wiring and lighting fixtures repaired and/or replaced. New, non-slip flooring will be provided.

18.8.2 OFFICE

The mill offices have four (4) outside window panels and four (4) inside window panels overlooking the plant, which requires glass replacement, as in the control room. One panel is missing the middle brace and may require a double wide window panel, if replacement parts cannot be found. The dry wall requires repair, cleaning, and painting and electrical wiring and lighting fixtures repaired and/or replaced. New, non-slip flooring will be provided.

18.8.3 Men's Washroom

All dry wall, including the ceiling, needs to be replaced/repaired, cleaned, and painted. Electrical wiring needs to be checked and lighting fixtures will be replaced. New, non-slip flooring will be provided. Plumbing appears to be serviceable with the sinks and toilets requiring cleaning or in some cases replacement.

18.8.4 Ladies' Washroom

All dry wall, including the ceiling, needs to be replaced/repaired, cleaned, and painted. Electrical wiring needs to be checked but lighting fixtures appear functional. New, non-slip flooring will be provided. Plumbing appears to be serviceable with the sinks and toilets requiring cleaning or in some cases replacement. The janitor's closet at the back of the ladies' washroom requires similar work with the wash tub and plumbing appearing serviceable.

18.9 ASSAY LABORATORY

The assay laboratory building requires some refurbishment. The dry wall will be repaired, cleaned, and repainted. One outside window pane requires replacement. The T-bar ceiling will be repaired and/or ceiling panels will be replaced, as required. The heating and ventilation systems, as well as the fume hooding systems, will be checked, repaired, as required, and recommissioned. A new hot water tank will be installed in the mechanical room as will a ceiling heater in the wet lab. All desks, benches, and cabinets will be refurbished and supplemented, as required.

The assay lab will be equipped (at Minnova's expense) and operated by a recognized and certified assay laboratory operator on contract, such as SGS or Act Lab.

18.10 CAMP ACCOMMODATION

18.10.1 Construction Camp

The present exploration camp can house and feed approximately 15 people. Accommodation trailers consist of rooms, which are approximately 4 m by 3.5 m with 2 beds per room, 2 closets, 2 chairs, and a work table, but as much as possible, until the extra accommodation space at the permanent camp site is provided, 2 people per room (on opposite shifts), may be required in some rooms. There is a central washroom facility consisting of 4 sinks, 2 showers, and 2 toilets. All units are forced air propane heated and air conditioned.

Minnova owns and has stored in Flin Flon enough accommodation trailers to increase the camp size to approximately 30 people. The configuration of the accommodation trailer block is similar to the one presently on-site. This would provide accommodation and meals for employees for the first 6 months of the mine pre-production period, when only underground ramp development and limited processing plant and surface infrastructure work is being performed.

The additional 6 trailers would be located adjacent to the existing block of trailers and connected via a corridor to the existing trailer block. The new rooms' occupants would use the existing catering and common room facilities. The area where the new trailers would be located requires minimal site preparation, as it has already been, cleared, levelled, and crushed rock placed to grade. The other services (electricity, water, and propane), except for sewage disposal for the present camp, can be extended into the new trailers block. With the enlarged camp and significant increase in occupancy, the construction camp will be connected to a new sewage treatment facility in the plant area. This will require the sewage holding tanks to be connected by a 200 m heat traced pipe line to the sewage treatment facility. The pipe will be a 102 mm HDPE pipe and be laid in a trench covered in crushed rock to allow vehicles and transport trucks to cross over it at specific points in the road to the warehouse.

The existing kitchen and eating areas will be sufficient to cater to the number of people located in the construction camp given that approximately one-half the people will be split between two shifts with staggered meal times.

18.10.2 PERMANENT CAMP

It is planned that during the first year of mine operations, as the viability and longevity of the PL Mine is established, that a new purpose built accommodation, catering, and recreational facilities complex will be constructed near to Jay Lake, approximately 1.5 km from the processing plant site. The total capacity of the permanent camp will be 100 people.

A portion of the permanent camp accommodation at the permanent camp site will be installed for approximately the last 6 months of the pre-production period when more than 30 workers will require accommodation and meals while performing their construction duties. This portion of the camp will consist of 40 rooms and a kitchen/dining facility and common rooms equipped with a television. All areas will be forced air propane heated and air conditioned. As the Project proceeds to production, construction of the entire 100 man camp will be completed.

The accommodation for the hourly employees would be 2 rooms sharing a common washroom. The rooms would be 4 m by 3.5 m with a window in every room. The room would be equipped with 2 single beds, 2 closets, 2 chairs, and a work table. The common washrooms would consist of a sink, toilet, and shower. The accommodations for the management staff would be of similar size and set-up, but with individual washrooms.

The new permanent camp kitchen facilities will be installed in time to enter service approximately 6 months after the start of the pre-production period and feed all personnel located in the construction and permanent camp accommodation. The construction camp catering facilities will be closed at that time. The camp kitchen and catering facilities will feed all personnel, providing three meals per day per person. All kitchen facilities will be operated by a catering contractor. The facilities will be connected to the indoor recreational facilities areas. The food storage and kitchens are designed to store and handle food in bulk. All food is prepared and cooked on-site. The kitchen facilities will include separate areas for frozen and refrigerated food storage, vegetable storage, non-perishable food storage, food preparation, cooking, serving, dishwashing, and storage, meeting regulation hygiene standards. Storage areas will be separated from the main cooking areas and all cooking facilities will be constructed from standard commercial kitchen materials, which are easily cleanable and ensure hygienic conditions at all times. The dining facilities will be 14 m by 14 m and be capable of seating 70 personnel at a time.

Other facilities will include two offices for camp staff, separate washroom facilities for kitchen staff, and a small commissary/shop for employees to buy personal items.

In Year 1 of production, the remainder of the permanent accommodation and recreational facilities will be completed and commissioned. At this time, the exploration camp will be decommissioned and sold.

18.11 FUEL AND LUBRICATION STORAGE AND DISTRIBUTION

The fuel and lubrication storage and distribution facility will consist of two diesel fuel storage tanks. Each tank will have a capacity of 18,000 litres. This fuel storage facility will be located in proximity to the mine portal, as the major consumer of diesel will be the underground mining equipment.

There will be one 1,000-litre tank that will hold used lubricating oil and will be situated near the maintenance shop facility. Fresh lubricants will be stored in drums.

The fuel and lubricants storage areas will be provided with a concrete floor and walls high enough to ensure that the total volume of the tanks inside the containment area is contained in the situation of a leak.

18.12 WATER SUPPLY AND TREATMENT

The PL Mine has received in September 2017 a new permit to draw up to 950 cubic decimeters of water for industrial purposes under the Water Rights Act. In addition, process water will be recycled from the Ragged TMF.

The process water supply will utilize water pumped out of the underground mine as part of the mine dewatering programme. Clean service water for uses, such as gland water for pumps, firefighting, cooling circuits, etc., will be supplied directly from sources of industrial water drawn from Jay Lake.

The main losses of water will occur from water, which is not recycled, dust suppression on site roads, and evaporation.

Water for the processing plant, offices/dry complex, and the camp will be drawn from Jay Lake and all non-processing water will be treated.

The water treatment plants for both the camp and the mill/office complex will be containerized and will include filtration for suspended solids, reduction of organic colour, disinfection of the water, and distribution of the treated water to the site. Both systems will use the same treatment process and equipment to filter and disinfect the water, and both systems will have some over capacity to support future expansion, if necessary.

The camp system will need to be installed outside, so it will be supplied in a fully insulated 40 foot sea container with 2 - 5.6 m^3 (1,500 gallon) tanks inside. The mine site plant will be installed within the warehouse, so it does not require winterizing. The mine site plant will be installed in a 20 foot sea container and will have a 22 m^3 (6,000 gallon) poly tank installed beside it (102 inch diameter, 182 inches tall). Both plants will have Ontario ESA approval. The camp system assumes electric heating.

The treatment process is filtration using a high performance granular media backwashing filter system with Duplex Organic Colour reduction using ion-exchange resin. Primary disinfection is a duplex Upstream UV system while secondary disinfection consists of chlorine injection into the 22 m^3 potable water storage tank at the mine site and the 11 m^3 potable water storage at the camp.

The systems include an automatic filtration system, self-cleaning duplex UV primary disinfection systems, reliable chlorine feed pumps, water meters for keeping track of production, VFD control of distribution pumps, full duplexing of critical components, and an alarm for system faults. Each system is designed to filter +80 m^3 of water per day (20 gpm maximum production rate) and deliver water at a maximum flow of 22 m^3/hr (90 gpm). A 600 VAC 20 A main service supply system includes 10 kVa step-down transformer for low voltage circuits.

18.13 SEWAGE TREATMENT

The old sewage treatment plant and septic field near the processing plant building have been deemed unsuitable after 30 years of sitting idle and will be replaced with a modern state-of-the-art Membrane BioReactor (MBR) system, employing the most advanced membrane technology, and providing a cost-effective and user friendly wastewater treatment facility.

The MBR system is a bioreactor that combines the activated sludge process with advanced membrane technology. The process utilizes naturally occurring micro-organisms in an environment where they can biodegrade the organic material present in the wastewater into carbon dioxide and water.

Steps in the process include:

1. Pre-treatment and Equalization
2. Bio-reduction
3. Membrane filtration with
4. Ultraviolet (UV) Disinfection optional

MBR systems were originally developed over 40 years ago and have been in widespread use for over 20 years. They were developed to overcome the physical constraints of gravity clarification and media filtration through the use of a synthetic membrane to act as a fine-particle sieve to capture and reject suspended and colloidal solids in excess of 0.15 *u*m in size. This barrier prevents these solids from exiting the aeration portion of the activated sludge process and it allows the Mixed Liquor Suspended Solids (MLSS) concentration to be increased to the 8-15,000 mg/L range. At this MLSS concentration, the mass of micro-organisms available to do the work is dramatically increased and the MBR can accomplish significantly more work in a given aeration tank volume.

Wastewater generated will be pumped through a 1 mm perforated screen/side hill screen and into an Equalization tank. This Equalization will reduce the peak flow rates transferred to the BioReactor.

The BioReactor is a complete mix reactor with a specific volume dedicated for biological reduction in the secondary aeration step of the activated sludge process. The biological reduction volume is dictated by the Food to Micro-organism (F/M) Ratio and a mass loading of BOD per cubic volume of reactor. The mixed liquor is recirculated within the MBR at a rate of approximately 4-5 times the influent flow.

The activated sludge process converts the soluble organic material present in the wastewater into CO_2, H_2O, and biological cell mass.

Positive displacement or regenerative blower/motor units will provide the oxygen required for this process. The mass of oxygen transferred is based upon the peak daily influent BOD load. These units will also provide air scour for the membranes. Approximately 50% of the oxygen transferred to the system for membrane air scouring is credited to the overall oxygen demand of the system.

The liquid phase of the mixed liquor is pulled through the membrane at a predetermined rate, or flux, established for each specific application. The mixed liquor suspended solids are rejected and moved away from the membrane by the air scour and hydraulic action. Permeate will be pulled through the membranes under suction by permeate pumps and transferred under pressure to any downstream processes or for discharge.

Surplus biomass generated by the conversion of BOD into the cell mass will be periodically wasted from the system as Waste Activated Sludge ("WAS"). The system will convert BOD into WAS at a ratio 2.5:1.0. The sludge will be pumped directly from the reactor on an as needed basis for further processing.

Separate treatment plants will be required to treat sewage from the offices/dry complex and washrooms complex as well as the expanded construction camp, located close to the processing plant building and for the permanent camp facilities. The office and construction camp will generate 30-40 m^3 of sewage and dirty water per day requiring treatment.

In addition, in Year 1 of operations, once the new permanent camp is completed near Jay Lake, a new sewage treatment plant will be included as part of the camp facilities. This treatment plant will also be required to process approximately 30-40 m^3 of sewage and dirty water per day. Both permanent sewage treatment facilities will be housed in 40 foot sea containers. The mine portal area building will utilize a small portable toilet unit with the wastewater sent to the camp sewage treatment plant.

18.14 GARBAGE

Garbage created on the site will be put in a compactor bin and transported to the nearest municipal landfill site for disposal by a local contractor.

18.15 STANDBY DIESEL POWER PLANT

The back-up diesel generator appeared to be in good condition and has been serviced and recommissioned. It can provide 250 kV of standby power and is presently powering the exploration camp.

18.16 COMMUNICATIONS

The site presently is serviced by a satellite telephone and internet communication system, which is provided on a monthly rental basis. There is no major utility fibre or telephone connection point near the property and any new lines would have to connect to main services at Sherridon.

To provide voice, data, and internet services to the operation and camp, a satellite communications system will continue to be employed. The head end of the system will employ an Enterprise Satellite high use R-C-HU-10x1.5 Plan. Enterprise Satellite Internet provides speeds of up to 10 mb/s download and up to 1.5 mb/s upload with lower latency, no beam saturation, and unlimited data per month. The system will include traffic shaping protocols to ensure fair usage of data and bandwidth including, but not limited to, user speed limitations and the blocking of streaming video, streaming audio, movie downloads, and gaming. The system solution will be on a rental basis and include all hardware and licensing included (racks, modems, router, access points, switches, etc.) and 24/7 cloud monitoring. VoIP phone lines include unlimited calling within Canada.

The service would consist of voice, e-mail, internet, and data services for the mine operations departments. This would be distributed throughout the property using fibre, copper cable, and wireless networking equipment inside buildings and between buildings where costs effective and using wireless networking equipment to connect more remote locations to the main offices complex.

In the camp, the service will provide television service and wireless Internet to each room. The system will allow the camp to communicate with video-conferencing methods, send/receive e-mails, and access the general internet.

18.17 GENERAL AND ADMINISTRATION (G&A)

Administration comprises senior and general management, accounting, environmental, and information technology functions. As well as direct salaries and fringe benefits for personnel, other components include employee re-location, travel allowances for business away from the property, insurance (property, business interruption, and political risk), taxes, permits, and licences, mining rights fees, professional fees, and operating surface vehicles for the personnel.

Accounting functions would include providing payroll, accounts payable and receivable processing, expenditures budgeting and forecasting, and other corporate cost accounting for the entire operation and reporting to head office.

Information technology comprises all components associated with operating and maintaining the telephone, computer network, and internet systems for the mine site. Allowances for local and overseas telephone services and the internet charges are also include.

The environmental components' costs would be associated with monitoring of the mine's environmental performance and reclamation work.

18.17.1 PROCUREMENT

Procurement encompasses all functions associated with on- and off-site procurement of materials and supplies, warehousing and inventorying, transportation from point of origin to the site, and other associated support services. Actual freight costs for items required by the mine, processing plant, and maintenance departments are included in those department's costs. The main cost components comprise wages and warehouse supplies. Salaries and fringe benefits for staff and warehouse supplies, such as personal protective gear (gloves, etc.) and small equipment (pallet lifters, forklifts, etc.) parts for operating the warehousing, purchasing, and logistics groups, are included. Surface support includes unloading and loading of trailers and shipping containers, movement of materials on-site, and maintenance of the warehousing and associated facilities.

18.17.2 HUMAN RESOURCES

Human resources encompass all functions associated with personnel and industrial relations, health and safety, training, and community relations. Personnel and industrial relations costs comprise salaries and fringe benefits for the staff to undertake recruitment of required personnel, managing of company salary policies and fringe benefits, managing and negotiating of collective agreements with hourly employees, and overseeing of company policies and procedures. Health and safety provides for salaries, fringe benefits, and supplies for the services of on-site first aid personnel and all first aid supplies and vehicles required by them.

Community relations costs will provide funds to aid in supporting local community efforts and facilities.

18.17.3 SECURITY

Security would be provided on a contract basis by a security firm and include all personnel and supplies. Security surveillance equipment would be provided by the mine to the security firm. Small security equipment for the security personnel (metal detectors, etc.) would be provided by the contractor.

18.17.4 AIR TRAVEL TO SITE

The employees living in camp will be rotated in and out using charter air services. Employees will be expected to arrive in Winnipeg, where the charter flights originate and will be flown to Flin Flon. From the airport, everyone will be bussed to the mine. Charter flights will be flown once per week.

18.17.5 MANPOWER

The general and administration manpower requirements are presented in Table 18.1. It is estimated that there would be 15 staff and 5 surface department personnel.

Table 18.1: Manpower Requirements

Position	Total Complement
Maintenance Clerk/Planner	1
Surface Mechanic	2
Labourer	2
Total Surface Dept.	**5**

Position	Complement
General Manager	1
Comptroller	1
Accountant	2
Head of Health/Safety and Security	1
Environmental Technician	2
Office Clerk/Secretary	1
Purchasing Agent	1
Warehouseman	4
Medical Services (Contract)	1
Security Contract	1
TOTAL COMPLEMENT	**15**

The operation will be managed by a senior management team led by the mine manager. The other senior management will be the:

- Mine Superintendent;
- Processing Plant Superintendent;
- Maintenance Superintendent;
- Chief Engineer;
- Senior Geologist;
- Head of Health/Safety and Security; and
- Comptroller.

All supervision, employees, and contractors report to or through these positions.

The complete workforce would be comprised of people housed locally in the Sherridon area or on rotation from other parts of Canada.

19 MARKET STUDIES AND CONTRACTS

All gold produced will be sold to the Royal Canadian Mint or other refiner (such as Johnson Mathey) as is the common practice in Canada. The gold value will be paid upon receipt of refiner and referee assay results for doré bars. The price paid is based on LME's daily set gold prices for spot market sales.

Long-term gold pricing assumptions for this Feasibility Study used the average of the 3-year and 5-year moving average gold price, to the end of September 2017. The graph in Figure 19.1 presents the monthly average gold price for the last 3 years and the equivalent 3-year moving average gold price. The 3-year and 5-year moving average gold prices are US$1,216 and US$1,291 per ounce, with an overall average gold price of US$1,253 per ounce.

The cash flow modelling incorporated a long-term gold price of US$1,250 per ounce.

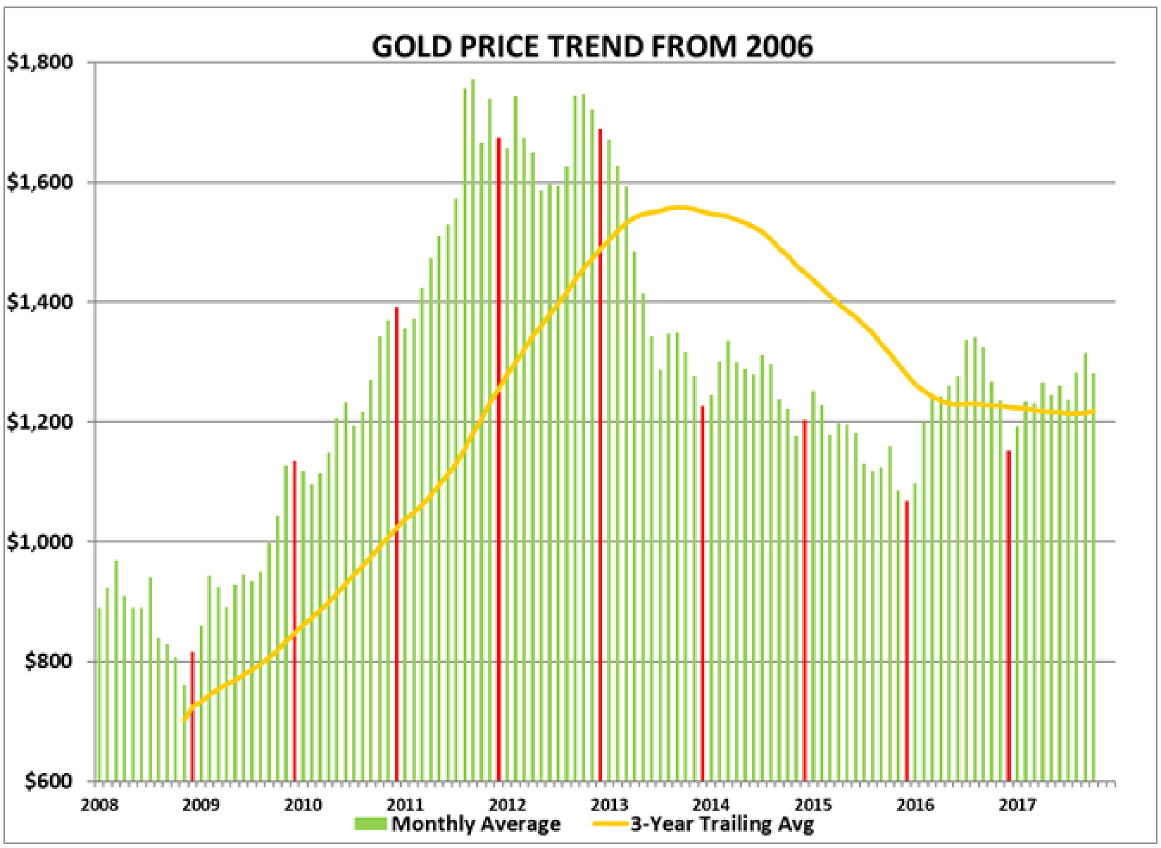


Figure 19.1 Monthly average gold price for the last 3 years and equivalent 3-year moving average gold price

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Puffy Lake Gold Project ("PLGP") is a past operating mine site that was developed in 1987 and operated between January 1988 and March 1989 under Environment Act Licence No. 1207E. It was developed by Pioneer Metals Corporation ("Pioneer") as an underground mine. Approval of the Puffy Lake ("PL") Mine operating license was supported by baseline terrestrial and aquatic investigations carried out in May 1987, which formed the basis for the 1987 Environmental Impact Statement ("EIS") prepared by Ilam Associates Ltd. and filed at that time with Manitoba Environment (as it was then known). The Environment Act License remains in full force and effect relating to proposed re-start of underground mining activities.

