THE NEW POLARIS GOLD PROJECT, BRITISH COLUMBIA, CANADA 2019 PRELIMINARY ECONOMIC ASSESSMENT

Northwestern British Columbia
Atlin Mining Division
NTS: 104 K 12
133º37'W Longitude and 58º42'N Latitude

Report Authors
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Tracey Meintjes, P.Eng.

Report Effective Date
February 28, 2019

CERTIFICATE OF QUALIFIED PERSON

I, Marc Schulte, P.Eng. am a Mining Engineer with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.

This certificate applies to the Technical Report titled "The New Polaris Gold Project, British Columbia, Canada, 2019 Preliminary Economic Assessment" that has an effective date of 28 February 2019 (the "Technical Report"). 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.

I am a Professional Engineer in the Province of Alberta. (#71051). I graduated with a Bachelor of Science in Mining Engineering from the University of Alberta in 2002.

I have worked as a Mining Engineer for a total of 17 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the New Polaris Property.

I am responsible for Sections 1-3, 18, 19, 21-27 of the Technical Report.

I am independent of Canarc Resources Corp. as independence is described by Section 1.5 of NI 43–101.

I have not previously co-authored reports on the New Polaris Project.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 28 February 2019

"Signed and sealed"

____

Marc Schulte, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Robert J. Morris, P.Geo., am a Geologist with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.

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I am a member of Engineers and Geoscientists British Columbia (#18301). I graduated with a Bachelor of Science in Geology from the University of British Columbia in 1973 and have a Master of Science from Queens University, 1978.

I have worked as a Geologist for over forty years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the New Polaris Property during the period 22-23 August 2006. My prior involvement with the project includes a site visit 21 August 1988, and work on the geology, resource estimate, and exploration program during 1988 and 1989, while with Beacon Hill Consultants.

I am responsible for Sections 14-12 of the Technical Report.

I am independent of Canarc Resources Corp. as independence is described by Section 1.5 of NI 43–101.

I co-authored reports on the New Polaris Project, including the following:

- Giroux G. H. and Morris R. J. 2007: Resource Potential New Polaris Project Technical Report, report prepared by Giroux Consultants Ltd. and Moose Mountain Technical Services for Canarc Resource Corp., effective date 5 March 2007
- Giroux G.H., Gray J.H. and Morris R.J. 2007: New Polaris Project - Preliminary Assessment, report prepared by Moose Mountain Technical Services for Canarc Resource Corp., effective date 4 October 2007
- Giroux G.H., Gray J.H. and Morris R.J. 2009: New Polaris Project - Preliminary Assessment, report prepared by Moose Mountain Technical Services for Canarc Resource Corp., effective date 23 December 2009
- Giroux G.H., Gray J.H. and Morris R.J. 2011: New Polaris Project – Preliminary Assessment Update, report prepared by Moose Mountain Technical Services for Canarc Resource Corp., effective date 10 April 2011

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 28 February 2019

“Signed and sealed”

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Robert J. Morris, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Sue Bird, P.Eng. am a Geological Engineer with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.

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I am a Professional Engineer in the Province of British Columbia. (#25007). I graduated with a Geologic Engineering degree (B.Sc.) from the Queen's University in 1989 and a M.Sc. in Mining from Queen's University in 1993.

I have worked as an engineering geologist for over 25 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the New Polaris Property.

I am responsible for Section 14 and portions relevant to Resource Estimation of Sections 1, and 25-27 of the Technical Report.

I am independent of Canarc Resources Corp. as independence is described by Section 1.5 of NI 43–101.

I have not previously co-authored reports on the New Polaris Project.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 28 February 2019

"Signed and sealed"
____
Sue Bird, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Michael Petrina, P.Eng. am a Mining Engineer with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.

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I am a Professional Engineer in the Province of British Columbia. (#15651). I graduated with a Bachelor of Applied Science in Mining Engineering from Queen's University in 1983.