Minnova acquired the property from Barrick Gold in 2011 and commenced a review of all mineral resources following, which it determined that the re-start of operations was warranted. Further, Minnova determined that the near surface ore included in the PL Deposit mineral resource would be accessed more economically and efficiently from the surface than from the existing underground workings. Minnova, therefore, initiated base line environment studies that included environmental data collection and analysis related to the future development of shallow open pits and related infrastructure, which would require an alteration to the existing Environment Act License 1207E.

The PLGP is currently subject to the terms and conditions of licence 1207E and the Metals Mining Effluent Regulations ("MMER"). The MMER allow mine effluent to be released to the receiving environment, if the effluent pH is within the range of pH 6 to 9.5, if the concentrations of the MMER deleterious substances in the effluent do not exceed authorized limits, and if the effluent is demonstrated to be non-acutely lethal to rainbow trout. The authorized limits of deleterious substances are shown in Table 20.1. The MMER also include comprehensive Environmental Effects Monitoring ("EEM") requirements to evaluate the potential effects of effluents on fish, fish habitat and the use of fisheries resources and encompass as examples effluent characterization, biological monitoring, subsequent biological monitoring, and final biological monitoring prior to mine closure.

Table 20.1: MMER Authorised Limits of Deleterious Substances[1]			
Deleterious Substance	Maximum Authorized Monthly Mean Concentration	Maximum Authorized Concentration in a Composite Sample	Maximum Authorized Concentration in a Grab Sample
Arsenic	0.50 mg/L	0.75 mg/L	1.00 mg/L
Copper	0.30 mg/L	0.45 mg/L	0.60 mg/L
Cyanide	1.00 mg/L	1.50 mg/L	2.00 mg/L
Lead	0.20 mg/L	0.30 mg/L	0.40 mg/L
Nickel	0.50 mg/L	0.75 mg/L	1.00 mg/L
Zinc	0.50 mg/L	0.75 mg/L	1.00 mg/L
Total Suspended Solids	15.00 mg/L	22.50 mg/L	30.00 mg/L
Radium 226	0.37 Bq/L	0.74 Bq/L	1.11 Bq/L
[1]Concentrations are total values. Source: MMER Schedule 4.			

Future pit development will require the filing of a Notice of Alteration ("NOA") with the Manitoba Conservation and Water Stewardship, Environmental Approvals Branch describing proposed alterations to the licenced PL Underground Mine including the development of open pits on the PL gold property. The NOA will seek approval for the inclusion of open pit mining as an additional mining method for the already licenced PL Mine (Environment Act License 1207E). The Proposed Alteration would be restricted to the near surface, up-dip portion of the existing development and open pits would be developed sequentially and progressively rehabilitated as mining progresses. Mining and concurrent rehabilitation of the open pits would occur over a 3 to 4 year period, commencing after all required permits are in place (Table 20.2), which is anticipated in Year 2 of operation.

Table 20.2: Current List of Permits in Place			
	Permit/License	Expiry Date	Description
Environment Act License	1207E	N/A	Permit to Operate as Underground Mine
Mining Lease	65	01-Apr-34	
Crown Land Permit	GP0002799	31-Dec-17	All Weather Road
Crown Land Permit	GP0003758	31-Dec-17	Commercial Lot and Access Roads
Crown Land Permit	GP0004038	31-Dec-17	All Weather Road
Crown Land Permit	GP0004134	31-Dec-17	Commercial Lot and Access Roads
Water Rights License	2017-116	Issued September 2017	License to Use Water for Industrial Purposes
Casual Quarry Permit	CP-2012-1001080	31-Dec-17	Local Quarry for Road and Site Maintenance

Re-activation studies completed by Minnova include:

- AECOM PL Mine: Environmental Baseline Assessment, May 2014
- WESA: Puffy Lake Preliminary Assessment of Pit Inflows and Water Quality, December 2011
- Parks Environmental Inc.: Acid Rock Drainage and Metal Leaching at the Puffy Lake Gold Project, May 2012

- North/South Consultants Inc.: Status of Ragged TMA, December 2011
- AECOM: Fish Population study, August 2014

In mid-2014, the Company sought out and received clarification on the status of the designated Ragged Tailings Management Area ("Ragged TMA"). Since the Ragged TMA has not been used for tailings deposition since the late 1980s, Fisheries and Oceans Canada ("DFO") requested that Minnova undertake a fish population study to determine if the Ragged TMA is a water body "frequented by fish," as this will be a key factor in determining whether it should be added to the Schedule 2 of the Metal Mining Effluent Regulations ("MMER"). The only historical fish population information dates back to a 1987 Project Description and Environmental Impact Assessment conducted by Ilam and Associates (Ilam, 1987). Therefore, DFO has no basis for determining whether or not the water is currently frequented by fish.

Ragged Lake is a small headwater lake that was used for disposal of approximately 350,000 tonnes of mine tailings during the first operating period of the mine in 1987-1988. Ragged Lake was authorised for use as a tailings management facility by letter dated January 30, 1989 by the Honorable Mr. Tom Siddon, under Section 33.1 of the Fisheries Act upon confirmation by regional officials that: (a) public hearing into the proposal were undertaken and "there were no adverse public reaction to the use of Ragged Lake as a tailings basin" and (b) that "My regional officials are satisfied with your assessment that no significant fishery will be lost as a result of the designations [as a tailings facility]".

Following a fish population study completed in August 2014, Environment Canada ("EC"), in collaboration with Fisheries and Oceans Canada ("DFO"), determined on September 30, 2014 that because fish now occur within the Ragged TMA, they consider the Ragged TMA to be a body of water frequented by fish. As such, EC stated the "Ragged TMA will require a listing on Schedule 2 of the Metal Mining Effluent Regulations (MMER)." In order to achieve a Schedule 2 listing, the Company will be required to submit an Assessment of Alternatives report to determine if the Ragged TMA is in fact the best option for deposition of new tailings. The assessment will consider all possible alternatives for safe, long term tailings storage from environmental, socio-economic, and technical perspectives.

20.2 AIR QUALITY

There are no large urban centres and industrial sources near the PLGP. Background air quality and sound levels are likely typical of rural, relatively unpopulated forested regions; however, an air quality assessment would need to be completed to confirm.

Impacts of the Project on the air quality in the area are expected to be minimal. Dust will be generated during dry summer weather conditions on access roads to the site and on site roads. Water will be sprayed on the roads, if the dust becomes problematic.

Dust emissions from the tailings area are expected to be negligible. The major portion of the tailings will be submerged below water in the Ragged TMA. Because the mill circuit will employ a relatively coarse grind, the tailings will tend to settle and compact into a dense mass; thus, minimising the potential for dust generation from those portions of the tailings above water.

The mill itself uses a wet process during the crushing, grinding, and treatment of the ore and in-plant dust generation is avoided. Other emissions from the plant are expected to be minor. Once a week, the precious metal precipitate from the Merrill-Crowe precipitation unit will be mixed with flux and fed into the reverberatory furnace where it will be refined to bullion.

20.3 NOISE CONDITIONS

The PLGP site is approximately 12 km from the nearest human habitation at Sherridon and is naturally screened by the surrounding hilly terrain and forest. The mill will generate noise mostly from the crushing and grinding circuits and the noise levels will be typical of milling operations. Noise will also be created during open pit blasting. These will be short-duration events. Noise will also be created by haul trucks and from ore and waste rock management activities.

20.4 WATER QUALITY

All liquid effluent from the mine and mill will be discharged, along with those mill tailings not required for backfill operations, to the Ragged TMA.

Hydrogen peroxide treatment of the barren bleed from the cyanide leaching process will destroy most of the cyanide so that total cyanide levels in the discharge to the Ragged TMA will not exceed 0.5 mg/L. Retention time of the liquid effluent in the tailings basin will vary, depending on the extent to which the basin has been filled, but decreasing progressively over the life of the mine. Nevertheless, retention in the tailings basin will allow further natural degradation of the cyanide to take place. Contact water from the waste rock area will be sent to the Ragged TMA and water in the Tagged TMA will be treated in an effluent treatment plant before discharge to the environment. The ETP will be designed to remove metals and ammonia (if required) so that effluent from the plant will meet MMER's effluent limits.

20.5 CURRENT LAND USE

The PLGP area is a past operating mine site. Prior to its development in 1987 and its operation between 1988 and March 1989, there was no pre-existing access to the area and no infrastructure development. The site is currently accessed by a 9 km long, all weather, gravel road. Paralleling the road is a 24 kVa power line that is connected to the Manitoba Hydro substation located in Sherridon. The licenced operation included a decline and underground mine workings; a 1,000 tpd capacity on-site processing plant; the Ragged TMA; supporting infrastructure including, mine office, change house, lab assay office, emergency diesel generator, parking lot, paste backfill facility, concrete transformer pad, explosives storage area, fuel tank storage area, telecommunications system; power distribution and the access road and security gate. The Ragged TMA is separated from downstream lakes by a 1.5 m high outlet control structure (weir) and contains approximately 350,000 tonnes of mill tailings deposited during previous operations.

20.6 CLIMATE AND TOPOGRAPHY

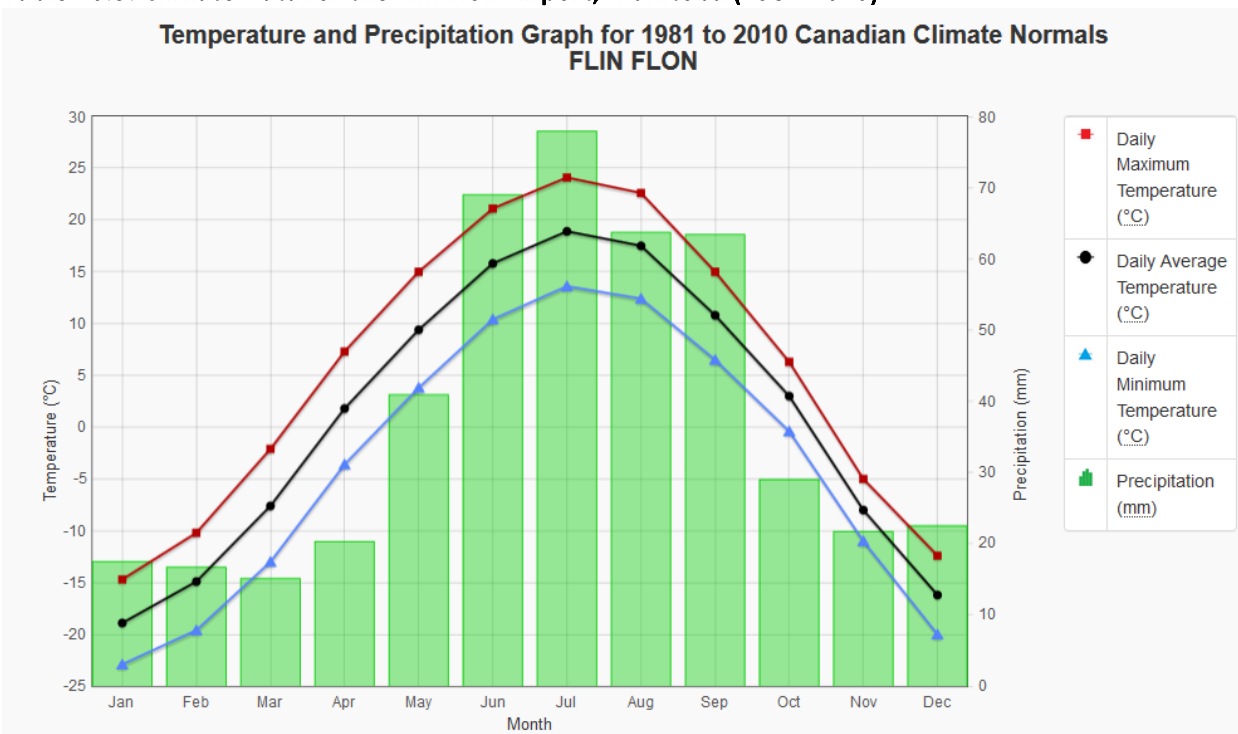
The region lies within a warmer, more humid subdivision of the High Boreal Ecoclimatic Region in Manitoba. In the Granville Lake Ecodistrict, summers are cool and short with an average growing season of 151 days. Winters are characterized as long and cold.

Although the closest community to the Project is Sherridon, the closest weather station to the site is located near Baker's Narrows at the Flin Flon airport, approximately 60 km southwest of the Puffy property. The Flin Flon airport is located at an elevation of 304 masl and is considered climatically representative of the region.

The mean annual air temperature at the Flin Flon airport is -0.2°C. The daily mean temperature ranges from approximately 18°C in July to -21°C in January. Total mean annual precipitation at the Flin Flon airport includes 336 mm of rain and 141 cm of snow. July has the highest average rainfall (approximately 77 mm), whereas November and December have the highest average snowfall (approximately 25 cm and 24 cm, respectively) (Environment Canada 2012a).

The daily maximum-minimum and average temperatures and mean monthly precipitation conditions measured at the Flin Flon airport between 1981 and 2010 are provided in Table 20.3.

Table 20.3: Climate Data for the Flin Flon Airport, Manitoba (1981-2010)



Note: Latitude 54° 41' N Longitude 101° 41' W Elevation 303.90 masl.
Data obtained from Environment Canada Flin Flon A meteorological station (2012a)

20.7 TOPOGRAPHY AND DRAINAGE

The area surrounding the PLGP site consists of rolling hummocky terrain typical of this portion of the Churchill province of the Canadian Shield. Bedrock and gravel ridges are interspersed with low swampy areas and numerous small lakes. Generally, the bedrock ridges tend to be linear and follow the structure of the underlying bedrock, particularly in the more resistant quartzo-feldspathic and granitic gneisses. The area has moderate relief with elevations ranging from approximately 330 m at Puffy Lake to around 365 m in the area north of the site. Puffy and Ragged Lakes lie very close to the divide between the Churchill and Burntwood River watersheds. Ragged Lake drains south into Puffy Lake and then east into a small unnamed lake, which in turn drains north into Paddy Lake and Nokomis Lake. From there, the water flows in a generally northeasterly direction through the Limestone Creek and File River system to join the Burntwood River at Burntwood Lake. “Northeast” Lake, another small unnamed lake 1 km northeast of the mine site drains north and east into Hutchinson Lake. Jay Lake, which lies 2 km due west of “Northeast” Lake is on the other side of the divide. From Jay Lake, the water flows north to eventually join Kississing Lake and then northeast via the Kississing River to join the Churchill River at Flatrock Lake.

20.8 GROUND WATER

Field activities to determine preliminary hydrogeological conditions were completed in 2011 and 2012 and included the installation, hydraulic testing, and sampling of ten (10) groundwater monitoring wells in bedrock exploration boreholes located throughout the PLGP property (WESA, 2012). Results from this groundwater characterisation study are summarised below, including estimated pit inflow quantities and quality at five proposed pits.

Static water levels and hydraulic conductivities were estimated as part of the WESA Study. Static groundwater levels were generally shallow, encountered at 0.01 to 3.31 m below ground surface (mbgs). Hydraulic conductivities at the ten (10) groundwater monitoring wells ranged from 1×10^{-8} m/s to 2×10^{-4} m/s (Table 2.2), with a geometric mean bulk hydraulic conductivity of 5×10^{-7} m/s. The accuracy of the inflow estimates is dictated by the appropriateness of the bulk hydraulic conductivity value used to perform the calculations. Single well hydraulic tests provide an initial estimate of the hydraulic properties of the subsurface in the immediate vicinity of the wells. Large-scale aquifer pumping tests lasting 72 hours or more and using a pumping well and several monitoring wells should be conducted to provide a representative estimate of the hydraulic characteristics within the radius of influence of the pumping well. Pumping tests will improve the representativeness of the pit dewatering and radius of influence estimates, and associated water management and potential treatment prior to discharge into the environment of the extracted water.

Potential pit inflow estimates from dewatering were derived using the steady state analytical solution presented by Marinellia and Niccoli (1998) for each of the five pit areas (WESA, 2012). This method accounts for groundwater inflow from the pit walls as well as from the pit bottom. The underlying assumptions for this method are reasonable considering the moderate to high permeability of the rock mass. Considering that the pits will be operated sequentially, estimated groundwater inflow into the pits was estimated for each pit area independently (*i.e.*, no cumulative impacts from operating multiple pits concurrently is anticipated). The radius of influence predicted to result from pit dewatering was also estimated.

Using the best available estimates from single well hydraulic tests, the geometric mean hydraulic conductivity was used to calculate predicted inflow rates and radius of influence. Pit inflow rates estimated for the individual pits ranged from 1.3 L/s in Pit #5 to 4.4 L/s in Pits #1 and #4, resulting in a predicted radius of influence ranging from 200-300 m around each of the pits. To provide a sensitivity analysis of the hydraulic conductivity on inflow rates, the pit inflow estimates for Puffy Mine open pits were made using the bulk (geometric mean) and the higher and lower hydraulic conductivity values obtained from the single-well tests.

For the underground workings (access ramps, drifts, etc.), preliminary predicted dewatering rates were estimated using Darcy's Law.

Groundwater samples were collected in 2011 (three wells) and 2012 (seven wells) and submitted to an accredited laboratory for analysis of general chemistry and dissolved metals (WESA, 2012) in order to characterise background groundwater quality. Analytical results from the groundwater samples were compared against the limits from the Metal Mining Effluent Regulations ("MMER"), the Canadian Water Quality Guidelines ("CWQG") for Aquatic Life, and to the Manitoba Water Quality Standards, Objectives, and Guidelines ("MWQSOG"). None of the concentrations exceeded the MMER. Concentrations of arsenic, copper, iron, molybdenum, selenium, silver, and uranium at some monitoring locations exceeded the Federal Canadian Water Quality Guidelines for the protection of aquatic life ("CWQG-AL"). Similarly, the same parameters, except mercury, selenium, and uranium, were above the Provincial Manitoba Water Quality Standards, Objectives and Guidelines (WQSOG Tier II Limit).

To better assess future pit dewatering requirements with more certainty and assess the potential impacts dewatering may have on local surface-water bodies, the following is recommended:

1. Major bedrock fractures systems and faults should be identified, and large-scale aquifer pumping tests should be conducted to determine accurate aquifer hydraulic conditions, including horizontal and vertical hydraulic conductivities.
2. Based on the conceptual groundwater model developed through the characterisation of bedrock fracture systems and the large-scale aquifer hydraulic conditions (pumping tests), the appropriateness of analytical solutions or numerical modeling to pit inflow estimation should be evaluated. The pit inflow estimates should be re-evaluated using the more accurate aquifer hydraulic conditions with consideration of boundary effects and cumulative drawdown effects.
3. Additional characterisation and predictions of dewatering effluent quality are recommended.

Manitoba Conservation and Water Stewardship well records indicate no groundwater utilization near the site, with no registered groundwater wells in use within at least 9.6 km of the site (Manitoba Water Stewardship, 2009).

20.9 SOIL

Associated with the acidic granitoid bedrock in the Granville Lake Ecodistrict are "well to excessively drained, shallow, sandy, and stony veneers of water-worked glacial till on which Dystric Brunisols have developed. On clayey deposits, Eutric Brunisols and Gray Luvisols prevail. Peat-filled depressions are common and "form complexes of very poorly drained, Typic (deep) and Terric (shallow) Fibrisolic and Mesisolic Organic soils overlying loamy to clayey glaciolacustrine sediments. Widespread patches of Organic Cryolsols occur where permafrost is present in peatlands (peat plateau and veneer bogs) and in some clayey mineral soils" within the Granville Lake Ecodistrict (Smith, et al., 1998).

In general, the soils within the Granville Lake Ecodistrict are described as compact clayey subsoil with poor structure, low soil temperatures, large amounts of cobbles and stones, coarse surface textures, and the poor water and nutrient holding capacity of the till all greatly constrain the use of mineral soils

(Smith, et al., 1998). In the site, overburden depths, are estimated (based on open pit area and overburden volumes provided by Minnova) to range from 1.5 m to 6.0 m in depth.

Historic Soil Capability Mapping project of the Canada Land Inventory (1973a) covered the Cormorant Lake area (N.T.S. area 63K) immediately to the south of the PLGP area, but did not extend north beyond latitude 55°N. However, by extrapolation, it would appear that the immediate area around Puffy and Ragged Lakes has either organic soils or Class 7 soils. The latter are described as having no capability for arable culture or permanent pasture, particularly because of the shallowness to bedrock. The organic soils are described as Fibrisols, mainly associated with poorly drained sphagnum peat and Mesisols, which are associated with: (a) very poorly drained fen peat, derived mostly from sedges and mosses; and (b) poorly drained forest peat derived from black spruce, tamarack, ericaceous shrubs, and feather mosses.

The area lies in the zone of discontinuous permafrost. About 5% of the organic soils in the Cormorant Lake area to the south are described as being permanently frozen.

20.10 TERRESTRIAL PLANT AND ANIMAL LIFE

The area is located in the Churchill River Upland Ecoregion, which is established on shield rock with fens and bogs developed extensively across the landscape. Bedrock outcrops are common and typically support open lichen woodlands. The area is in the eastern extent of this ecoregion where sand outwash areas support Jack Pine (Pinus banksiana) stands. Wildfire is the predominant natural source of tree stand renewal that supports widespread Jack Pine, White Birch (Betula papyrifera), and Trembling Leaf Aspen (Populus tremuloides) growth (Smith, et al., 1998).

20.11 REGIONAL VEGETATION ANALYSIS

The vegetation that may be present at the site, area, and region was characterised using a regional analysis of vegetation, as defined by the Forestry Branch of Manitoba Conservation. It should be noted that the site encompasses an area and is greater than the anticipated footprint resulting from the re-start of the underground mining and proposed development of open pits, waste stockpiles, and existing and new haulage roads. Potentially unique vegetation types within the area or region were determined using Forest Management Units (“FMU”). The Forestry Branch of Manitoba Conservation creates forest inventory maps that are developed from interpretation of 1:15,840 aerial photography.

Each forest inventory map covers one township (96 km^2). For purposes of indexing and assembling the data, multiple townships of data are packaged into a FMU. Within each FMU package, the individual townships are maintained as separate files along with their associated attributes. Each FMU identifies the vegetation cover class of the FMU and identifies the species composition based on a hierarchical series of attributes (*i.e.*, land cover, productivity, tree type, and species composition). This cover class identifies a unique area of tree canopy that combines a series of attributes and species composition that can be interpolated into a general habitat classification. The FMU is the most detailed vegetation identification information available for the undeveloped portions of the province.

The forestry inventory maps created for this region predate the 1989 forest fire that impacted the majority of the site. Therefore, although these maps do not accurately reflect the current baseline conditions, the vegetative communities present on the forestry inventory maps are those that will likely develop over time as the forest regenerates following the 1989 forest fire. Further, it is anticipated that the vegetation regeneration will be determined by soils, topography, and water availability that, for the most part, were all similarly affected by the fire. Therefore, although the FMU mapping does not capture the vegetation changes by the 1989 forest fire, it does reflect the similar conditions for vegetation growth within the broader area and region and can provide context to determine if the growth conditions for vegetation are unique to the site or if similar conditions are available in the broader area and region. A description of the vegetation observed at the site is provided in the next section.

20.12 TERRESTRIAL FIELD SURVEYS

During a Heritage Resource Impact Assessment ("HRIA") conducted by Ilam Associates Ltd in May 1987, an examination of the floral and faunal communities near the PLGP area, including the Ragged TMA, was conducted. Subsequent to the HRIA, a large forest fire came through the area of the PLGP property in 1989. More recently, the PLGP area was surveyed on September 24-25, 2012 by AECOM to assess the current vegetation and animal communities in this area, and the potential for the presence of rare or endangered species. The AECOM field team walked each proposed open pit location in a random pattern covering distinct vegetation types present in the area. The characteristics and dominant species present were recorded for each area surveyed. In addition, a specific search was made for unique vegetative habitats that might harbour rare or endangered species within the site. Photographs were collected of each proposed pit and recorded by location on a handheld GPS unit.

The site shows evidence of the 1989 forest fire. Natural regeneration is progressing at a normal rate and the area is returning to a typical post-burn forest stand. The majority of forest on the rocky uplands shows evidence of being affected by the 1989 forest fire. There are isolated stands of older forest that may have escaped burning. Fire events, and subsequent regeneration, are a natural and common feature of the boreal zones of Canada. The bog areas that are targeted for pit development show little evidence of fire due to their wet nature. Some tree growth in the bogs was burned and ground cover in the wet meadows adjacent to the bogs shows evidence of past burning.

Ground nesting birds and small mammal species, such as Snowshoe Hare (Lepus americanus) and Meadow Vole (Microtus pennsylvanicus), will make use of burned areas once re-growth has progressed to the high shrub stage. This makes the area attractive to hawks and owls, as well as mammalian predators, such as Coyote (Canis latrans) and Short-Tailed Weasel (Mustela ermine). Burned areas do not provide high quality nesting habitat for birds, with the exception of the boundary between the burned areas and unburned remnant areas (edge habitats). A burned area is highly disturbed and does not contain habitat critical to wildlife species. Due to the impact of the 1989 forest fire and historical development at the PLPG as an operating mine site, the terrestrial habitat is in this area is not, at present, ideal to support diverse wildlife and would not represent an area that would be attractive for most animals that are commonly considered for trapping or hunting.

The existing clearings, trails, and roads provide edge habitat that benefits many species of nesting migratory birds, especially warblers. The extensive system of lakes and rivers in the region offers a large area potentially suitable for nesting migratory waterfowl. It should be noted, however, the waterfowl do not typically make extensive use of boreal areas for nesting. Similar to the findings of the 1987 terrestrial investigation, the mammal population of the site, as encountered in 2012, was quite limited, especially the ungulates, likely because of a lack of suitable habitat and food supply. Other mammals, including fur-bearers, game birds, and water fowl were also not numerous. One Lynx (Lynx canadensis) was observed along the Sherridon Road south of the PLGP main access road during the AECOM terrestrial survey in September 2012. The wildlife species that could potentially be found in the Mid-Boreal Lowland Ecoregion are listed in Table 20.4.

Table 20.4: Species of Mammals Found Throughout the Mid North Planning Zone of Manitoba	
Common Name	**Generic Name**
Masked Shrew	Sorex cinereus cinereus
Water Shrew	S. palustris palustris
Arctic Shrew	S. arcticus arcticus
Northern Pygmy Shrew	Microsorex hoyi intervectus
Little Brown Bat	Myotis lucifugus lucifugus
Hoary Bat	Lasiurus cinereus cinereus
Snowshoe Hare	Lepus americanus americanus
Little Northern Chipmunk	Eutamias minimus borealis
Woodchuck	Marmota monax canadensis
Red Squirrel	Tamiasciurus hudsonicus hudsonicus
Beaver	Castor canadensis canadensis
Meadow Vole	Microtus pennsylvanicus drummondii
Northern Bog Lemming	Synaptomys borealis smithi
Hudson Bay Jumping Mouse	Zapus hudsonius hudsonius
Muskrat	Ondatra zibethicus albus
Porcupine	Erethizon dorsatum dorsatum
Saskatchewan Timber Wolf	Canis lupus griseoalbus
Arctic Fox	Alopex lagopus innuitus
Red Fox	Vulpes fulva regalis
Black Bear	Ursus americanus americanus
Ermine (Richardson Weasel)	Mustela erminea richardsonii
Least Weasel	M. nivalis rixosa
Mink	M. vison locustris
Wolverine	Gulo gulo luscus
River Otter	Lutra canadensis preblei
Lynx	Lynx canadensis canadensis
Moose	Alces alces andersoni
Source: Teillet, 1979	

20.13 AQUATIC PLANT AND ANIMAL LIFE

Re-development of the PLGP, including future proposed open pits at the property, will require installation of a berm around Fire Pond and installation of haulage roads to support the proposed open pits. AECOM fisheries biologists conducted aquatic habitat assessments in the vicinity of the PLGP site in September 2012 to describe fish habitat potentially affected by re-development of the existing operations and future proposed open pit development, in three areas:

- Along the main access road, from the Sherridon road to the existing mill facility;
- Along the on-site road leading from the existing underground mine portal to the mill facility; and

- Within the vicinity of the proposed open pits, with particular emphasis within and downstream of Fire Pond.

At potential fish-bearing waterbodies or fish habitat locations, habitat characteristics, such as water depth, cover types, and substrates, were documented. The presence of culverts or other structures was also noted. Potential fish habitat use (*e.g.*, spawning or overwintering) was also assessed at the time of the field survey. At the time of the assessment, water levels were very low and as a result, potential fish habitats present during high water conditions could not be determined.

Unique or critical fish habitat was not observed during the aquatic habitat assessments. Fish habitat at watercourse crossings along the main access road provided, at most, marginal aquatic habitat value. Marginal habitats have low productive capacity and contribute marginally to fish production (Fisheries and Oceans Canada, 1998). Several small ponded areas were identified along the Main Access Road with no apparent connectivity to other waterbodies, even with higher water levels. Significant Beaver (Castor canadensis) activity in the area creates impediments to fish passage (Table 20.5).

Table 20.5: Summary of Aquatic Habitat Assessment along the Main Access Road, September 2012

Crossing ID	KM from Sherridon Road	Fish Use				Connectivity	Aquatic Habitat Sensitivity	Aquatic Habitat Value
		Spawning	Migration	Rearing	Over-Wintering			
PLM-01	8.4	Poor	Poor	Poor	Poor	Poor	Low	None
PLM-02	7.9	Poor	Poor	Poor	Poor	None	Low	None
PLM-03	5.5	Good	Poor	Good	Poor	Good	Low	Marginal
PLM-04	1.7	Good	Poor	Good	Poor	Poor	Low	Marginal

Several ponded areas and culverts were observed along the main access road. The ponded areas are generally low-lying areas in which water accumulates between bedrock outcrops and culverts at these locations permit drainage across the roadway. These ponded areas do not support fish and provide no fish habitat.

There were no potential fish-bearing waterbodies or fish habitat identified along the on-site roads and only two small ponded areas were observed. Limited connectivity and shallow water that would freeze to the bottom in winter limits the habitat value of these ponded areas.

There were no potential fish-bearing waterbodies or fish habitat observed in the vicinity of the proposed open pits with the exception of Fire Pond. General observations of the site indicate that there are no other waterbodies that could support fish habitat in the areas of other proposed developments (*e.g.*, waste rock stockpiles). Within the Fire Pond, cover was provided by overhanging vegetation, woody debris, and to a lesser degree, submergent vegetation. Maximum depth was 2.2 m and the substrate largely consisted of organics. A portion of the western shore was bed rock, but the majority of shoreline was composed of wetland grasses and shrubs. Typical of headwater lakes and ponds, Brook Stickleback was captured in Fire Pond.

Brook Stickleback are widely distributed and abundant throughout the province. There was no evidence of a creek or potential for fish or fish habitat downstream of the Fire Pond. The aquatic habitat value of Fire Pond is categorised as Marginal as it provides habitat only for small-bodied fish (*i.e.*, Brook Stickleback) and there is no connectivity to other waterbodies.

20.14 POTENTIAL FOR ACID ROCK DRAINAGE AND METAL LEACHING

A geochemical assessment was carried out to characterise potential of ore and waste development from PLGP to generate acidic leachate enriched in soluble contaminants (metals or metalloids), which may impact the environment following operation and closure of the PLGP. An understanding of the geochemistry of water-rock interactions is critical in order to assess the potentially adverse effects of mining projects on surface and groundwater quality.

Water quality may be adversely impacted by two types of rock/water interactions:

- Neutral rock drainage (NRD), involving:
  - Elevated total dissolved solids (TDS);
  - Oxy-anions, such as complexes of chromium, arsenic, and selenium (*i.e.*, chromate, arsenate, and selenate);
  - Elevated metals and metalloids soluble at neutral pH, such as selenium (Se), nickel (Ni), and copper (Cu).
- Acid rock drainage/metal leaching (ARD/ML):
  - Elevated sulphate and TDS;
  - Low pH water; and
  - Elevated metals soluble at acidic pH, such as aluminum (Al), iron (Fe), manganese (Mn), and copper Cu.

20.14.1 ACID DRAINAGE/METAL LEACHING

This section describes the basic science pertaining to the formation and mitigation of Acid Rock Drainage/Metal Leaching ("ARD/ML"). Neutral rock drainage is prevalent in high carbonate and/or ultrabasic facies and due to the presence of ultrabasic lithologies must be considered for the PLGP. Acid rock drainage, or ARD, commonly occurs in sulphide-enriched mine wastes via the microbially mediated oxidation of pyrite (or other sulphide minerals) as it is exposed to oxygen and water by mining activities. ARD impacted waters tend to have pH levels in the range of 2 to 4, and often contain elevated metal(loid) concentrations.

The following chemical reactions describe the generation of ARD due to oxidation of pyrite (FeS_2) and the resulting formation of sulphate (SO_4), amorphous ferric hydroxide (Fe(OH)3(s)), and acidity (H).

1. $FeS_2 + 15/4O_2 + 7/2H_2O \leftrightarrow Fe(OH)_{3(s)} + 2SO_4 + 4H$ where pH > 5
2. $FeS_2 + 14Fe^{3+} + 8H_2O \leftrightarrow 15Fe_2+ + 2SO_4 + 16H$ where pH < 3
3. $Fe^{2+} + 1/4O_2 + H^+ \leftrightarrow Fe^{3+} + 1/2H_2O$ where pH < 3

It should be noted that the molar ratio between pyrite and acidity is not the same for all three reactions. Reaction 1 produces four (4) moles of acidity for each mole of pyrite and Reaction 2 produces sixteen (16) moles of acidity per mole of pyrite oxidized.

Oxidation of ferrous to ferric iron in Reaction 3 requires bacterial activity to promote rapid acid generation, which typically occurs in low pH waters. The critical bacteria are usually site-specific strains of Acidi-Thiobaccillus ferro-oxidans that utilize the ferrous iron as a metabolic electron acceptor instead of oxygen. These bacteria do not require organic carbon as an energy source and obtain their nutritional needs from atmospheric gases (nitrogen, oxygen, carbon dioxide, and water) and from minerals (sulphur and phosphorus). While the bacteria are not catalysts by true definition, they do act as accelerating agents in the generation of ARD.

Acidity formed by either Reaction 1 or coupled Reactions 2 and 3 may be neutralized by other minerals present in the same rock. For example, calcite (Reactions 4 and 5) rapidly neutralizes acidity and buffers the mine water at a pH of around 6.5 to 8.0. Many other minerals, such as anorthite (Reaction 6) or chlorite, may neutralize acidity (termed silicate buffering), but reactions are often kinetically slow and the pH may be buffered at lower levels (*e.g.*, 5.5 or less) and silicate buffering tends to liberate increased concentrations of Al into solution.

1. $CaCO_3 + 2H^{+1} \rightarrow Ca^{+2} + CO_2 + H_2O$ below a pH of 6
2. $CaCO_3 + H^{+1} \rightarrow Ca^{+2} + HCO_3$ above a pH of 6
3. $CaAl_2Si_2O_8 + 2H^{+1} + H_2O \rightarrow Ca^{+2} + Al_2Si_2O_5(OH)_4$

Once acidic water has been generated, the ongoing oxidation is self-perpetuating and may even continue (at a reduced rate) in the absence of oxygen, if a sufficient concentration of Fe^{3+} has accumulated. Acidic water will readily mobilize cations (such as copper (Cu), lead (Pb), zinc (Zn), etc.) present in minerals in the rock resulting in potentially disastrous impacts to the receiving environment.

ARD can be mitigated by controlling the availability of water to the pyrite; without water, ARD can neither form nor migrate. ARD can also be mitigated by controlling the oxygen available to the pyrite. As can be seen from Reactions 1 and 3, oxygen is a critical reactant. Removing this element will cause the reactions to slow or end. A common method of preventing ARD/ML is to submerge rocks likely to generate it under at least 1 m of water although there are a number of alternative long-term storage techniques intended to achieve the same end. All sulphide minerals may generate ARD although the rates of reaction vary quite widely.

20.14.2 Assessing Acid Mine Drainage and Metal Leaching at the PLGP

The testing methodology for ARD/ML is broken into two sections: (i) static testing and (ii) kinetic testing. Static testing involves a series of relatively low cost screening tools including determining rock sample mineralogy, total element content of rock samples, and Acid-Base Accounting ("ABA"). ABA testing involves measuring the potential of the rock to Produce Acidity ("AP") and the ability of the rock to buffer acidity ("NP").

Thirty-four (34) samples of drill core (ore and waste rock) from ten (10) drill holes at PLGP were submitted for the following static tests in order to characterize ARD/ML:

- Acid base accounting (including, sulphide, sulphate, paste pH, NP, AP, etc.);
- Whole rock analysis with lithium metaborate fusion for major elements;

- Four acid digest with ICP-MS/OES for trace elements; and
- Shake flask testing with leachate analysis by ICP/MS and IC.

20.14.3 SAMPLE REPRESENTATIVITY

It is important that the appropriate number and size of samples be selected for assessment of ARD/ML so that realistic conclusions can be drawn regarding the potential of the rocks remaining exposed in the open pit after the cessation of mining to generate ARD or ML. The number of samples that must be collected in order to detect the presence of potentially ARD/ML generating lithologies depends upon the variability of those lithologies. Lithologies that are essentially homogenous require only a few samples in order to characterize their ARD/ML potential. Lithologies, which demonstrate greater variability, require a greater sampling rate in order to thoroughly test for ARD/ML. Core from ten (10) drill holes was available for sampling. The core was inspected visually and thirty-four (34) samples, each weighing at least 2 kg were selected. Samples of historic tails and waste rock were also collected although it is considered they are not representative of the deposit, as a whole, they do at least provide a guide to potential historic contamination issues. Samples of drill core and historic waste rock (3 samples) are listed as Table 20.6.