I have worked as a Mining Engineer for a total of 35 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the New Polaris Property in October and December 2006.

I am responsible for Section 16 of the Technical Report.

I am independent of Canarc Resources Corp. as independence is described by Section 1.5 of NI 43–101.

I have not previously co-authored reports on the New Polaris Project.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 28 February 2019

"Signed and sealed"
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Michael Petrina, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Tracey Meintjes, P.Eng. am a Metallurgical Engineer with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.

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I am a member in good standing Engineers and Geoscientists British Columbia (#37018). I am a graduate of the Technikon Witwatersrand, (NHD Extraction Metallurgy – 1996).

My relevant experience includes metallurgy and process engineering, and mine planning in South Africa and North America. My experience includes both operations and metallurgical process development including base metals, precious metals, industrial minerals, coal, uranium and rare earth metals. My precious metals experience includes both operations and metallurgical process development. I have been working in my profession continuously since 1996.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the New Polaris Property.

I am responsible for Section 13 and 17 of the Technical Report.

I am independent of Canarc Resources Corp. as independence is described by Section 1.5 of NI 43–101.

I have not previously co-authored reports on the New Polaris Project.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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Dated: 28 February 2019

“Signed and sealed”

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Tracey D. Meintjes, P.Eng.

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1.1 Introduction

Mr. Marc Schulte, P.Eng. Mr. Bob Morris, P.Geo., Mrs. Sue Bird, P.Eng., Mr. Michael A Petrina, P.Eng., Mr. Tracey Meintjes, P.Eng., have prepared an NI 43-101 Technical Report (the Report) on the New Polaris Gold Project (the Project) for Canarc Resource Corp (Canarc). The Report is based on an updated Mineral Resource Estimate and Preliminary Economic Assessment (PEA) on the Project.

The New Polaris Gold Project is located in northwestern British Columbia (BC), about 100 km south of Atlin, BC, and 60 km northeast of Juneau, Alaska.

1.2 Terms of Reference

The Report has been prepared in support of disclosures in Canarc's news release dated 4 March 2019, entitled "Canarc Announces Robust Preliminary Economic Assessment on the New Polaris Gold Mine Delivering Post-Tax IRR of 38%".

A Mineral Resource Estimate and a PEA on the Project were completed in 2007. The PEA was updated in 2009 and 2011, based on the same Mineral Resources from 2007, but updated gold prices and capital and operating cost estimates.

An updated Mineral Resource Estimate and PEA have been completed in 2019. Information from this study has been summarized into this Report in the relevant sections.

The Preliminary Economic Assessment is preliminary in nature; that it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment results will be realized.

Units used in the report are metric units unless otherwise noted. Monetary units are in Canadian dollars (C$) unless otherwise stated.

Figures throughout the Report are plotted on two different coordinate systems, a UTM set and a mine specific set. The UTM coordinate system is WGS 84 Zone 8V. Resource estimation and mine planning have been run within the mine specific coordinate system, and all Figures related to these portions of the Report are shown within this space. Translation to UTM space is assumed to be a normal shift in space with no rotation.

1.3 Project Setting

New Polaris (formerly Polaris-Taku Mine) is an early Tertiary mesothermal gold mineralized body located in northwestern British Columbia about 100 km south of Atlin, BC and 60 km northeast of Juneau, Alaska (Figure 1-1). The nearest roads in the area terminate 20 km south of Atlin, and approximately 100 km from the Project. Access at the present time is by aircraft. A short airstrip for light aircraft exists on the property. Shallow draft barges have been used in the past to access the site via the Taku River to transport bulk supplies and heavy equipment to site, as well as ship flotation concentrate from site.

The New Polaris project area lies on the eastern flank of the steep, rugged, Coast Range Mountains, with elevations ranging from the sea level to 2,600 metres. The climate is one of heavy rainfalls during the late summer and fall months, and comparatively heavy snowfall, interspersed with rain during the winter.