Table 20.6: PLGP – ARD/ML Sample Location Data

DDH ID	Zone	Sample	From (m)	To (m)	Interval (m)	Lithology Notes
A3-03	PIT 4	1193863	5.18	5.78	0.6	Mafic Schist
A3-03	PIT 4	1193864	14	14.6	0.6	Intermediate Schist
A3-03	PIT 4	1193865	66	66.6	0.6	Inter-Mafic Schist trace pyrite
A3-03	PIT 4	1193866	77	77.6	0.6	Intermediate Schist trace pyrite
A3-04	PIT 4	1193859	20.42	21.02	0.6	Inter-Mafic Schist trace pyrite
A3-04	PIT 4	1193860	41.76	42.46	0.7	Intermediate Schist
A3-04	PIT 4	1193861	67	67.6	0.6	Intermediate Schist trace asp 1% py
A3-04	PIT 4	1193862	88	88.6	0.6	Conglomerate
A3-10	*	1193856	5.18	5.78	0.6	Mafic Schist minor pyrite
A3-10	*	1193857	32.61	33.21	0.6	Hangingwall Gneiss
A3-10	*	1193858	75.28	75.88	0.6	Inter-Mafic Schist minor pyrite
A3-11	*	1193854	25.9	26.52	0.62	Mafic Schist
A3-11	*	1193855	209.98	210.6	0.62	PLAZ (no mention in log of sulphide but could contain trace py,asp)
A3-24	PIT 1	1193873	17.4	18	0.6	Mafic Schist
A3-24	PIT 1	1193874	19.5	20.7	1.2	Felsic Schist Intrusive?
A3-24	PIT 1	1193875	23.5	24.7	1.2	Mafic Schist trace to 1% py, asp
A3-24	PIT 1	1193876	30	40.2	10.2	Mafic Schist >3% py, asp (note 1/4 split why interval is so large)
A3-25	PIT 1	1193877	19	19.6	0.6	Felsic Schist Intrusive?
A3-25	PIT 1	1193878	22	23.2	1.2	Mafic Schist trace to 1% py,asp
A3-25	PIT 1	1193879	28	29.2	1.2	Mafic Schist >3% py, asp

Table 20.6: PLGP – ARD/ML Sample Location Data						
DDH ID	**Zone**	**Sample**	**From (m)**	**To (m)**	**Interval (m)**	**Lithology Notes**
A3-25	PIT 1	1193880	31.5	32.7	1.2	Mafic Schist >3% py, asp
A3-31	PIT 3	1193886	5	5.6	0.6	Intermediate Schist
A3-31	PIT 3	1193887	10	10.6	0.6	Intermediate Schist
A3-31	PIT 3	1193872	15.27	15.87	0.6	Mafic Schist trace to 3% py,asp
A3-31	PIT 3	1193888	18.6	19.8	1.2	Mafic Schist trace to 1% py,asp
A3-31	PIT 3	1193871	23.47	24.07	0.6	PLAZ (no mention in log of sulphide but likely contain trace py,asp)
A3-32	PIT 3	1193881	5	5.6	0.6	Mafic Schist
A3-32	PIT 3	1193882	10	10.6	0.6	Altered Schist
A3-32	PIT 3	1193883	15	15.6	0.6	Mafic Schist
A3-32	PIT 3	1193884	21	21.6	0.6	Mafic Schist
A3-32	PIT 3	1193885	26	26.6	0.6	Mafic Schist trace >3% py, asp
A3-35	PIT 3	1193868	8.23	8.83	0.6	Mafic Schist
A3-35	PIT 3	1193869	14.33	14.93	0.6	Altered Schist minor py
A3-35	PIT 3	1193870	29.56	30.16	0.6	Mafic Schist minor pyrite
A3-37	PIT 4	1193867	27.89	28.49	0.6	Mafic Schist 1-5% pyrite trace asp
Waste Rock Pile		1193851	NA	NA	NA	Muck Pile
Waste Rock Pile		1193852	NA	NA	NA	Muck Pile
Waste Rock Pile		1193853	NA	NA	NA	Muck Pile

20.14.4 STATIC TESTING DATA

Rock samples collected from drill core were submitted to Agat Laboratories, Toronto, Canada for static testing.

Samples collected at PLGP were collected for both waste rock and ore. It is important to distinguish between these two rock classifications since ore is not usually stored at a mine site for a significant length of time; and therefore, does not usually have an opportunity to develop acid rock drainage or metal leaching. Gold was not determined on the samples submitted for static ARD testing, which makes it difficult to separate ore from waste rock samples. Ostry and Halden (1995), however, note that the gold mineralization at the PLGP is invariably accompanied by sphalerite and arsenopyrite although the latter mineral is often found up to 2 m beyond the edges of the auriferous veins as well. Figure 20.1 shows the close correlation between arsenic and zinc for the drill core geochemical data.

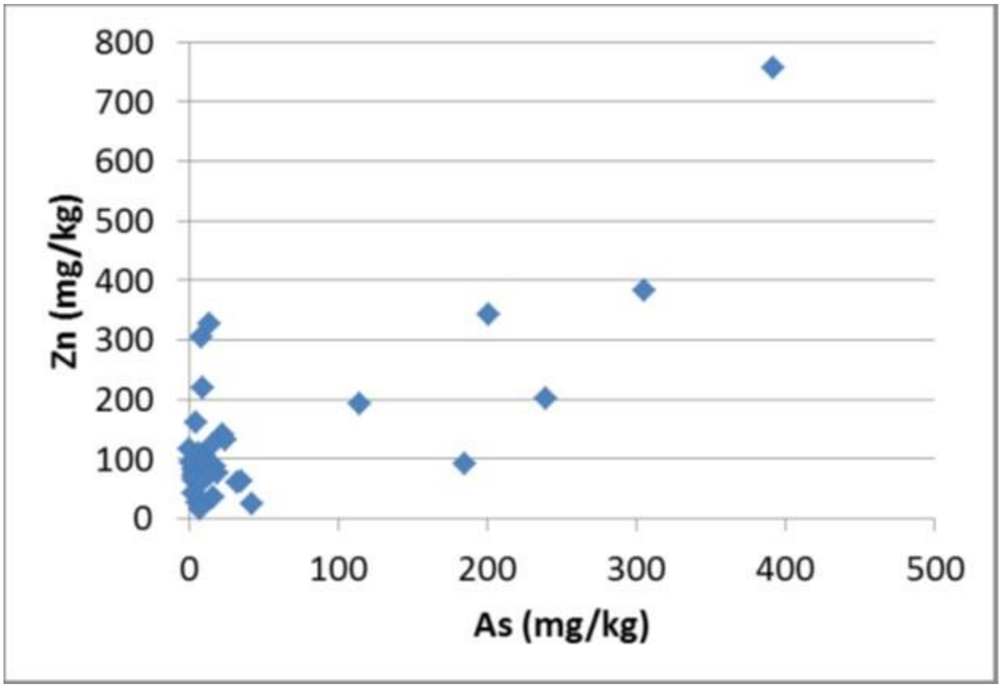


Figure 20.1 PLGP – Correlation between arsenic and zinc in drill core samples

A threshold value for zinc in auriferous ore samples was selected using a cumulative probability plot (Figure 20.2). A threshold for zinc content of auriferous ore, approximately 85 mg/kg, was selected as being the most appropriate. The static data are interpreted on the basis of this division.

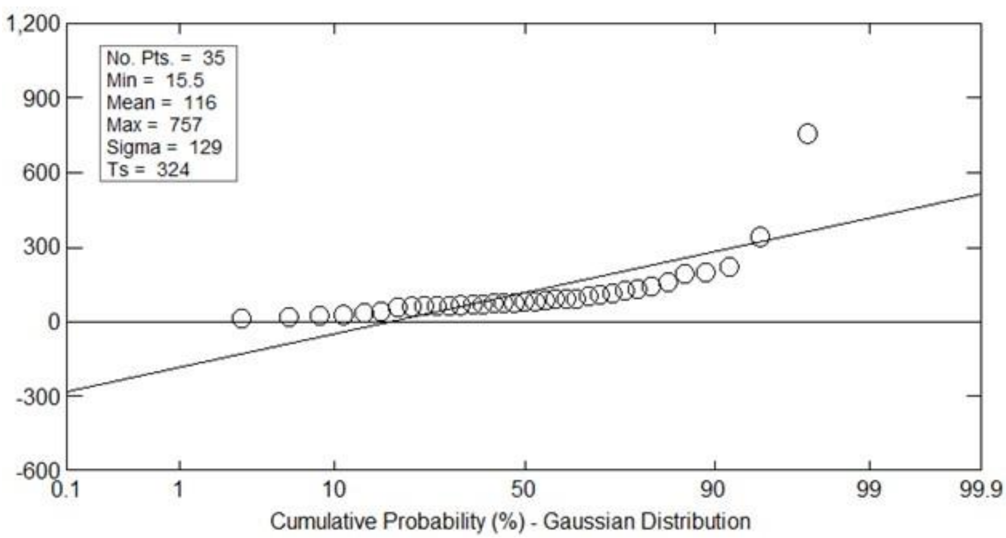


Figure 20.2 PLGP – Cumulative probability Plot of Zn in drill core

Means and standard deviations of multi-element whole rock analyses for ore and waste rock as well as crustal abundance data (5XCA) is also provided in Table 20.7. Parameters that are higher than five times the crustal abundance data represent potential contaminants of concern for the PLGP. Based on this comparison, potential contaminants of concern include: Ag, As, Bi, Cd, Ga, Mo, S, Sb, Se, Te, and W. This static test does not mean that these potential contaminants will be an issue; rather that if present in soluble or reactive minerals, they might be a contamination issue. The waste rock data shows marked fewer elements that exceed the 5XCA (As, Bi, Se, W). It is worth noting that from an exploration perspective, these elements may be considered 'pathfinder' elements.

Table 20.7: PLGP – Mean and Standard Deviations of Multi-Element Data Compared with Crustal Abundances

	Ag	As	Bi	Cd	Mo	S	Sb	Se	Te	W
SAMPID	0.01	0.2	0.01	0.02	0.05	0.01	0.05	0.5	0.01	0.1
Units	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm
5XCA	0.4	9	0.041	0.8	6	0.85	1	0.25	0.005	5.5
WASTE		12	0.2					0.7		23
SD		11	0.2					0.1		57
ORE	0.5	78	0.3	0.5	3.5	0.6	0.5	1.5	0.1	18
SD	0.4	120	0.5	0.5	4.2	0.6	0.4	1.2	0.1	21

Major elements were also determined on the drill core and trench samples using a lithium metaborate fusion and an ICP-MS finish. The lithium metaborate fusion ensures that the whole sample is completely digested, and therefore, this analytical method represents a total analysis for the major elements.

20.14.5 Shake Flask Testing

Shake Flask Extraction ("SFE") testing typically involves the addition of deionized water to jaw crushed sample and end-over-end shaking for twenty-four (24) hours. The extraction is, therefore, quite vigorous, exceeding what may be expected in most natural situations, and therefore, provides a worst case estimate of leachate water quality. Humidity cell leachate data after equilibration has been reached usually provides a much better estimate of leachate water quality in field conditions. The SFE leachate was analyzed by inductively coupled argon plasma-mass spectrometry ("ICP-MS") after completion of the SFE extraction. The SFE leachate data by sample together with Manitoba water quality guidelines ("MBQL") are presented in Table 20.8. Statistics (mean and standard deviation) are only presented for elemental concentrations regulated by MBQL.

Table 20.8: PLGP – Shake Flask Extraction Data and Manitoba Tier III Water Quality Standards

Units	Al mg/L	As mg/L	Cu mg/L	Fe mg/L	Pb mg/L	Mo mg/L	Ni mg/L	Se mg/L	Ag mg/L	Th mg/L	Zn mg/L
ORE											
Mean	8.27	0.3	0.0061	0.0076	7.1	0.016	0.0073	0.0054	0.001	0.002	0.046
Std Dev	9.53	0.6	0.0060	0.0084	8.3	0.020	0.0069	0.0067	7E-04	0.002	0.066
WASTE											
Mean	14.09	0.87	0.0038	0.0089	4.4	0.0070	0.0087	0.0010	6E-04	0.001	0.021
Std Dev	15.46	1.641	0.0018	0.0076	3.5	0.0072	0.0090	0.0000	4E-04	6E-04	0.023
Tier III	0.1	0.15	0.0021		0.0095	0.073	0.012	0.001	1E-04	8E-04	0.028

20.14.6 ACID BASE ACCOUNTING

Acid Base Accounting ("ABA") is a methodology initially devised in British Columbia to delineate the Acid Rock Drainage ("ARD") potential of rocks. The current standard of ABA analysis is amply described by Price (2009). Basically, the technique analyses powdered rock for minerals (as elements) that will produce acidity (*e.g.*, pyrite, pyrrhotite) and minerals that contribute neutralization potential (*e.g.*, calcite, plagioclase). The ABA data for the PLGP ore samples is presented as Table 20.9. The ABA data for the ore samples at PLGP show that a large majority of the samples (approximately 75% depending on whether NP/APP or NNP is used as the criterion of ARD) are very likely to generate ARD. This is to be expected for samples of ore and since the ore will not be stored for extended periods at the mine, no particular mitigation strategies are required for its storage on the basis of ARD generation.

Table 20.9: PLGP – Acid Base Accounting Data for Ore Samples

Sample	Fizz Test	Paste (pH)	S (total) (%)	Sulfate (%)	Sulfide (%)	NP $kgCaCO_3/t$	APP $kgCaCO_3/t$	NNP $kgCaCO_3/t$	NP/APP (%)
ID			0.005	0.01	0.01				
1193878	0	9.52	0.965	<0.01	0.96	14.7	30.1	-15.4	0.489
1193880	1	9.58	4.05	<0.01	4.05	24.6	127	-102	0.195
1193859	0	9.87	0.165	0.02	0.14	26.7	4.43	22.2	6.01
1193876	0	9.94	1.88	<0.01	1.88	-6.14	58.6	-64.7	-0.105
1193875	0	9.92	0.875	<0.01	0.87	29.7	27.3	2.4	1.09
1193865	0	9.89	0.395	0.04	0.36	14.2	11.2	2.95	1.26
1193872	0	9.75	0.771	0.04	0.73	9.17	22.9	-13.7	0.4
1193879	1	10.2	0.53	<0.01	0.53	32	16.5	15.4	1.94
1193861	0	10.1	0.707	0.04	0.66	16.4	20.7	-4.33	0.791
1193874	2	9.97	0.236	<0.01	0.24	62.8	7.36	55.5	8.54
1193871	0	9.72	0.392	0.03	0.36	13.6	11.4	2.27	1.2
1193864	0	9.69	0.038	0.02	0.02	8.68	0.515	8.16	16.9
1193855	0	10.3	0.233	0.01	0.22	18.6	6.86	11.7	2.7
1193873	0	9.32	0.414	<0.01	0.41	6.13	12.9	-6.77	0.475
1193888	0	8.72	0.652	0.03	0.63	7.4	19.6	-12.18	0.378
1193867	0	9.88	0.361	0.02	0.34	12.4	10.6	1.81	1.17

The ABA data for the waste rock samples at PLGP (presented as Table 20.10) show that a large majority of the samples (approximately 90% depending on whether NP/APP or NNP is used as the criterion of ARD) are very unlikely to generate ARD. One of the PAG samples has an NNP value very close to zero although the NP/APP ratio is zero suggesting that this sample may generate ARD. Approximately 10% of the waste rock samples will require special mitigation measures for their long term storage.

Table 20.10: PLGP – Acid Base Accounting Data for Waste Rock Samples									
Sample	Fizz Test	Paste (pH)	S (total) (%)	Sulfate (%)	Sulfide (%)	NP $kgCaCO_3/t$	APP $kgCaCO_3/t$	NNP kg $CaCO_3/t$	NP/APP (%)
ID			0.005	0.01	0.01				
1193854	0	10.1	0.011	<0.01	<0.01	18.6	0.186	18.4	100
1193884	0	9.92	0.009	<0.01	<0.01	2.48	0.014	2.46	178
1193877	0	9.82	0.22	<0.01	0.22	23.5	6.85	16.6	3.43
1193887	0	9.9	0.011	0.01	<0.01	1.24	0.03	1.21	41
1193881	0	9.8	0.034	<0.01	0.03	-7.37	0.816	-8.18	-9.03
1193866	0	10.1	0.198	0.02	0.18	9.85	5.49	4.36	1.79
1193863	0	9.64	0.054	<0.01	0.04	11.2	1.39	9.79	8.04
1193856	0	9.93	0.137	0.02	0.12	17.8	3.65	14.2	4.88
1193860	0	9.85	0.159	0.03	0.13	20.2	4.09	16.1	4.95
1193868	0	9.02	0.114	<0.01	0.11	13.5	3.29	10.2	4.11
1193885	0	9.46	0.287	<0.01	0.28	14.9	8.8	6.1	1.69
1193870	0	9.99	<0.005	<0.01	<0.01	11	0.011	11	1,010
1193886	0	9.37	0.011	<0.01	<0.01	7.42	0.199	7.22	37.3
1193869	0	9.45	0.005	<0.01	<0.01	8.59	0.056	8.53	154
1193883	0	8.98	0.031	0.02	0.02	9.94	0.483	9.46	20.6
1193882	0	9.07	0.012	<0.01	0.01	0	0.318	-0.318	0
1193862	0	9.48	0.049	0.01	0.04	3.12	1.15	1.97	2.72
1193858	0	9.39	0.041	0.03	0.01	10.6	0.429	10.2	24.7
1193857	0	9.87	0.008	<0.01	<0.01	11.8	0.131	11.7	90.2

20.14.7 HISTORIC WASTE ROCK AND TAILINGS

A relatively small amount of waste rock and tailings are present on the property from previous mining activities. This material was tested to assess whether it would need mitigation during mining operations. Five (5) samples of waste rock and six (6) samples of historic tails were collected and put through the suite of standard static tests.

Table 20.10 presents a comparison between a five (5) acid digest with ICP-MS analysis of waste rock (whole rock analysis) compared to five (5) times the crustal abundance (5XCA) data and the same comparison for the tailings samples is presented as Table 20.11.

Table 20.11: PLGP – Waste rock Analytical Data Compared to 5X Crustal Abundance Data[1]								
Waste Rocks		RDL	5XCA	WR1	WR3	WR5	WR2	WR4
Ag	ppm	0.01	0.4	1.13	2.27	11.3	1.85	1.62
As	ppm	0.2	9	13.6	8.4	305	29	49.4
Bi	ppm	0.01	0.041	0.66	0.53	14.7	5.97	0.73
Cd	ppm	0.02	0.8	1.47	4.04	3.31	1.97	1.2
Co	ppm	0.05	130	28.7	5.1	17.9	37.2	24.7
Cr	ppm	0.5	700	210	176	803	550	229
Mo	ppm	0.05	6	8.48	24.6	7.85	2.4	15.5
Pb	ppm	0.1	65	9.3	43.1	5,160	334	42.1
Re	ppm	0.002	0.013	0.016	0.037	0.008	0	0.023
S	%	0.01	0.85	3.13	6.36	1.69	1.26	4.6
Sb	ppm	0.05	1	0.34	0.15	7.11	0.48	0.39
Se	ppm	0.5	0.25	4.3	8.1	5	2.1	5.6
Te	ppm	0.01	0.005	0.23	0.31	0.79	<0.01	<0.01
W	ppm	0.1	6	61.4	2.8	9.4	3.4	5.8
Zn	ppm	0.5	380	327	304	384	329	208
[1]Only parameters exceeding the 5XCA are presented.								

Potential contaminants of concern for the waste rock are Ag, As, Bi, Cd, Co, Cr, Mo, Pb, Re, S, Sb, Se, Te, W, and Zn. There are fewer potential metals of concern for the tailings (Table 20.12).

Table 20.12: PLGP – Historic Tailings Analytical Data Compared to 5X Crustal Abundance Data									
Tailings	Units	RDL	5XCA	Tail 3	Tail 4	Tail 5	Tail 6	Tail 10	Tail 11
Ag	ppm	0.01	0.4	0.99	1.02	0.91	1.41	1.02	1.66
As	ppm	0.2	9	4,670	2,390	2,230	8,490	215	1,700
Bi	ppm	0.01	0.041	1.11	1.24	1.25	2.04	1.36	2.14
Cd	ppm	0.02	0.8	0.85	2.28	1.55	0.93	1.87	2.14
S	%	0.01	0.85	0.93	0.78	0.71	1.08	1.19	2.2
Sb	ppm	0.05	1	1.11	1.04	0.86	1.53	0.85	2.21
Se	ppm	0.5	0.25	2.1	1.5	1.6	1.9	1.6	2.9
U	ppm	0.005	9	1.55	1.5	1.5	1.35	1.5	1.8
W	ppm	0.1	6	21	22.3	15.7	31.6	18.7	60.5
Zn	ppm	0.5	380	217	637	331	232	475	579

20.14.8 Shake Flask Testing Historic Waste Rock and Tails

Shake flask testing was also carried out on the historic waste rock and tails in order to test which potential contaminants may actually present a problem in terms of ML at the PLGP. These data are presented in Table 20.13 as means and standard deviations for both historic waste rocks and tailings. Tier III standards are exceeded for waste rocks and tailings for all but Mo and the tailings results indicate a much higher tendency toward metal leaching as might be expected from grain size effects.