Operations will include year-round underground mining activities and onsite processing to produce doré, and seasonal barge shipping of supplies to site. Onsite support for the operation and management of a camp with fly-in/fly-out service to an onsite landing strip has been planned.

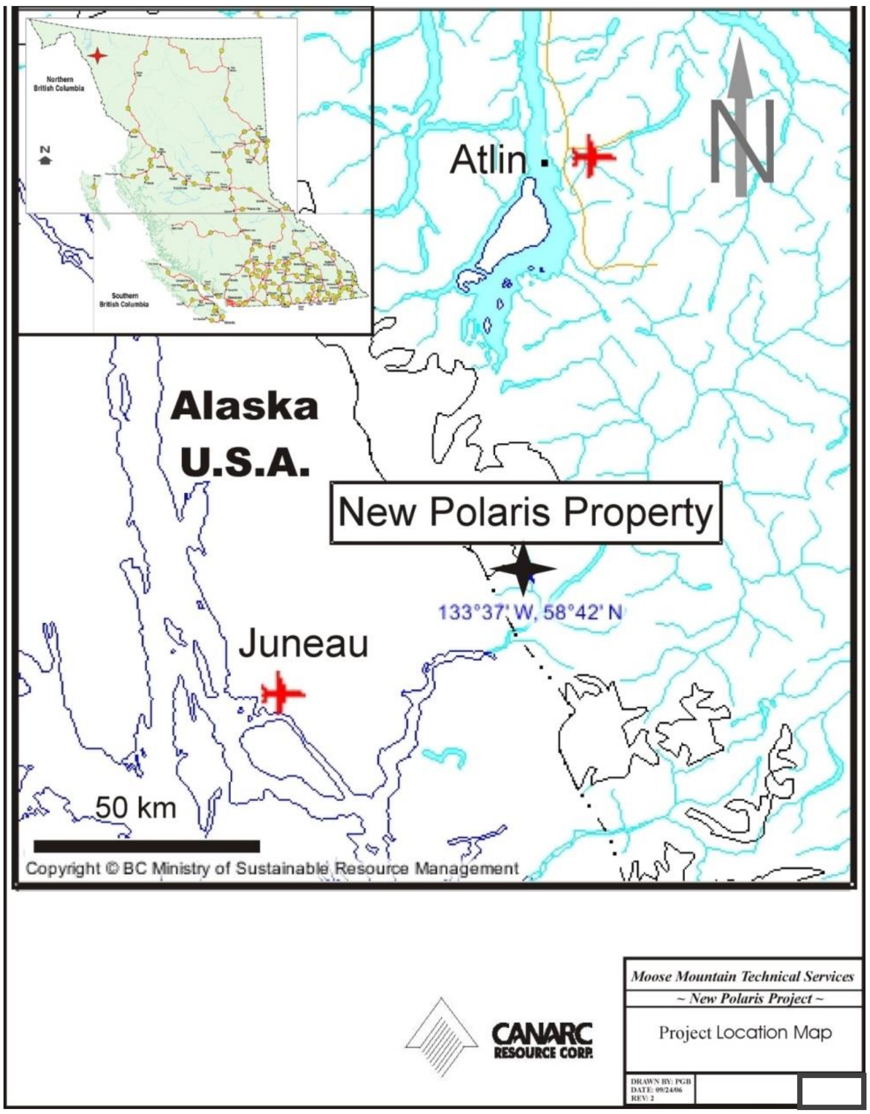

Figure 1-1 Location Map

1.4 Mineral Tenure, Royalties and Agreements

The property consists of 61 contiguous Crown-granted mineral claims and one modified grid claim covering 2,100 acres. All claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canarc Resource Corp. subject to a 15% net profit interest held by Rembrandt Gold Mines Ltd. Canarc can reduce this net profit interest to a 10% net profit by issuing 150,000 shares to Rembrandt.

Some surface rights for the proposed Project operations lie with the Crown and will need to be obtained from the Province of British Columbia. Water rights for proposed Project operations will need to be obtained from the Province of British Columbia.

Prior to commencing further exploration on the property, a Notice of Work is required to be submitted to the Mining and Minerals Department of the BC Ministry of Energy and Mines. Work can commence once approval has been received.

1.5 Geology and Mineralization

The deposit is composed of three sets of veins (quartz-carbonate stringers in altered rock), the “A-B” veins are northwest striking and southwest dipping, the “Y” veins are north striking and dipping steeply east and finally the “C” veins are east-west striking and dipping to the south to southeast at 65º to vertical. The “C” veins appear to hook around to the north and south into the other two sets of veins so that their junctions form an arc. The gold is refractory and occurs dominantly in finely disseminated arsenopyrite grains that mineralize the altered wallrock and stockwork veins. The next most abundant mineral is pyrite, followed by minor stibnite and a trace of sphalerite. The zones of mineralization range from 15 to 250 metres in length and 0.3 to 14 metres in width.

1.6 History

The deposit was mined by underground methods from 1938 to 1942, and from 1946 to early 1951, producing a total of 740,000 tonnes of ore at an average grade of 10.3 g/t gold. Recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones. Geological mapping, geochemical surveys, geophysical techniques, and drilling have expanded the resources at the Project.

1.7 Drilling and Sampling

Canarc explored the “C” vein system between 1988 and 1997, and carried out infill drilling in 2003 through 2006, to better define the continuity and grade of the vein systems.

Sampling of the vein was done by wire line diamond drills using NQ-size rods. True widths of the mineralized zone vary from 70% to 100% of the drill core intercept angle.

The 2006 QA/QC program was similar to the previous programs in that samples were collected by employees of Canarc on site and shipped to ALS Chemex laboratory in Vancouver. For quality control and quality assurance, core samples were regularly mixed with blanks, duplicates, and standards. The program in the field was run in an efficient and proper manner following accepted engineering standards.

Sample preparation, analysis and security procedures undertaken by Canarc are generally performed in accordance with exploration best practices and industry standards.

Sufficient verification checks have been undertaken on the databases to provide confidence that the databases are reasonably error free and may be used to support Mineral Resource estimation.

1.8 Mineral Resource Estimate

An updated Mineral Resource Estimate has been prepared in 2019. The updated estimate uses drillhole data from 1989-2006 and excludes drilling prior to this, or for which the drill year is not known. The resource is based on 174 drillholes and 1,464 assay intervals which intersect the veins within the 1989-2006 data set. Ordinary kriging has been used to interpolate the gold grade of six veins as modelled by Canarc geologists.

The geologic continuity of the "C" vein system has been well established through historic mining and diamond drilling. Grade continuity has been quantified using a geostatistical semi-variogram, which is used to determine the distances (ranges) and directions of maximum continuity in the three principle directions. The four main veins in the semi-variogram model produced ranges between 60 and 120 m along strike and down plunge.

Capping of the assays in each vein has been evaluated using cumulative probability plots (CPPs).

For this study, the classification to Measured, Indicated or Inferred also required that the true thickness of the vein is at least 2 m. Blocks are considered Indicated if the average distance to the nearest two drillholes used in the interpolation is within 30 m, or if there is at least one drillhole within 10 m and at least two drillholes used in the interpolation. Veins 7 and 8 (Y19 and Y20) are considered Inferred due to lack of QA/QC documentation for the drilling within these veins.

1.9 Mineral Resource Statement

Confining shapes have been created targeting material above a series of cutoff grades. The total material within each confining shape is reported in the table below (i.e. no cutoff has been applied within each confining shape in order to report the underground Mineral Resource Estimate, which is in line with the mining method).