Table 20.13: PLGP – Historic Tailings SFE Data Compared To Tier III Manitoba Water Quality Standards									
	Al mg/L	**As mg/L**	**Cu mg/L**	**Pb mg/L**	**Mo mg/L**	**Ni mg/L**	**Se mg/L**	**Ag mg/L**	**Zn mg/L**
WASTE									
Mean	0.53	0.049	0.0030	0.017	0.00185	0.071	0.0043	0.00090	0.073
Standard Deviation	0.72	0.051	0.0010	0.023	0.00092	0.093	0.0033	0.00071	0.131
TAILS									
Mean	51.34	1.18	5.98	0.015	0.011	1.34	0.018	0.0020	9.20
Standard Deviation	77.81	1.12	8.99	0.0074	0.017	2.00	0.010	0.0012	13.96
Tier III	0.1	0.15	0.0021	0.0095	0.073	0.012	0.001	1.00E-04	0.028

Table 20.14 shows the results of the ABA testing on both the historic waste rock and tailings and shows that the samples of both materials are indicative of potentially acid generating (PAG) material.

Table 20.14: PLGP – ABA Data for Historic Waste Rock and Tails								
	Paste (pH)	**S (total) (%)**	**Sulfate (%)**	**Sulfide (%)**	**NP $kgCaCO_3/t$**	**APP $kgCaCO_3/t$**	**NNP**	**NP/APP**
							kg $CaCO_3/t$	**(%)**
WR1	9.19	3.77	<0.01	3.77	8.47	118	-109	0.072
WR3	5.75	7.67	<0.01	7.66	0.619	239	-238	0.003
WR5	7.68	1.85	<0.01	1.85	3.73	57.8	-54.1	0.064
WR2	8.8	1.17	0.023	1.14	19	35.7	-16.72	0.53
WR4	6.11	4.72	0.07	4.65	3.09	145	-142.1	0.02
Tails #3	5.34	0.852	0.6	0.25	0	8.03	-8.03	0
Tails #4	6.85	0.665	0.08	0.59	1.2	18.2	-17	0.07
Tails #5	7.65	0.597	0.25	0.35	4.92	10.8	-5.88	0.46
Tails #6	4.37	0.893	0.5	0.39	30.3	12.2	18.1	2.48
Tails #10	9.04	1.18	<0.01	1.18	11.7	36.7	-24.9	0.32
Tails #11	8.22	2.16	0.3	1.86	10.2	58.3	-48	0.18

20.14.9 AMD CONCLUSIONS AND RECOMMENDATIONS

The key conclusions of the various studies assessing the environmental impact of ARD/ML are summarized below.

- Waste rock generated at the PLGP is potentially acid generating (“PAG”). Runoff from the waste rock will be managed in the TMA and will be treated before re-use as process water or discharge to the environment. Waste rock may require co-disposal in the tailings storage facility;

- Wall rock left exposed after extraction of the ore are generally very likely to generate acid rock drainage and metal leaching after exposure to atmospheric air and water where any ore remains in the wall rock. Mitigation strategies will probably be required for any open pits left open to the atmosphere;

- Stockpiles of ore containing any sulphide are generally very likely to generate acid rock drainage and possibly arsenic leaching prior to processing. Tailings will also likely be PAG. A layer of water should be maintained over the tailings to minimize oxygen contact. Water from the TMA will be treated before being recycled for process use or before being discharged to the environment; and

- Historic waste rock and tailings are very likely to generate ARD/ML but will be treated with other such material during mining operations. The historic tails do not necessarily mean that tailings from the proposed operations will generate ARD/ML but do strongly suggest that ARD/ML testing should be carried out on bench test tailings. In addition, it is not considered that the historic waste rock is representative of waste rock to be extracted by the proposed mining operations; in fact, the historic waste rock bears a strong similarity to ore grade rocks.

It is expected that the mine will regularly sample and test exposed rock formations and tailings and take measures, as necessary, to prevent or control ARD/ML.

20.15 SITE WATER BALANCE

The mean annual precipitation recorded at the closest weather station (Flin Flon) is 478 mm and the mean annual pan evaporation rate is estimated to be approximately 400 mm for this area. The average runoff coefficient is estimated to be 0.70. These precipitation, evaporation, and runoff coefficient data and estimates were used to estimate mean annual runoff for the TMA and waste rock area.

20.15.1 TAILINGS RUNOFF

The TMA catchment area is 320 ha. The mean annual precipitation for the TMA is estimated to be 1,529,600 m^3 and the mean evaporation volume is roughly estimated to be approximately 610,000 m^3.

20.15.2 RUNOFF FROM WASTE ROCK

The waste rock catchment area is 5.25 ha. The mean annual runoff from the waste rock is estimated to be 17,566 m^3.

20.15.3 MINE WATER SEEPAGE

Preliminary estimates of seepage rates into the open pits and underground workings have been made based on the groundwater testing conducted on ten (10) monitoring wells installed by WESA in 2011. Hydraulic conductivities at the ten (10) groundwater monitoring wells ranged from 1×10^{-8} m/s to 1×10^{-4} m/s. Using a geometric mean bulk hydraulic conductivity of 5×10^{-7} m/s, potential pit inflow estimates ranged from 1.3 L/s (112 m^3/day) to 4.4 L/s (380 m^3/day). The higher rate has been used in the water balance to be conservative. Similarly, a preliminary estimate has been made of the potential seepage into the underground working using the hydraulic conductivity estimates and the estimated extent of the underground workings. A maximum seepage rate of 550 m^3/day has been estimated.

The TMA will be used to store runoff from the TMA area and from the waste rock area, seepage into the underground workings and open pits, and process water. Runoff from the waste rock area will be conveyed to the TMA via a channel. Mine water will be pumped to the TMA from underground and in pit sump pumps.

Water in the TMA will be treated by an Effluent Treatment Plant ("ETP") prior to discharge to the environment. Treated water will also be recycled back to the mill and underground for process use

The water balance indicates that there will be a surplus of water that will be discharged to the environment over the course of the operational season. For a 600 tpd milling rate, the surplus is approximately 5,710 m^3/day (873 IGPM) and for a 1,000 tpd operation the surplus is 6,546 m^3/day (1,000 IGPM). The TMA should be managed so that a layer of water is maintained over deposited tailings to minimize oxygen contact with the tailings. While maintaining this minimum water column thickness, the water level in the TMA should be drawn down to the lowest level possible (via treatment and recycling/discharge to the environment) so that maximum TMA capacity is available for the spring melt. The largest impacts on the water balance are the precipitation and evaporation rates, so the balance will vary from year to year depending on snow and rain levels and temperatures. The largest unknown in the water balance is the amount of groundwater that will seep into the pits and underground workings. The balance assumes that treated water from the TMA will be used for process use in the mill and underground (Table 20.15).

Table 20.15: Site Water Balance

Component	600 tpd m^3/year	m^3/day over 180 day Operational Period	1 000 tpd m^3/year	m^3/day over 180 day Operational Period
TMA Precipitation	1,529,600		1,529,600	
Tailings Pore Water	564,144		940,240	
Seepage into Underground and Open Pits	340,000		340,000	
Waste Rock Runoff	17,566		17566	
Total Inputs	2,451,310	16,812	2,827,406	
TMA Basin Evaporation	610,000		610,000	
Mill Reclaim	338,486		564,144	
Process Water	475,000		475,000	Process Water
Total Outputs	1,423,486		1,649,144	
Surplus	1,027,824	5,710	1,178,262	6,546

20.16 CAMP WATER AND SEWAGE

The current plan for the construction camp indicates it will contain sixty (60) beds and the permanent camp is estimated to be one hundred (100) beds. The water treatment and sewage plants were estimated at a maximum of eighty-five to ninety (85-90) people on site at any one time, with a daily usage of 454 litres/person for a total daily usage of 40,860 litres, or 40 m^3/day. Two identical plants were specified to treat up to 35 m^3/d, one for the main camp and one for the mill/dry, which would also service the construction camp. This should supply approximately double the required on site daily usage. The sewage treatment systems were also sized for similar volumes.

20.17 WASTE ROCK MANAGEMENT

Waste rock will be stored in the historical waste storage area near the existing portal. The footprint of the waste rock pile will be approximately 350 m × 150 m. Approximately 2,700,000 tonnes will be generated from the open pit and approximately 660,000 tonnes will be generated from underground mining. Mill throughput is estimated at 1,266,438 tonnes for the current life-of-mine. Of this, 60% should go underground as backfill.

Water balance estimates indicate approximately 15,566 m^3 of contact water will need to be managed from the waste rock area annually. This water will be collected in perimeter ditching and will be conveyed to the TMA for mixing and subsequent treatment in the ETP.

20.18 TAILINGS MANAGEMENT

Tailings are proposed to be stored in the existing Ragged Lake TMA. Regulatory changes that have come into effect since the mine last operated will require that an application be made for an MMER Schedule 2 listing that will need to assess the options that are available and that were assessed for tailings management. The TMA has an area of approximately 1.54 km^2. It is estimated that approximately 350,000 m^3 of tailings exist in the TMA from past operations, and it is estimated that 1,300,000 m^3 of tailings will be generated over the course of mine development. To accommodate this volume, a dam with flow control structure will need to be constructed at the existing outlet and dams will need to be built along the west side of the TMA. Conceptually, it is estimated that three (3) dams will be required (two 25 m long dams and one 250 m long dam). These dams will have crest elevations of 350 masl and the outlet control dam will have an outlet elevation of 348 masl.

Additional civil and geotechnical engineering work will be required to advance the designs to determine required dam heights, appropriate dam construction method and materials, and the volumes of dam construction materials that will be required. This work will include conducting geotechnical drilling along the proposed dam alignments to determine the subsurface conditions and dam foundation requirements. A borrow source investigation should also be conducted to identify potential sources of fill for the construction of the dam.

20.19 EFFLUENT TREATMENT

Water from the TMA will be pumped to the ETP for treatment and either re-use as process water or discharged to the environment. On a conceptual level, the ETP has been sized for an average design flow rate of 7,855 m^3/day for a 1,000 tpd production rate. Based on the TMA water balance, this is the estimated rate (including a 20% factor of safety) that water will need to be pumped from the TMA over the 180 day operational period each year to keep the TMA from discharging untreated water. The plant will be designed to treat metals and ammonia.

The plant will be designed to precipitate metals from solution by chemical reaction with lime and ferric sulfate (ferric). Metals will be precipitated as metal hydroxide and adsorb on flocs. In the event that additional polishing treatment of other metals is required when degraded influent quality is encountered, the optimization of the lime/ferric sulfate injection will be performed or injection of a sulphur compound in the solution (such as sodium metabisulfite or SMBS) will be used to assist in the precipitation of those elements, as metal sulphides which display lower solubilites. The mine water and runoff from the waste rock will contain ammonia from blasting agents. Some ammonia will also be generated from the cyanide destruction process. Ammonia will be treated in the ETP using ion-selective resin in an ion exchange process.

20.20 SOCIAL AND COMMUNITY REQUIREMENTS

The development of the PLGP will provide employment and business opportunities for the regional communities and indigenous people located near the Project in central Manitoba.

It is estimated that the PLGP will employ up to two hundred sixty-five (265) people depending on the mine phase (construction, operations, or reclamation). During construction, approximately seventy-five (75) workers (not including contractors – estimated at 200) will be required, with up to 50% of the workforce coming from the local region.

During operations, average personnel requirements are estimated at one hundred twenty (120) people per year, including sixty (60) people on site at any one time, the remainder being rostered out on rotation. During the operations phase, up to 50% of the workforce could be from the local region.

In addition to the employment and business benefits, Minnova's Social Responsibility Statement will guide management, operations personnel, and all contractors to operate within the statement's guiding principles.

A new mine development can bring increased economic activity, create local jobs, develop much-needed infrastructure, and provide tax revenue that helps support local communities in delivering services.

Minnova has continued its dialogue about the status of the PLGP with First Nations and local communities and businesses.

AECOM (2014) reports that the Mathias Colomb Cree Nation (MCCN), located approximately 73 km north of Sherridon at the community of Pukatawagan, is the closest First Nation community to the Project area. Pukatawagan, which had a population of 1,826 in 2011 (1,478 in 2006), is accessible year-round by air and by rail, and by winter road for a period of approximately three (3) months of the year, depending upon conditions.

AECOM (2014) also reports that the Metis communities of Sherridon/Cold Lake had a population of one hundred thirteen (113) people in 2001 according to census information. Sherridon is located about 156 km north of The Pas adjacent to the First Nations-owned Keewatin Railway Company (KRC) rail line that extends from Sherritt Junction to Pukatawagan. The rail line crosses the mine access road about 6 km from the mill. Sherridon is an incorporated community under the Province of Manitoba's Northern Affairs Act and is administered by a mayor and council. The Cold Lake settlement is located approximately 1.5 km west of Sherridon.

Other First Nations, located within a similar distance to the site, include the Opaskwayak Cree Nation at Opaskwayak (86 km); the Nisichawayasihk Cree Nation at Nelson House (150 km); the Mosakahiken Cree Nation at Moose Lake (150 km); the Cross Lake First Nation at Cross Lake (195 km); and the Norway House Cree Nation at Norway House (240 km) (AECOM, 2014).

Additional stakeholder consultation will be conducted, as more details are developed regarding mine development.

20.21 CLOSURE PLAN

Mine closure will involve removing mine infrastructure, including the mill building, underground mining systems, and administration and support buildings. The TMF will be managed by maintaining a water cover on the tailings. The ETP will remain in place and operational until contact water quality from the TMF and waste rock is acceptable for direct discharge without treatment. A detailed closure plan will be developed prior to the site development.

The greenhouse gas emissions generated during the mining and closure of the Proposed Alteration is considered to have a negligible effect on climate change. It is anticipated that the reduction in CO_2 uptake by plants, due to clearing, will be minor during the mining of the open pits in the site. Following closure, the residual effect of vegetation loss on GHG emissions is anticipated to be negligible.

20.22 SUMMARY OF POTENTIAL IMPACTS ON THE ENVIRONMENT

The key conclusions of the various studies contributing to the environmental effects assessment are summarized in Table 20.16.

Table 20.16: Conclusions of the Environmental and Social Impact Assessment		
Component	**Anticipated Impact**	**Key Conclusions**
Topography	Minor	Re-starting the operation as an underground mine is not expected to have a new impact on site topography as the site foot print will not change. The proposed addition of open pits represents the most substantial long-term alteration to topography as it is today. The impact of open pits (Proposed Alteration) will be mitigated by the sequential backfill and remediation. Further, their final configuration will represent only a minor change in topography in context with the area of the mine and the region in which it is found. Overall, given the mitigation measures in place, the natural terrain of the area and region and the residual areal extent of disturbance, the residual effect on topography following closure is considered minor in the area and region.
Soil	Negligible	Given the implementation of mitigation measures, the nature of the waste rock and the short length of time the waste rock and pit walls will be exposed to weathering, it is anticipated that effects to soil quality because of ARD/ML will be negligible. Soil disturbance will be limited in extent and duration to the practical extent possible and is anticipated to result in a minor to negligible residual effect during the mining phase and a negligible residual effect following closure. Waste management strategies and policies employed and disposal of any wastes at licensed facilities will result in a negligible effect on soil. Progressive rehabilitation of the site will include remediating potentially contaminated soils, contouring, applying overburden and topsoil, and re-vegetating the developed areas and as a result, the residual effect on soil quality and quantity will be negligible following closure.
Air	Minor to Negligible	The residual effect of dust generation on air quality is anticipated to be minor to negligible in the area during mining activities. The residual effect of ARD/ML dust on air quality is anticipated to be minor to negligible at the site. Although the increase in traffic associated with the Proposed Alteration is considered major, the increase in emissions due to the increase in vehicles is anticipated to have a minor to negligible effect on air quality in the area.
Noise and Vibration	Minor to Negligible	All activities performed on the PL Mine site will be carried out in accordance with the Provincial Workplace Safety and Health Act and pit contractor's health and safety plans, which will minimize potential effects on humans due to noise. Noise levels are anticipated to return to existing ambient levels within the region. Based on the implementation of the proposed mitigation measures and the distance to human receptors, it is anticipated that noise and vibration effects on humans will be minor to negligible during the mining phase of the Proposed Alteration.
Climate	Negligible	

Table 20.16: Conclusions of the Environmental and Social Impact Assessment		
Component	**Anticipated Impact**	**Key Conclusions**
Groundwater	Negligible	No registered groundwater users have been identified within 9.6 km of the site. Any effects on shallow groundwater quality and quantity are anticipated to be limited in spatial extent to the site and immediate area. The residual effect of pit dewatering on shallow groundwater quantity is anticipated to be minor during mining. The hydraulic conductivity of the bedrock is relatively low and the zone of influence to pit/mine dewatering is expected to be minimal; however pumping tests would be required to confirm this expectation. Groundwater levels are anticipated to return to pre-mining conditions within months to several years, and given the implementation of appropriate mitigation measures and the progressive pit rehabilitation, it is anticipated that the residual effect on groundwater quantity and quality following closure will be negligible.
Surface Water	Minor to Negligible	Contact water from tailings and waste rock will be managed in the TMA and will be treated before discharge to the environment. This treatment system will be designed to remove metals and ammonia (from blasting). Discharge from the ETP to the environment will be monitored and must meet specific effluent limits that will be applied during the permitting process for the ETP.
Protected and Other Aquatic Resources	Minor to Negligible	The mitigation measures recommended for the protection of surface water are anticipated to sufficiently mitigate potential surface water effects and will prevent adverse effects on aquatic resources. Short Jaw Cisco will not be affected by the Proposed Alteration. The aquatic habitat value of Fire Pond is categorized as Marginal, as it provides habitat only for small-bodied fish (*i.e.*, Brook Stickleback) and there is no connectivity to other waterbodies that prevents it from forming part of or supporting commercial, recreational, or Aboriginal fisheries. Given that the type of habitat potentially lost (*i.e.*, slow moving, unconnected boggy areas including Fire Pond) is readily available throughout the area and that the species using this type of habitat (Brook Stickleback) are widespread and abundant, the residual effect to aquatic resources within the area is minor to negligible.