A cutoff grade of 4.0 g/t gold, highlighted in the table below, is selected as the economic cutoff for the Project. The confining shape which targets material above this grade is used to define the "reasonable prospects of eventual economic extraction" for the Mineral Resource Estimate. The 4.0 g/t target includes the following considerations: gold price of US$1,300/oz, exchange rate of 0.77 US$:C$; Payable gold % of 99.9%, Offsite refining costs of US$7/oz, mining costs of C$65.20/t, process costs of C$62.70/t, G&A (General and Administration) costs of C$37.00, sustaining capital costs of C$19.83/t, and a 90.5% process recovery.

MMTS is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that could materially affect the Mineral Resource Estimate. Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.

Table 1-1 New Polaris Indicated Resources

Confining Shape Target Grade (g/t Au)	In Situ Tonnage (tonnes)	In Situ Au Grade (g/t)	In Situ Au Content (Oz.)
2.0	1,880,000	10.0	605,000
3.0	1,798,000	10.4	599,000
4.0	**1,687,000**	**10.8**	**586,000**
5.0	1,556,000	11.3	567,000
6.0	1,403,000	12.0	540,000
7.0	1,260,000	12.6	509,000
8.0	1,105,000	13.3	472,000
9.0	947,000	14.1	428,000

Table 1-2 New Polaris Inferred Resources

Confining Shape Target Grade (g/t Au)	In Situ Tonnage (tonnes)	In Situ Au Grade (g/t)	In Situ Au Content (Oz.)
2.0	1,639,000	9.5	502,000
3.0	1,582,000	9.8	497,000
4.0	**1,483,000**	**10.2**	**485,000**
5.0	1,351,000	10.7	464,000
6.0	1,223,000	11.2	441,000
7.0	942,000	12.5	380,000
8.0	753,000	13.8	334,000
9.0	653,000	14.6	306,000

Notes for Mineral Resource Estimate:

- *The Mineral Resource Estimate was prepared by Sue Bird, P.Eng. in accordance with CIM Definition Standards (CIM, 2014) and NI 43-101, with an effective date of February 28, 2019.*
- *Mineral Resources that are not mineral reserves do not have demonstrated economic viability.*

1.10 Mining Methods

The mine production plan is based on a subset of the Mineral Resources stated above, focusing on resources within a confining shape targeting material above a 6.0 g/t cutoff grade.

The mine plan uses a combination of conventional cut and fill mining on 24% of the deposit and longhole stoping on 58% of the deposit; depending on mineralization thickness and continuity. Development in ore makes up the remaining 18% of production.

Waste development will include a decline from surface, extraction drifts on sublevels across the footwall of the orebody, and ventilation raises to the surface.

1.11 Metallurgy and Recovery Methods

Gold is associated with arsenopyrite and is refractory. Metallurgical test work has demonstrated that bio-oxidation (BIOX) and Carbon-in Leach (CIL) processing of flotation concentrate to produce doré results in an overall gold recovery of 90.5%.

An onsite process plant will consist of crushing, grinding and floatation to produce a bulk floatation concentrate. The flotation concentrate will be treated using a bio-oxidation and CIL plant, followed by carbon stripping, Electrowinning and refining to produce doré gold bars. Leach tails will be detoxified using the ASTER™ treatment and SO_2 air processes.

Floatation and leach tails will be thickened and 42% of this material will be pumped underground to a paste plant and distribution system into mined out voids and the remaining 58% filtered and hauled to a co-disposal facility (CDF) along with mined waste rock from underground development.

1.12 Project Infrastructure

Capital and operating cost estimates include the cost to purchase, transport, construct and operate onsite infrastructure, equipment, supplies, and personnel to support the operation. The project will be fly-in/fly-out, with an onsite camp and airstrip. Major bulk supplies for mining and processing will be barged along the Taku River on a seasonal basis between May and September each year.

A portion of the process tailings and all underground mined waste rock is planned to be co-disposed in a facility constructed 2.5 km north of the process plant location. The co-disposal methodology employed will be co-placement into an integrated disposal facility.

The locations of the site facilities are shown in the following General Arrangement maps.