Table 20.16: Conclusions of the Environmental and Social Impact Assessment		
Component	**Anticipated Impact**	**Key Conclusions**
Protected and Other Flora Species	Minor to Negligible	Although the Proposed Alteration will result in a loss of vegetation in the site, no unique vegetation communities will be lost as confirmed through a terrestrial survey. In addition, the actual extent of disturbance is expected to be a fraction of the site and the roads, stockpiles, and open pits will be progressively rehabilitated, which will further minimize the duration and extent of the potential effect. Overall, the residual effect of vegetation loss during mining and closure phase is considered minor to negligible in the site. No protected flora species are known to occur within the region and as a result, no effects on protected species are anticipated during the mining or closure of the underground mining operation or the Proposed Alteration.
Protected and Other Fauna Species	Negligible	No habitat of specific or critical value to wildlife was observed at the site (such as calving or over-wintering areas) and, based on site conditions and limited field observations, it is expected that there is no critical wildlife value in the area. A large portion of the area was affected by historical development at the PL Mine site or severe forest fires in 1989, and as a result, the quality of habitat available in the area is reduced. Although the Proposed Alteration will result in a loss of wildlife habitat (through vegetation loss) at the site, the type of growing conditions that has been lost is not uncommon in the area and region. As a result, the residual effect of habitat loss on flora during mining is considered minor. Wildlife abundance in the region is anticipated to be low, based on past and recent surveys as well as the reduced habitat value, as a result of forest fires. Given the mitigation measure implemented and the reasons above, the potential noise effects on wildlife during mining of the open pits is anticipated to be moderate within the site and minor within the area. During mining and closure, light pollution is anticipated to result in a negligible residual effect on fauna. Although the increase in traffic on the main access road and the Sherridon Road are anticipated to be major, it is anticipated that road speed limits on the site, given the low abundance of wildlife species, will result in a minor residual effect of collisions on wildlife. It is anticipated that potential effects of noise and light pollution on fauna following closure are anticipated to be negligible to minor. The Kississing-Naaosap Boreal Woodland Caribou herd, whose snow-free season range overlaps with the site, is composed of an estimated 150 individuals and is currently considered stable (COSEWIC 2002). According to Manitoba's Conservation and Recovery Strategy for Boreal Woodland Caribou (Government of Manitoba 2005), the conservation risk of the Kississing and Naaosap herds are considered to be high risk and medium risk, respectively. Boreal Woodland Caribou have typically avoided young forest or disturbed areas, including those areas impacted by the 1989 forest fires. As a result of the lack of suitable habitat and historical and recent presence of activity, the residual effects of noise, light pollution, collisions, and habitat loss on Boreal Woodland Caribou is anticipated to be minor to negligible during mining and following closure, the residual effects are anticipated to be negligible.
Resource Use	N/A	Minnova has indicated that they are committed to working with local trappers and interested stakeholders to ensure access to trap lines and

Table 20.16: Conclusions of the Environmental and Social Impact Assessment		
Component	**Anticipated Impact**	**Key Conclusions**
		other resource harvesting is not impacted by the Proposed Alteration.
Heritage Resources	Negligible	There are no known or potential historic or heritage resources at the PL Mine site. Therefore, the residual effect on heritage resources is anticipated to be negligible during mining and closure of the Proposed Alteration.
Aesthetics	Negligible	Based on the mine's remote location, surrounding vegetation, and historical disturbances associated with the 1989 forest fires or previous development at the PL Mine site, effects on aesthetics during the mining and closure phase are anticipated to be negligible.

20.23 RECOMMENDATIONS

Groundwater Monitoring Program – Install monitoring wells in vicinity of the TMA and waste rock storage area. It is estimated that approximately twenty (20) monitoring wells should be installed around the TMA and ten (10) wells should be installed around the waste rock area. These wells should be sampled for metals and general chemistry parameters and each well should be subjected to single well response testing to determine hydraulic conductivities. Single well response testing should be conducted on each well to improve the confidence level in the estimated bulk hydraulic conductivity of the rock. These data should then be used to predict groundwater seepage rates into the open pit and underground workings.

Surface Water Monitoring Program and Hydrological Modeling – Establish surface water monitoring stations around the TMA and downgradient from the waste rock storage area. These stations should be sampled for metals and general chemistry parameters and flows should be measured at time of sampling. A permanent flow station should be established at the outlet of the TMA. Prepare a detailed hydrological model for the site to refine the water balance.

Air Emissions Study – A study should be conducted to determine levels of particulates and other emissions that could be generated from the proposed mining activities. This study should consider the types of equipment that will be present at the site and particulates generated from the milling and tailings deposition processes.

Noise Study – A study should be conducted to determine the noise levels that will be generated from the proposed mine development. This study should consider the types of equipment that will be used and should model impacts from drilling, blasting, milling, and other equipment usage.

Effluent Treatment – Conduct effluent water quality modeling using the available groundwater data and tailings and waste rock static and kinetic testing results. Geochemical modeling should be conducted to determine the quality of water that will be created by combining TMA runoff, waste rock runoff, mine water, and process water. The results of the modeling should be used to prepare a design for the effluent treatment plant. This plant should be tested at a pilot scale to confirm the appropriate treatment technology.

TMA Design – Conduct geotechnical testing and prepare detailed design for the TMA (including dams and outflow structure). Drilling should be conducted along the proposed dam alignments and at the proposed location of the outflow structure.

TMF Management – Complete a detailed tailings management operation and deposition plan.
Waste Rock Storage Design – Conduct geotechnical testing and prepare detailed design for the waste rock storage area.

Aquatic and Terrestrial Study Updates – Updated studies of aquatic and terrestrial habitats in the vicinity of the Project area should be conducted. These studies should establish baseline aquatic and terrestrial conditions.

Kinetic Testing of Tailings and Waste Rock – Representative samples of tailings and waste rock should be subjected to kinetic testing (humidity cells) to determine the long-term potential for these materials to generate acid mine drainage.

Closure Plan – A detailed closure plan should be prepared outlining how the site will be decommissioned at site closure.

21 Capital and Operating Costs

21.1 CAPITAL EXPENDITURES

21.1.1 Basis for Estimates

The capital expenditures estimate includes the following:

- Mine development, mining equipment mobile (leased), and fixed and associated consumables and maintenance parts for development and infrastructure;
- Project infrastructure equipment and materials;
- Construction materials;
- Labour;
- Temporary buildings and services;
- Construction support services;
- Spare parts;
- Initial fills (inventory);
- Freight;
- Vendor supervision;
- Owner's cost;
- Engineering, Procurement and Construction Management;
- Commissioning and start up; and
- Contingency.

21.1.1.1 Direct Costs

Direct costs are all costs associated with permanent facilities. This includes mine development openings, equipment and material costs, as well as construction and installation costs.

Mine infrastructure costs for facilities, such as maintenance shops, mine dewatering, refuge stations, etc., were developed based on the conceptual plans and general arrangements presented earlier. Wherever possible, equipment and material quotes and contractor installation costs were used.

Other major equipment expenditure estimates are based on quotes obtained from suppliers and installation costs estimated as part of this study.

During the pre-production and sustaining development periods, all materials and equipment pricing are based on quotes obtained from local Canadian or international suppliers, where Canadian suppliers do not exist.

All major equipment expenditures include freight only. Applicable taxes and duties have not been included in the capital expenditure estimates.

Where possible, direct costs are based on actual takeoffs:

- Earthwork/site work;
- Concrete;
- Structural steel;
- Buildings and architectural;
- Electrical;
- Instrumentation and controls; and
- Piping.

Commodity pricing for earthwork, concrete, steel, architectural, and piping were based on local, Manitoba and Canadian costs (in some cases escalated to 2017 costs from past costs). Labour rates and equipment usage rates used throughout the estimate were provided by the same source as the commodity prices.

It was assumed that rock required for site preparation will be provided, at no cost, during the pre-production stage. Only costs for placement have been allowed for in estimates.

Labour rates generally reflect Manitoba and Canadian levels for the types of work performed and in some cases adjusted for locally applied rates. The mine labour costs are based on four types of estimates:

- Owner/Operator expected labour rates;
- Contractor budget prices for undertaking the tasks associated with constructing a specific installation;
- Average industry rates a contractor will be expected to charge for performing specific tasks; and
- Lateral and raise development rates, developed and based on expected productivity and labour, materials, and equipment costs for such an underground development program.

All labour costs include government mandated contributions and the costs for company provided benefits.

21.1.1.2 Indirect Costs Estimate

The indirect costs cover all the costs associated with temporary construction facilities and services, construction support, freight, vendor representatives, spare parts, initial fills and inventory, Owner's costs, EPCM, commissioning, and start-up assistance.

The costs for construction facilities include all temporary facilities, services and operation, site office operations, security buildings and services, construction warehousing and material management, construction power and utilities, site transportation, medical facilities and services, garbage collection and disposal, and surveying.

Spare Parts: The cost for spare parts is factored based on equipment costs where the vendors did not provide cost for spares needed for the first year of operation.

Initial Fills (Inventory): The estimated cost for initial fills is based on 3-months of operating requirements. Present Manitoba purchasing costs were used for pricing.

Freight: The freight costs were either provided by the vendor or estimated based on weights and typically include containerised and break-bulk shipping, and each are respectively divided into ocean freight and inland freight. For imported equipment, the cost of freight and export packing, ex-works to the nearest port, is included with the cost of the equipment. Freight insurance is included in the Owner's cost.

Vendor Representatives: The requirement for the vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment, as deemed necessary for equipment guarantees or warranties, has been included in the estimate. Typically, the cost for this item is inclusive of salary and travel.

Taxes and Duties: Taxes and duties have been excluded.

Engineering, Procurement, and Construction Management (EPCM): EPCM has been calculated based on the Minnova project managing development and construction, and using consultants where deemed appropriate by Minnova.

Capital Cost Qualifications and Exclusions: All surface construction work will be executed by contractors.

Capital expenditures estimates exclude:

- Sunk costs;
- Taxes and duties;
- Deferred capital;
- Financing and interest during construction;
- Additional exploration drilling;
- Escalation;
- Corporate withholding taxes;
- Legal costs;
- Metallurgical testing costs; and
- Condemnation testing.

All expenditure estimates are in 2017 constant US Dollars. Exchange rates used are:

- US$0.80:C$1.00

21.1.2 Mining

21.1.3 Underground Mining

Underground capital cost estimates are based on quote pricing from suppliers, consultants, and contractors, provided with reasonable detail specifications to ensure equipment or service provided is specific to the Project and includes all costs specific to the Project and application. Some small equipment and facilities component costs were factored based on norms for the type of facility being constructed and adjusted to reflect local conditions.

Construction and installation labour rates are based on Owner/Operator costs for the types of work envisaged for the Project.

Most mobile mining equipment is leased by Minnova. Alimaks, slushers, and some smaller mobile equipment will be purchased by Minnova.

The mine pre-production capital expenditures are estimated to total $12.8 million including a 10% contingency. The breakdown of the mine capital expenditures is presented in Table 21.1.

Table 21.1: Breakdown of Mine Capital Expenditures

Component	Total Expenditures ($ millions)
Mine Development	5.2
Underground Infrastructure	0.3
Surface Infrastructure	2.7
Mine Services	0.9
Mining Equipment	2.5
Contingency	1.2
Total Mine Expenditures	**$12.8**

The initial capital expenditure for the underground mine will include the development of an access decline from the existing portal down to an elevation of 145 m below the surface. From the ramp, production levels will be established on the 40 m and 75 m Levels. Stopes will be developed for production mining with the excavation of Alimak raises in ore from one level up to the overlying level.

Mine development will begin with the dewatering of the existing workings to a depth of 40 m below the surface. The new decline will branch northwards from the existing decline at a 30 m depth and continue downwards in a spiral designed for minimal access distance at the chosen level depths.

Five stopes will be developed to enable production at a commercial volume of mined ore. The mine development will also include the development of a ventilation raise, installation of mine fans and heaters, installation of a pumping system and reticulation systems for electricity, communications network, compressed air, process water, and mine drainage water.

The pre-production period for the PL Mine is expected to be 18-months and will be conducted simultaneously with plant refurbishment, infrastructure investments, and mine site preparatory work.

The mine development capital expenditures and underground mine infrastructure capital expenditure estimates are shown in Table 21.2 and Table 21.3, respectively. Capital Expenditures for equipment purchases and leasing total $2.5 million dollars including a 10% contingency.

Table 21.2: Mine Development Capital Expenditures Estimates

Heading	Quantity	Units	Unit Rate Pre-Production ($)	Year -1
Existing Development Rehabilitation				
Ramp	1,000	metres	$3,798	$760,000
Levels	400	metres	$3,798	$0
Lateral Development				
Spiral Ramp Surface to 145 Level	1,160	metres	$4,357	$2,527,000
Spiral Ramp 145 to 530	1,680	metres	$4,357	$0
40 Level	940	metres	$4,243	$424,000
75 Level	730	metres	$4,243	$424,000
110 Level	730	metres	$4,243	$0
145 Level	1,110	metres	$4,243	$0
180 Level	1,130	metres	$4,243	$0
215 Level	900	metres	$4,243	$0
250 Level	800	metres	$4,243	$0
285 Level	200	metres	$4,243	$0
320 Level	200	metres	$4,243	$0
355 Level	200	metres	$4,243	$0
Intake Ventilation Connection Drifts	300	metres	$4,243	$0
Exhaust Ventilation Connection Drifts	300	metres	$4,243	$0
Pre-Production Development - Ore				
75 Level	200	metres	$2,100	$420,000
40 Level	100	metres	$4,243	$424,000
Raise Development				
Intake Raise 1 - 145 to Surface	150	metres	$2,800	$196,000
Intake Raise 1 - 495 to 145	210	metres	$2,800	$0
Exhaust Raise 1 - 145 to Surface	150	metres	$2,000	$0
Exhaust Raise 1 - 495 to 145	210	metres	$2,000	$0
Boreholes				
Backfill Boreholes	360	metres	$250	$0
Drainholes	360	metres	$250	$18,000
Total Existing Development Rehab	**1,400**	**metres**		**$760,000**
Total Lateral Development	**10,680**	**metres**		**$4,219,000**
Total Raise Development	**720**	**metres**		**$196,000**
Total Boreholes	**720**	**metres**		**$18,000**
Contingency	**10%**			**$519,000**
Total Underground Development Expenditures				**$5,712,000**

Table 21.3: Underground Mine Infrastructure Capital Expenditure Estimates

Component	Total Cost ($)
SURFACE INFRASTRUCTURE	
Mine Dewatering Equipment	$97,000
Surface Intake Vent Fan Installation	$581,000
Exhaust Ventilation Fans Installations	$441,000
Underground Service Water Storage Pond	$0
Underground Water Treatment Plant	$200,000
Portal Rehabilitation	$97,000
Explosives Magazines (Supplier Provided)	$127,000
Mine Rescue/Backfill/Shop Building	$185,000
Compressor/Storage Building	$93,000
Compressors	$340,000
Mine Rescue Equipping	$316,000
Development Crew Gear Equipping	$201,000
Mine Lamps and Lamproom	$24,000
Total Surface Infrastructure	**$2,702,000**
UNDERGROUND SUPPORT SERVICES FACILITIES	
Fuelling Station (Marcotte)	$73,000
Breakdown Shop Equipping	$124,000
Explosives & Detonators Magazine Construction & Equipping	$50,000
Refuge Station Construction & Equipping	
Portable Toilet	$10,000
Total Underground Support Services Facilities	**$257,000**
MINE SERVICES	
Portable Substations	$278,000
Mine Communication	$231,000
Backfill Distribution System	$16,000
Underground Booster Fans & Auxilliary Ventilation	$118,000
Electrical Tools	$43,000
Computers, Peripherals & Software	$108,000
Engineering & Geology Equipment	$80,000
Mine Department Office Equipment & Small Tools	$2,000
Total Mine Services	**$876,000**
Contingency	**$767,000**
Total Mine Infrastructure Expenditures	**$4,602,000**

21.1.4 Sustaining Capital

Underground mine development continues after the mill is in full production and categorized as sustaining capital. Sustaining capital is presented in a separate Section 21.1.11.

Mine capital expenditures do not include:

- Mine office buildings or shop facilities – They are included in Section 21.1.7 of this report.
- Mine engineering equipment (computers, survey equipment etc.) – They are included in Section 21.1.7 of this report.
- Mobile equipment that is not required by the mine (*i.e.*, no mobile units for the plant).
- Infrastructure or process plant related costs.
- Mine above ground communication network and system – They are included in Section 21.1.7 of this report.

21.1.5 OPEN PIT MINING

Open pit mining will use a mining contractor. Quoted expected mining costs for the equipment, consumables and labour provided by the contractor are \$11.00-\$15.00 per cubic metre or \$4.00-\$5.40 per tonne of rock moved.

21.1.6 PROCESSING PLANT

The PL Project benefits from utilizing the concentrating mill that was constructed for the Puffy Lake Mine in the late 1980s. This significantly reduces the construction and equipment capital expenditures required.

Capital expenditures are based on the following:

- Contractor labour and materials costs for construction related to civils, structural architectural, etc. for installation of new equipment.
- Contractor labour, equipment and materials costs for refurbishing existing and installing new equipment and other fixed infrastructure (wiring, MCCs, etc.).
- Labour hour estimates for refurbishing existing facilities and equipment.

The total capital expenditures required to refurbish and install new processing equipment and associated civil, structural, mechanical, piping, and electrical components and facilities, and for re-commissioning the TMF is approximately \$8.5 million. A breakdown of the processing plant and TMF capital expenditures is presented in Table 21.4. New and replacement equipment for the plant totals approximately \$3.7 million, the TMF is \$1 million, and the remainder is labour and materials.

Table 21.4: Breakdown of the Processing Plant and TMF Capital Expenditures

Component	Total Cost ($)
Truck Dump	$11,000
Jaw Crusher	$94,000
Screen	$2,000
Cone Crusher	$69,000
Crushing Conveyors	$177,000
Crusher Building	$41,000
Fine Ore Bin	$94,000
Rod Mill	$558,000
Ball Mill	$301,000
Cyclones	$91,000
Gravity Circuit	$575,000
Flotation & Reagents	$615,000
Thickener	$161,000
Regrind	$86,000
Merill Crowe	$288,000
Leach	$64,000
Filtration	$957,000
Refinery	$471,000
Cyanide Destruction	$57,000
Electrical Cabling	$82,000
Large Motor Overhauls	$49,000
MCC Rooms	$200,000
Samplers, DCF and Communications	$552,000
Compressors	$7,000
Mill Building & Offices	$148,000
Office Equipment & Furnishings	$15,000
First Fills	$98,000
Assay Laboratory	$922,000
Indirects	$1,060,000
TAILINGS MANAGEMENT FACILITY	
Weir & Tailings Dam	$1,000,000
Contingency	$678,000
Total Processing Plant & TMF Expenditures	**$9,523,000**

21.1.7 Surface Infrastructure

Total pre-production capital expenditures for Project infrastructure and surface department are estimated to be approximately $5.8 million, including a 10% contingency. The breakdown of expenditures is presented in Table 21.5. Major expenditure components are for access road upgrading, power supply re-establishment, camp, shops equipping, water supply and treatment, and mobile equipment.

Table 21.5: Surface Infrastructure Capital Expenditures

Component	Total Cost ($)
Access Road	$750,000
Supply Powerline Upgrade	$507,000
On Property Main Powerline Rehabilitation	$341,000
Surface Substations	$380,000
Site Roads	$120,000
Mine Office/Dry	$250,000
Surface Shop & Warehouse Offices	$132,000
Shop Equipping	$358,000
Millwright Shop Equipping	$40,000
Office Furnishings	$30,000
Warehouse Equipping	$155,000
Camp	$533,000
Computers & Software	$123,000
Environmental Department Equipment	$50,000
Fuel Storage	$112,000
Propane Storage	$65,000
Potable Water Treatment Plant Mine	$127,000
Potable Water Treatment Plant Camp	$109,000
Fresh Water Pump & Pipeline	$185,000
Fire Fighting Equipment	$39,000
Fire Protection	$100,000
First Fills, Commissions, Vendor Reps	$75,000
Mobile Equipment	$720,000
Contingency	$550,000
Total Infrastructure Expenditures	**$5,851,000**

21.1.8 PROJECT INDIRECTS AND OWNER'S COSTS

Project in-directs and Owner's costs are estimated at $6.2 million over the 18-month pre-production period. A breakdown of the Owner's costs is presented in Table 21.6. Owner's costs also include manpower recruitment and training during the pre-production period and all equivalent G&A costs, which will be incurred during the construction phase. These costs also include a 10% contingency.

Table 21.6: Owner's Costs

Component	Total Cost ($)
Engineering Consultants	$525,000
Environmental	$20,000
Camp Operating Costs	$641,000
Roads and Yards Maintenance	$35,000
Power	$436,000
Water Treatment	$50,000
Mine Air Heating	$311,000
A-Z & Owner's Construction Team	$2,580,000
Office Supplies	$24,000
Surface Transportation	$126,000
Equipment Rentals	$24,000
Communications	$119,000
Road Servicing	$50,000
Surface Vehicles & Equipment Operation	$301,000
Travel & Expenses	$537,000
Saftey & First Aid	$10,000
PPE	$63,000
Non Mill Freight	$200,000
Courier/Postage	$10,000
Insurance	$100,000
Bank Fees	$8,000
Sub-Total Owner's Costs	**$6,170,000**

21.1.9 TOTAL CAPITAL EXPENDITURES

The estimated total Project pre-production capital expenditure, inclusive of contingencies and excluding working capital, is approximately $34.3 million. The total expenditures include EPCM, contractor overheads, and a 10% contingency on all estimated expenditures. A summary of Project pre-production capital expenditures is presented in Table 21.7. A working capital allowance of $1.1 million is estimated to be required.

Table 21.7: Project Pre-Production Capital Expenditures

CAPEX Cost Centre	Total Cost ($)	Year -2	Year -1	Total ($)
Mine	$11,492,000		$11,492,000	$11,492,000
Processing Plant	$7,845,000		$7,845,000	$7,845,000
Infrastructure	$5,301,000		$5,301,000	$5,301,000
Tailings Management Facility	$1,000,000		$1,000,000	$1,000,000
Owners Costs	$6,170,000		$6,170,000	$6,170,000
Contingency	$2,515,000		$2,515,000	$2,515,000
TOTAL CAPEX	**$34,323,000**		**$34,323,000**	**$34,323,000**

21.1.10 WORKING CAPITAL

A working capital allowance, in addition to capital expenditures, of $1.1 million has been included in the cash flow model. This represents approximately 1 month of operating costs, which will be incurred before the first revenue is realized. The working capital requirement is less than will normally be expected, as payment for mine product will be immediately after the concentrate has been shipped.

21.1.11 SUSTAINING CAPITAL

Sustaining capital estimates of $53.7million comprise ongoing mine development. The sustaining capital expenditures for the mine are shown in Table 21.8.

Table 21.8: Sustaining Capital Expenditures

		Year										Total Expenditures ($)
		1	2	3	4	5	6	7	8	9	10	
Mine Development		$7,751,000	$9,894,000	$11,673,000	$13,117,000	$0	$0	$0	$0	$0	$0	**$42,435,000**
Mining Equipment Rebuilds												**$0**
Mine U/G Infrastructure												**$0**
Refuge Station		$137,000		$137,000								**$274,000**
Explosives Magazines		$50,000		$50,000								**$100,000**
Fuel Station												**$0**
Main Dewatering Sumps			$110,000									**$110,000**
Main Storage Area		$36,000		$36,000								**$72,000**
Lavatories		$20,000	$20,000									**$40,000**
UG Information System		$100,000	$75,000	$75,000	$50,000							**$300,000**
Sub-Total Mine U/G Infrastructure		**$343,000**	**$205,000**	**$298,000**	**$50,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$896,000**
Mine Surface Infrastructure												
Ventilation Fans												**$0**
Backfill Plant		$1,051,000										**$1,051,000**
Sub-Total Mine Surface Infrastructure		**$1,051,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,051,000**
Processing Plant												**$0**
Tailings Management Facility		$1,935,000	$1,000,000	$250,000								**$3,185,000**
Surface Infrastructure												
Access Road		$989,000										**$989,000**
Permanent Camp												**$0**
Office Building		$250,000										**$250,000**
Computers and Software												**$0**
Sub-Total Surface Infrastructure		**$1,239,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,239,000**
Surface Equipment												**$0**
Owner's Costs		$0										**$0**
Contingency	10%	$1,232,000	$1,110,000	$1,222,000	$1,317,000	$0	$0	$0	$0	$0	$0	**$4,881,000**
Total Sustaining Capital Expenditures		**$13,551,000**	**$12,209,000**	**$13,443,000**	**$14,484,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$53,687,000**

Mine sustaining capital expenditures comprise definition diamond drilling, waste development for stoping, and additional infrastructure (refuge stations, explosives magazines, etc.).

Surface infrastructure sustain capital is associated with further access road and offices upgrades, and the TMF capacity increases.

21.1.12 CLOSURE COSTS

Closure costs have been estimated at $1.5 million at the end of the Project life and is included as a separate line item in the cash flow model.

21.2 OPERATING COST ESTIMATES

21.2.1 Basis for Estimates

Project operating costs, by department, are based on efficiencies and productivities generally achievable in Canada.

Project departmental operating costs were divided into two components – consumables/maintenance parts and labour. The consumables component includes all materials and parts needed for mining, processing, and surface facilities and the operation and maintenance of equipment for these areas. Costs for consumables were obtained from Canadian suppliers. Maintenance parts and consumables are based on Canadian and international equipment suppliers. The total mine labour force complement and salaries were calculated on a total yearly basis. The labour component was combined with the materials component to produce the yearly departmental operating cost estimates.

The G&A cost components include the materials and supplies used by the administration and surface services groups. These costs comprise office supplies, computer supplies, and computer and software upgrades, light vehicle, and surface equipment operating and maintenance consumables, crew rotation charter flights to Winnipeg, camp accommodation and catering operational costs, business travel inside Canada, fees for consultants, and communications costs.

Labour costs and salaries for all services labour and mine staff have been estimated on a yearly total cost basis.

Critical operating cost components are based on the following costs:

- The diesel fuel price of $1.10 per litre; and
- The electrical power cost of $0.05 per kWh.

Labour costs for the operating period are based on the manpower schedules presented for each department and the associated labour costs. The costs include a burden component of 35-40%. Labour rates are based on local rates where available and/or contractor costs in the region and country, for similar types of work. The rates used include all cost and profit components payable to contractors.

All costs are quoted in constant 2017 Canadian Dollars.

21.2.2 Mining

Individual costs for underground mining have been estimated for manpower, equipment operating, maintenance, and materials consumptions from first principles. The total underground mining cost is estimated to be $111.68 per tonne of ore (presented in Table 21.9). The underground mine operating costs have a 10% contingency included in the estimates.

Table 21.9: Mining Costs (US$)	
Component	**Cost ($/t)**
Delineation Drilling	4.63
Direct Stoping	47.86
Non-Direct Stoping	43.29
Equipment Leasing	12.89
Trucking Ore to Processing Plant	3.01
Total Mining Cost	**$111.68**

Mines services and overheads costs include all other non-direct stoping costs for the PL Mine. Mine services operating costs are associated with maintaining underground facilities and services (power, water supply, etc.), operating and maintaining ventilations fans, supplies for safety and training, including personal protective equipment and mine rescue, and operating and maintaining all support mobile and track haulage equipment used in the mine.

The open pit mining costs are based on costs provided by local quarrying and open pit mining contractors, who would provide all equipment and manpower. The expected contractor mining costs are $9.38 per tonne of ore and $8.13 per tonne of waste.

21.2.3 Processing Plant and Tailings Management

The total processing plant and TMF operating costs are based on a combination of first principles and past reagents and consumables consumption (as guidelines). The forecast operating costs is expected to be $24.37 per tonne of ore. A breakdown of the processing costs is presented in Table 21.10.

Table 21.10: Breakdown of the Processing Costs	
Component	**Cost ($/t)**
Operating Labour	9.65
Power	2.44
Reagents	4.22
Operating Supplies	1.08
Maintenance Labour	3.62
Maintenance Supplies	2.12
Total	**$23.13**
Contingency	$1.24
Total Cost	**$24.37**

21.2.4 General and Administration (G&A) Costs

The estimates for G&A costs encompass all operating costs associated with operating the camp only. The costs associated with operating the underground mine offices and providing materials and supplies for staff functions are included in the mine general expenses component of mining costs. The total yearly G&A costs including Surface Services Department are estimated to be approximately $8.9 million with G&A costs comprising $7.5 million of the total. Table 21.11 provides a breakdown of the major cost centres for G&A and surface services. Major costs are associated with G&A manpower, operations management, camp operations, crew rotation air travel, power, insurance, communications and freight.

Table 21.11: Breakdown of Major Cost Centres for G&A and Surface Services	
Component	**Total Cost ($/year '000)**
Manpower Remuneration	4,237
Camp Operation	1,445
Personnel Rotation Flights	537
Communications	115
Insurance	300
Power	263
Freight	100
Professional Services	120
Fees and Licences	25
Surface Vehicles	60
Safety, Training, and Human Resources	97
Other Costs	201
Total G&A Costs	**$7,500**

21.2.5 DORÉ TRANSPORT AND REFINING CHARGES

Transport and refining costs of $5.00 per ounce for gold have been included in the cash flow model and are based on Canadian norms.

21.2.6 PROJECT TOTAL OPERATING COSTS

The estimated total average operating cost (excluding smelting and refining) for the PL Mine is approximately $194.08 per tonne ore and $163.44 for per tonne ore for underground and open pit mining, respectively. Table 21.12 presents a summary table of life-of-mine average operating costs for each department on a cost per tonne of ore basis.

Table 21.12: Project Operating Costs Summary		
Department	**Underground Cost ($/t Ore Mined)**	**Open Pit Cost ($/t Ore Mined)**
Mining	111.68	81.04
Processing and Tailings	24.37	24.37
Surface Department, Environmental, and G&A	49.12	49.12
Royalty (2%)	8.91	8.91
Total	**$194.08**	**$163.44**

21.3 EXCLUSIONS

For the purpose of this study, value added taxes and other taxes, along with import duty costs, have not been included. Exploration costs and all costs associated with areas beyond the property limits have also not been included.

22 Economic Analysis

The expected cash flow estimates are calculated using the forecast mine plan, operating costs, and capital expenditures incorporating expected long-term metal prices based on the past 36-months' moving average prices for each metal, as follows:

Gold (US$/oz.) $1,250

The discounted cash flow analysis uses 2017 Constant Canadian Dollar values.

A summary of the expected parameters used for the financial analysis is presented in Table 22.1.

Table 22.1: Expected Project Parameters	
Reserves – Underground Reserves – Open Pit	952,727 tonnes at a grade of 7.00 g Au/t 313,711 tonnes at a grade of 4.35 g Au/t
Estimated Mining Dilution: Underground Open Pit	 13% at 0% grade 20% at 0% grade
Projected Mining Recovery	91%
Payable Gold Produced	47,000 to 55,000
Pre-Production Capital Expenditures	$34.3 million
Working Capital	$1.1 million
Total Sustaining Capital Expenditures	$53.7 million
Closure Cost	$1.5 million
Estimated Operating Costs ($/tonne): Mining – Underground Mining – Open Pit Processing G&A Royalty (2%)	 $111.68 $81.04 $24.37 $49.12 $8.91
Life-of-Mine	4.4 years

Processing plant recoveries and concentrate grades are varied by year within the model based on the precious metals feed grades to the processing plant.

Revenue is based on payables terms and smelter and refining costs for a third party refiner (*e.g.*, Canadian Mint, private refinery).

Operating costs, as presented in the operating costs section, are incorporated.

Capital expenditures, as shown in the capital section, would be incurred over approximately a 1.3-year period, which is reflected in the discounted cash flow calculations. The cash flows include sustaining capital.

An NSR royalty of 3% payable on net revenue is included.

Costs for metal sales and shipping are included in the deductions that the refiner makes.

US dollar denominated components are converted using C$:US$ exchange rate of 1:1.25.

The cash flow analysis excludes:

- Any element or impact of financing arrangements; and
- All exploration and acquisition costs incurred prior to the production decision.

22.1 TAXES

In calculating the after tax returns, the Project is subject to Federal and Manitoba Corporate Income Tax and Manitoba Mining Tax.

Federal and provincial Corporate Taxes use a series of capital pools Canadian Exploration Expenses (CEE), Canadian Development Expenses (CDE), and Class 41 a, b, and 2 for the Project capital investment. These pools can be deducted, at varying rates, from operating profit, to ensure the Project investment is fully paid back before taxes on profits are levied.

Accelerated Capital Cost Allowance (ACCA) is being eliminated, but the yearly amounts that can be claimed of total eligible capital expenditures is being reduced and claimed as follows:

- 2018 80%
- 2019 60%
- 2020 30%
- 2021 0%

Loss carry forwards from previous years and administration costs, until the Project enters production, will be carried and deducted in the taxation calculations.

Nominal Corporate tax rates are 15% and 12% federal and provincial, respectively, on taxable profits after all deductions.

22.1.1 MANITOBA MINING TAX

Manitoba Mining Tax is deductible for Federal Corporate taxation calculations.

Mining taxation allows limited deductions from net profit including:

- Capital cost allowances for mining assets (at 30%) and processing assets (at 15%) on a declining balance basis; and
- Processing allowance on processing assets.

The nominal Manitoba Mining Tax rate after deductions is 10%.

22.1.2 Basis of Taxation Estimates

In calculating the respective taxes payable, the following basis is used:

- No interest expenses included;
- Stock based compensation not included;
- Deferred tax implications not included; and
- Closure costs included.

Deductions have been made to pools to ensure minimal loss of carry forward values and to minimize the balance (if any) in pools at the end of the Project life.

22.2 FINANCIAL RETURNS

The overall level of accuracy of this study is approximately ±10%.

The Project's expected investment and returns, based on the expected cash flow parameters, are shown in Table 22.2.

Table 22.2: Expected Project Returns		
	Pre-Tax	**After-Tax**
Undiscounted Net Revenue	$376 million	$376 million
Undiscounted Total Cash Flow	$71 million	$47 million
NPV (5%)	$55.9 million	$36.7million
NPV (10%)	$44.3 million	$28.8million
IRR	65%	53%
Payback Period	1.2 years	1.2 years

Results indicate that at the expected parameters and metals prices, the Project is viable.

22.3 SENSITIVITY ANALYSIS

Several sensitivities on the Base Case scenario have been investigated to determine the effect on the key financial statistics, if increases and decreases of 5, 10, 15, and 20% occur to the following parameters:

- Mined Grade'
- Gold Price;
- Operating Cost; and
- Capital Expenditures.

The results of the sensitivity analysis are presented in Table 22.3 and Table 22.4.

Table 22.3: Sensitivity Analysis for After-Tax NPV									
Parameter	**After-Tax $NPV_{5\%}$ ($M)**								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade	-$24.8	-$9.2	$6.3	$21.7	$36.7	$51.6	$66.4	$81.3	$96.1
Gold Price	-$25.0	-$9.4	$6.2	$21.6	$36.7	$51.6	$66.5	$81.4	$96.3
Operating Costs	$77.3	$67.2	$57.0	$46.8	$36.7	$26.5	$16.2	$5.8	-$4.7
Capital Costs	$50.2	$46.8	$43.4	$40.0	$36.7	$33.4	$30.1	$26.8	$23.5

Table 22.4: Sensitivity Analysis for After-Tax IRR									
Parameter	**After-Tax IRR (%)**								
	-20%	**-15%**	**-10%**	**-5%**	**0%**	**5%**	**10%**	**15%**	**20%**
Mined Grade		2	22	39	53	68	81	94	107
Gold Price			16	37	53	65	76	87	98
Operating Costs	108	97	84	70	53	34	6		
Capital Costs	84	75	67	60	53	48	42	37	32

Figure 22.1 and Figure 22.2 show the sensitivity analysis in graphical form.

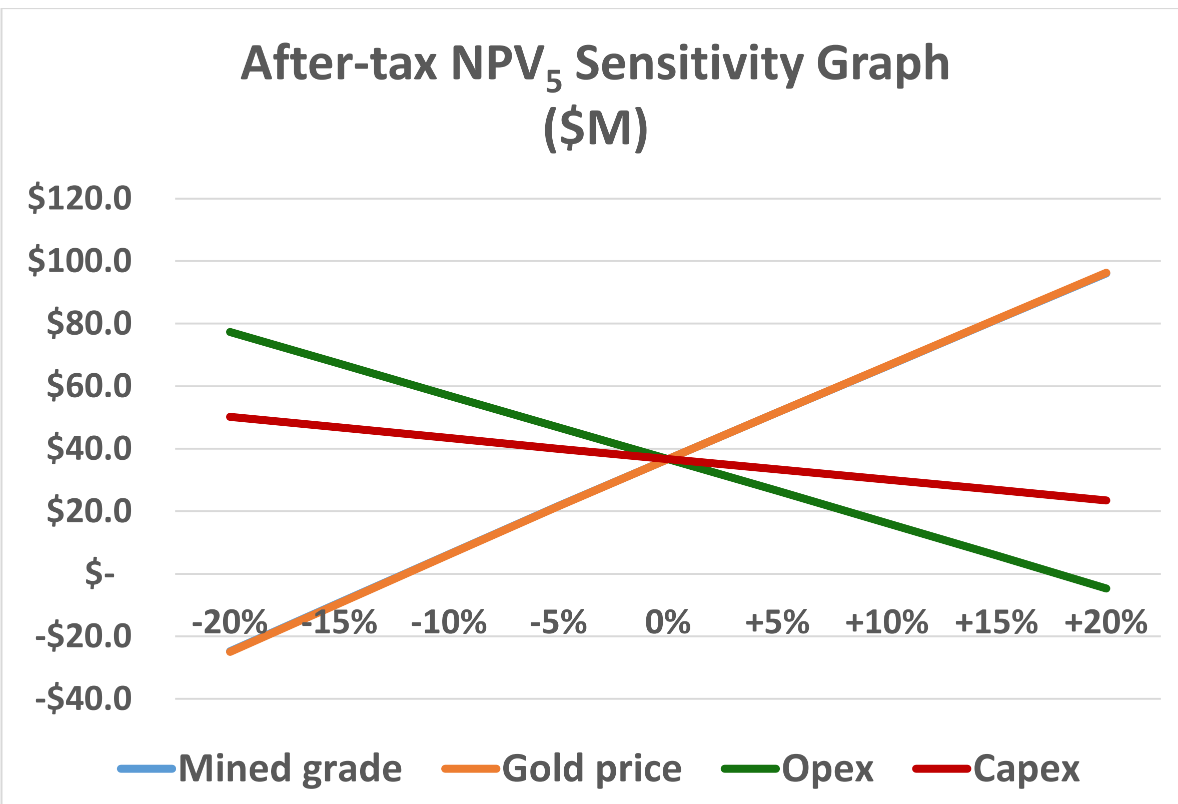


Figure 22.1 Graph of NPV sensitivity analysis

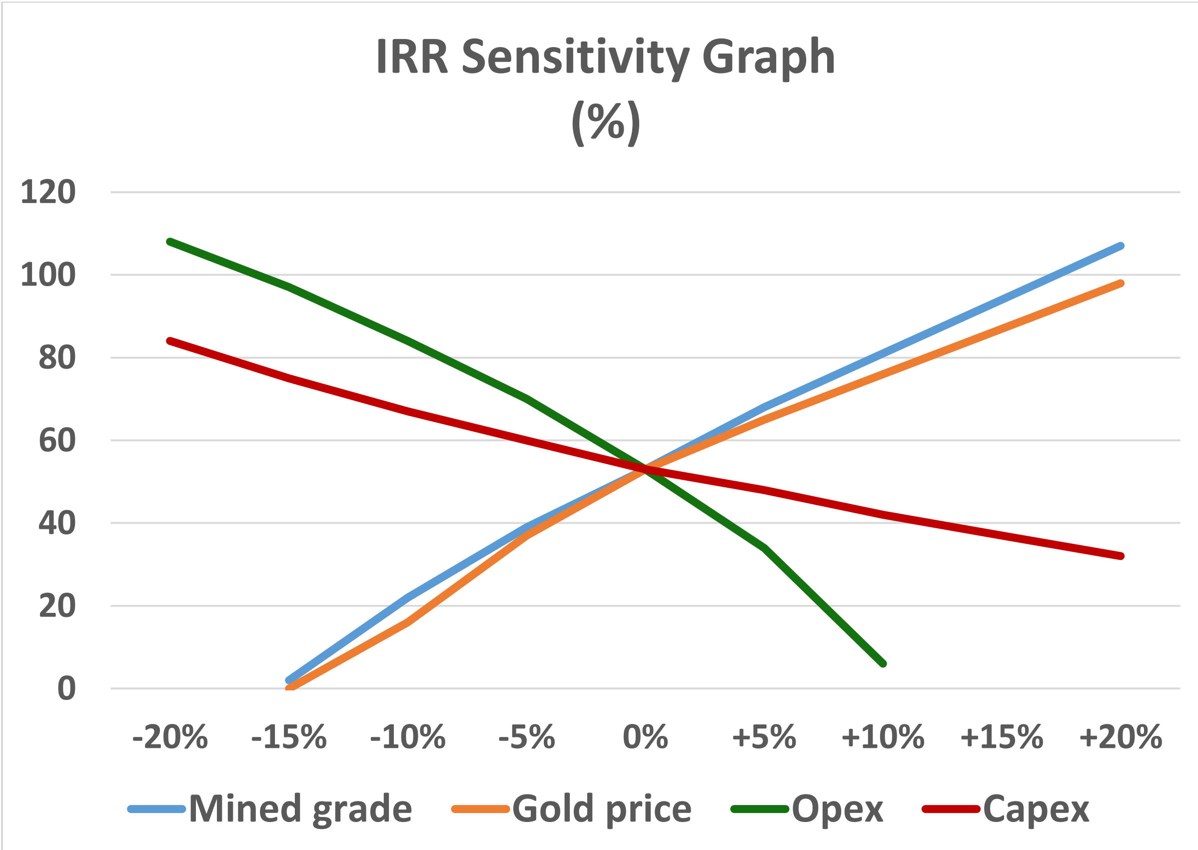


Figure 22.2 Graph of IRR sensitivity analysis

23 ADJACENT PROPERTIES

There are no adjacent properties to the PL Mine Property.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information.

25 Interpretation and Conclusions

The QP has reviewed the Maverick Project data provided by the Company, including drilling database; has visited the site; and has reviewed sampling procedures and security. The QP is of the opinion that the data presented by the Company are generally an accurate and reasonable representation of mineralization styles encountered at the Project.

It is the QP's opinion that Minnova's current sampling programs are conducted to industry standards and that the QA/QC programs undertaken by Minnova are sufficient to provide confidence in the analyses undertaken at the assay laboratories employed. Based on a review of available historical QA/QC data, and a positive comparison of the distribution of gold assay grades obtained by previous operators, the QP concludes that databases for the PL Deposit are of sufficient quality to provide the basis for the conclusions and recommendations reached in this Report.

Minnova's 2016 to 2017 in-fill drilling programs at the PL Deposit has provided further detail on the nature of the mineralized zones and permitted the completion of a NI 43-101-compliant updated Mineral Resource Estimate.

Minnova's 2017 summer exploration program has successfully discovered new gold showings in the PL Property package. The discovery of high-grade gold samples and their spatial distribution suggests exploration potential for additional gold mineralized zones in the PL Property and PL Deposit area.

Based on the positive results of the study presented in this Report, further work on this Project is warranted.

The present resource model assumes that all of the previously mined stope and development openings have been properly located in three-dimensional space with respect to the resource model. There may be some error in the spatial locations still unknown but any required adjustments will have an equal chance of either positively or negatively affecting the final resource to reserve conversion ratio. Therefore, the probability that the overall quantity of presently identified tonnages and grades in the Mineral Reserves at PL will change is well within an acceptable margin of error to enable the Project economics to be estimated to the required degree of accuracy for a Feasibility Study.

The Mineral Reserves estimate for the PL Mine Property amount to 1.3 million tonnes of ore at a gold grade of 6.34 g/t Au.

The mine plan, as presented, will support operation of the on-site processing plant for approximately 4.4 years at a nominal production rate of 590 mtpd or an average annual production rate of 216,000 tonnes.

The mine will require one year of pre-production development and construction prior to commencing production. The pre-production period will involve rehabilitating the portal, dewatering, and rehabilitating approximately 200 m of the existing ramp and new ramp, level and stope development in the new eastern zones of the mine. Infrastructure and supporting services will also be constructed and installed.

The proposed mining method is up dip stoping using an Alimak to develop and drill longholes for stopes. The stopes will utilise slushers to move the ore from the stopes to locations where LHDs and trucks will be used to move ore to the surface.

Stopes will be backfilled with hydraulic tailings.

Open pit mining will take place from 5 small open pits to be developed to approximately 40 m depth. The open pits would be mined by a contractor.

Mine manpower, included in the operating costs, totals 45 people during pre-production to an average steady state of approximately 100 personnel at full production. Staff comprise 16 people of the total production period mine manpower complement and hourly employees make up the remainder.

The processing plant will include a conventional circuit comprising crushing, grinding, gravity concentration using a Knelson concentrator, flotation, Merrill Crowe, leaching, and refining for gold recovery to doré bars.

Based on the metallurgical testwork and process presented in the previous sections, the expected gold recovery is 90% with a plant operating availability exceeding 90%.

The past employed Ragged Lake Tailings Management Facility, which is presently in the process of being added to Schedule 2 of the Metal Mining Effluent Regulations (MMER), designating the area as a tailings disposal area, will be recommissioned.

The processing plant will be operated on 2 shifts 7 days a week requiring a total of 16 personnel. The processing plant staff will total 9 people.

The estimated total Project pre-production capital expenditure, inclusive of contingencies and excluding working capital, is approximately $34.3 million. The total expenditures include EPCM, contractor overheads, and a 10% contingency on all estimated expenditures. A working capital allowance of $1.1 million is estimated to be required. Sustaining capital expenditures are estimated to be $53.7 million mainly related to ongoing mine development and tailings management and water treatment facilities improvements. Closure costs have been estimated at $1.5 million at the end of the Project life and is included as a separate line item in the cash flow model.

The estimated total average operating cost (excluding smelting and refining) for the PL Mine is approximately $194.08 per tonne ore and $163.44 for per tonne ore for underground and open pit mining, respectively.

The overall economic results indicate that the Project will have positive economic returns and generate approximately $47 million undiscounted after-tax cash flow ($71 million pre-tax) over the Project's 4.4 year mine life.

At the Base Case metal prices, the Project's after-tax net present value is estimated to be approximately $28.8 million at a discount rate of 10%. The post-tax IRR is estimated to be 53% and payback has been calculated at 1.2 years from start of production.

The sensitivity analysis shows that the Project economics are most sensitive to mined grade and gold price. The Project is least sensitive to capital expenditures.

25.1 PROJECT RISKS ASSESSMENT

25.1.1 GEOLOGY AND RESOURCE ESTIMATE

Technical factors that may affect the Mineral Resource estimates include:

- Gold price and valuation assumptions;
- Changes to the technical inputs used to estimate gold content (*e.g.*, bulk density estimation, and grade model methodology); and
- Geological interpretation (revision of vein models and the modeling of internal waste domains, *e.g.*, dikes and structural offsets such as faults and shear zones);

25.1.2 MINE

Existing mine rehabilitation can be unpredictable with respect to ground conditions and remediation work required. Conditions are expected to be good requiring predictable ground support but to minimise the impact of rehabilitation on mine production, only a short portion of the ramp near the portal will be utilised in the early years of the mine. A new ramp and level development will access new mining areas and the old working areas will be accessed and rehabilitated, as required later in the Project's life, if the costs allow.

Qualified mining personnel capable of performing the different tasks are not readily available in the area. Highly qualified people will be hired in other parts of Canada and will be flown in on rotation and live in the camp.

The groundwater table and inflows that reflect average conditions in the Feasibility Study could be worse than expected leading to added pumping and sump requirements but the pumps and sumps have excess capacity and extra sumps can be added with minimal effect on Project economics.

25.1.3 PROCESSING AND TAILINGS

No recent testwork has been conducted and the reliance on historical data creates the risk that that data may not satisfactorily apply to the current resource. For example, operating records indicate that solution fouling was periodically experienced. If significant fouling is encountered, mill operation and gold recovery could be adversely affected.

25.1.4 PROJECT FINANCING AND ECONOMICS

As with all resource development projects, there is the inherent risk that the Project will not raise the necessary capital to fund any new construction.

This Project is exposed to gold price and shows its greatest sensitivity to gold pricing. Tight control on capital and operating spending will alleviate much of this sensitivity, but a prolonged low gold price environment could render the mine uneconomic.

25.1.5 Environment and Permitting

The mine has all permits to operate an underground mine at the planned rate of 595 tonnes per day except for reactivation of the tailing management facility.

The inclusion Ragged TMF requires a listing on Schedule 2 of the Metal Mining Effluent Regulations (MMER).” The Company is in the process of submitting an Assessment of Alternatives report to determine if the Ragged TMA is in fact the best option for deposition of new tailings. The assessment will consider all possible alternatives for safe, long term tailings storage from environmental, socio-economic, and technical perspectives. Should the approval of the Schedule 2 listing be delayed the mine can deposit tailings in the old workings for a number of years.

25.1.6 Public Consultation

Consultation with local communities and First Nations groups has been ongoing and will continue as the project progresses. There is at present no significant opposition to the project and therefore the risk of social concerns impacting mine development and operation is very low.

26 RECOMMENDATIONS

Based on this Feasibility Study, the PL Mine Project should proceed to the pre-production phase (subject to financing) with recommendations as follows:

26.1 MINING

- Mobile mining equipment sourcing will be critical to determining Project construction start-up and should be an initial high priority.
- Prior to stope development, an updated study for rock mechanics and in stope support should be completed.
- During early development, further underground diamond drilling in stoping areas should be performed, as included in the Feasibility Study plan.

26.2 METALLURGY AND PROCESSING

- During the first 2 months of the pre-production period, further metallurgical testwork should be performed including variability and potential fouling materials testing.

26.3 INFRASTRUCTURE

- Completion of the access road and powerline upgrades are critical to allow year round access and minimise power costs.
- The initial expansion to the camp (to house people in the first 6 months) is also the other critical component to initial Project execution.

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Fred Marinellia and Walter L. Niccoli, 1998: *Simple Analytical Equations for Estimating Ground Water Inflow to a Mine Pit*, Ground Water, vol. 38, No. 2, March-April 2000, pg. 311-314.

WESA Inc., 2012: *Puffy Lake – Assessment of Pit Inflows and Groundwater Quality*, report dated June, 2012.

28 CERTIFICATES

CERTIFICATE OF QUALIFICATION OF CO-AUTHOR – Malcolm Buck

1. I, Malcolm K. Buck, P.Eng. (ON), do hereby certify that:
2. I am employed as Principal – Mine Evaluations by A-Z Mining Professionals Ltd. located at 1 King Street West, Suite 4800, Toronto, Ontario, M5H 1A1, Canada.
3. I graduated with a degree in Bachelor of Engineering, from the Technical University of Nova Scotia in 1983 and a Master's of Engineering (Mineral Economics), from McGill University in 1986
4. I am a Professional Engineer registered with the Professional Engineers of Ontario (PEO No. 5881503). I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.
5. I have approximately 35 years of experience in engineering, operations and mining projects economic evaluations for precious, base and other metal mines in Canada and around the world. Experience includes the completion of numerous NI 43-101 technical reports for mining projects.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I visited the PL Property on March 26 to 31, 2017.
8. I am an author or co-author of the technical report entitled: "Feasibility Study PL Gold Project, Manitoba, Canada" for Minnova Corp. dated April 23, 2018 (the "Report"). I am responsible for portions or all of the Chapters of the report, except for Chapters 6-14.
9. I have no prior involvement with the Issuer or the Property.
10. As of the effective date of the technical report (October 31, 2017), to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I am independent of the Issuer, and the Property applying all the tests in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: October 31st, 2017

DATED this 30th day of April 2018

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Malcolm Buck, P.Eng.

CERTIFICATE of QUALIFIED PERSON
BRIAN LEBLANC, P.ENG.

I, Brian LeBlanc, B.Sc., P. Eng., residing at 781 Community Hall Road, Thunder Bay, Ontario, Canada do hereby certify that:

1. I am President and a Principal of A – Z Mining Professionals Limited.

2. This certificate applies to the technical report titled **"FEASIBILITY STUDY, PL GOLD PROJECT, MANITOBA, CANADA, Prepared for Minnova Corp.** (the "Technical Report"), with an effective date of October 31, 2017.

3. I am a graduate of the Haileybury School of Mines as a Mining Technician (1981). I have also obtained a Bachelor of Science degree in Mining Engineering from Michigan Technological University (1986).

4. I am licensed by the Professional Engineers Ontario (License No. 90427972).

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. My relevant experience for the purpose of the Technical Report is:

   - Extensive and progressively more senior engineering and operational duties at base metals, gold and nickel mining operations and development projects.
   - 11.5 years of experience directing and overseeing several scoping level, pre-feasibility level and feasibility level studies for mines and mining companies.
   - Mill Operator – Giant Yellowknife Mines....................1974 – 1975
   - Crusher Operator/Screening Plant Operator/Loadout Operator/Surveyor – Steep Rock Iron Mines Ltd...........1976 - 1979
   - Mine Planner/Chief Surveyor – Nanisivik Mines Ltd....................1981 - 1984
   - Mining Engineer/Underground Supervisor/General Foreman/ Technical Services Superintendent/ Mine Superintendent – Williams Mine....................1986 – 2003
   - Manager of Mining – Kinross Kubaka Mine (Russia)....................2003 – 2004
   - Technical Services Superintendent – Lac Des Isles Mines....................2004 – 2006
   - Project Superintendent – Redpath Indonesia....................2006 - 2007
   - Project Manager for Ontario – North American Palladium Ltd....................2007 - 2010
   - Vice President and General Manager/ President – NordPro Mine and Project Management Services Ltd....... 2010 - 2014
   - President – A – Z Mining Professionals Limited....................February 2014 to Present

7. I assisted in preparation of the Technical Report and acted as a Peer Review for Sections 1, 13, and 16 to 26 of the Technical Report. I co-authored sections 16 and 18 of the Technical Report.

8. I have visited the Property that is the subject of this Technical Report.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

10. I am independent of the issuer applying all of the tests in sect 1.4 of NI 43-101.

11. I have not had prior involvement with the Property that is the subject of this Technical Report.

12. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

Effective date: April 26, 2018
Signing Date: April 26, 2018

{SIGNED AND SEALED}
[Brian LeBlanc]

Brian LeBlanc, P.Eng.

ALFRED S. HAYDEN
284 Rushbrook Drive
Newmarket, Ontario, L3X 2C9
Telephone (416) 460-3048
Email: a.hayden@ ehaengineering.com

CERTIFICATE of AUTHOR

I, Alfred S. Hayden, P. Eng., do hereby certify that:

1. I am currently President of:

   EHA Engineering Ltd.,
   Consulting Metallurgical Engineers
   Box 2711, Postal Stn. B.
   Richmond Hill, Ontario
   Canada L4E 1A7

2. I am a graduate of the University of British Columbia, Vancouver, B. C. in 1967 with a Bachelor of Applied Science in Metallurgical Engineering.

3. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and a Professional Engineer and Designated Consulting Engineer registered with the Association of Professional Engineers of Ontario.

4. I have worked as a metallurgical engineer for a total of 50 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 13 and portions of Sections 17 and 21 of this Technical Report: "Feasibility Study, PL Gold Project, Manitoba, Canada with an effective date of October 31, 2017. The information and data used in this report were obtained from the references cited. I have not visited the subject property.

7. I have had limited prior involvement with the property that is the subject of the Technical Report. I co-authored an independent technical report NI 43-101 "Preliminary Economic Assessment, Open Pit Mining and Milling at the Puffy Lake Gold Property, Maverick Gold Project, Sherridon Area, Manitoba, Canada", with an effective date of October 31, 2011, and a previous report "Preliminary Economic Assessment, Open Pit and Underground Mining and On-Site Milling at

the Maverick Gold Project, Including Mining of the PL Gold Deposit and the Nokomis Gold Deposit, Sherridon Area, Manitoba, Canada", with an effective date of July 7, 2014.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 23 April, 2018

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Alfred S. Hayden, P.Eng.

CERTIFICATE OF QUALIFICATION OF CO-AUTHOR – Ian Trinder, M.Sc., P.Geo.

1. I, Ian D. Trinder, M.Sc., P.Geo. (ON, MAN), do hereby certify that:
2. I am employed as a Principal Geologist by CSA Global Geosciences Canada Ltd. located at 365 Bay St., Suite 501, Toronto, Ontario, Canada. M5H 2V1.
3. I graduated with a degree in Bachelor of Science Honours, Geology, from the University of Manitoba in 1983 and a Master of Science, Geology, from the University of Western Ontario in 1989.
4. I am a Professional Geoscientist (P.Geo.) registered with the Association of Professional Engineers and Geoscientists of Manitoba (APEGM, No. 22924) and with the Association of Professional Geoscientists of Ontario (APGO, No. 452). I am a member of the Society of Economic Geologists and of the Prospectors and Developers Association of Canada.
5. I have approximately 30 years of direct experience with precious and base metals mineral exploration in Canada, USA and the Philippines including project evaluation and management. Additional experience includes the completion of various National Policy 2A and NI 43-101 technical reports for gold and base metal projects.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited the Maverick Project (PL and Nokomis Properties).
8. I am a co-author of the technical report titled: "Feasibility Study PL Gold Project, Manitoba, Canada" for Minnova Corp. dated April 23, 2018 (the "Report"). I am responsible for Sections 4.1 to 4.3, 4.6, 5.1 to 5.3.4, 5.4, 6 to 10 and 11.1 to 11.4 of the Report and portions of Sections 1, 25 and 27 that pertain to the Report sections for which I am responsible.
9. I have no prior involvement with the Issuer or the Property.
10. As of the effective date of the technical report (October 31, 2017), to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I am independent of the Issuer, and the Property applying all the tests in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: October 31st, 2017

DATED this 23th day of April 2018

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Ian D. Trinder, M.Sc., P. Geo.

CERTIFICATE OF QUALIFICATION OF CO-AUTHOR – Leon McGarry, B.Sc., P.Geo.

1. I, Leon McGarry B.Sc., P.Geo. (ON, SASK), do hereby certify that:
2. I am employed as a Senior Resource Geologist by CSA Global Geosciences Canada Ltd. located at 365 Bay St., Suite 501, Toronto, Ontario, Canada, M5H 2V1.
3. I graduated with a degree in Bachelor of Science Honours, Earth Science, from Brunel University, London, United Kingdom, in 2005 and have practiced the profession of geoscience since my graduation.
4. I am a Professional Geoscientist (P. Geo.) registered with the Association of Professional Geoscientists of Ontario (APGO, No. 2348) and with the Association of Professional Geoscientists of Saskatchewan (APEGS, No.34929).
5. I have practiced my profession for over 10 years and have been employed as a consultant with ACA Howe since 2007 and then with CSA Global since 2016. I have over six years of direct experience with the preparation of mineral resource estimations for precious metal deposits in Canadian Proterozoic terrains. Additional experience includes over eight years of direct experience in the exploration and geological modeling of precious and base metal projects, and the completion of NI 43-101 technical reports.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have completed a site to the PL Deposit between May 28th and the 30th, 2017, and the Nokomis Deposit between January 20th and 21st, 2017.
8. I am a co-author of the technical report titled: Feasibility Study PL Gold Project, Manitoba, Canada" for Minnova Corp. dated April 23, 2018 (the "Report"). I am responsible for Sections 11.5, 12 and 14 of the Report and portions of Sections 1, 25 and 26 that pertain to the Report sections for which I am responsible.
9. I prepared a Mineral Resource Estimate for the Nokomis Deposit in 2014 and co-authored the Issuer's 2014 NI 43-101 technical report for the Maverick Project.
10. As of the effective date of the technical report (October 31, 2017), to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
11. I am independent of the Issuer, and the Property applying all of the tests in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Effective Date: October 31st, 2017

DATED this 23rd day of April 2018

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Leon McGarry, B.Sc., P. Geo

CERTIFICATE of QUALIFIED PERSON

BYRON O'CONNOR, P.ENG.

I, Byron O'Connor, P.Eng., residing at 446 Roosevelt Drive, Kingston, Ontario, Canada do hereby certify that:

1. I am a Senior Engineer and Director of Engineering at BluMetric Environmental Inc.
2. This Certificate applies to the technical report titled "FEASIBILITY STUDY, PL GOLD PROJECT, MANITOBA, CANADA, Prepared for Minnova Corp. (The Technical Report), with an effective date of October 31, 2017.
3. I am a graduate of the University of New Brunswick in geology (1986) and geological engineering (1989).
4. I am licenced by Professional Engineers Ontario (Licence No. 90323999), Professional Engineers and Geoscientists of Newfoundland and Labrador (Member 05067), Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories and Nunavut (Licence L1670) and Engineers Yukon (Licence 1976).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have been providing environmental consulting services since 1989 and have been providing environmental consulting services to the mining sector since 1999.
7. I co-authored Section 20 of the Technical Report.
8. I visited the Property that is subject of this Technical Report in 2011.
9. I am independent of the issuer applying all of the tests in Section 1.4 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1.

Effective Date: April 30, 2018

Signing Date: April 30, 2018

{Signed & Sealed}

Byron O'Connor

Byron O'Connor, P.Eng.

LICENSED PROFESSIONAL ENGINEER
B. R. O'CONNOR
PROVINCE OF ONTARIO