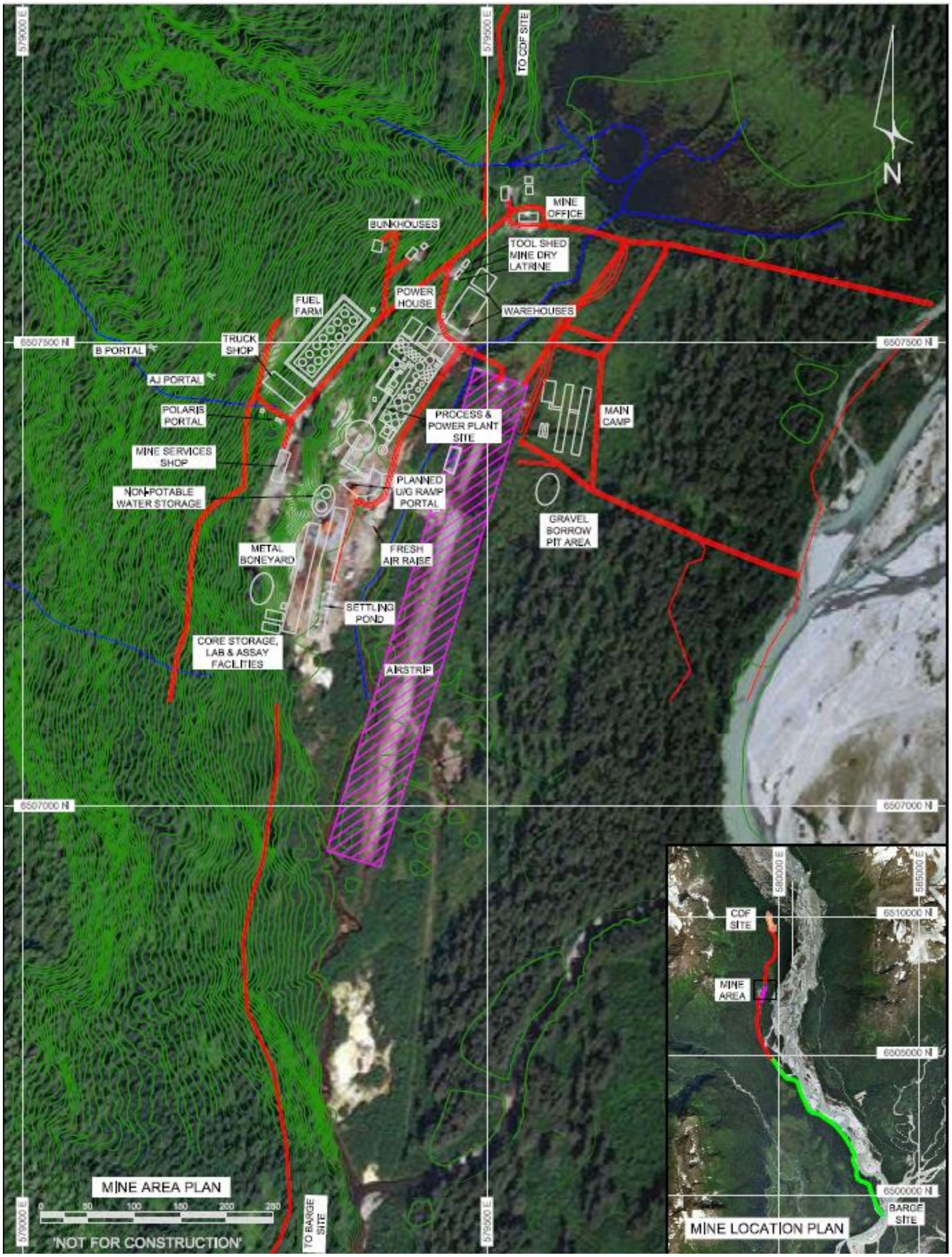

Figure 1-2 New Polaris General Arrangement

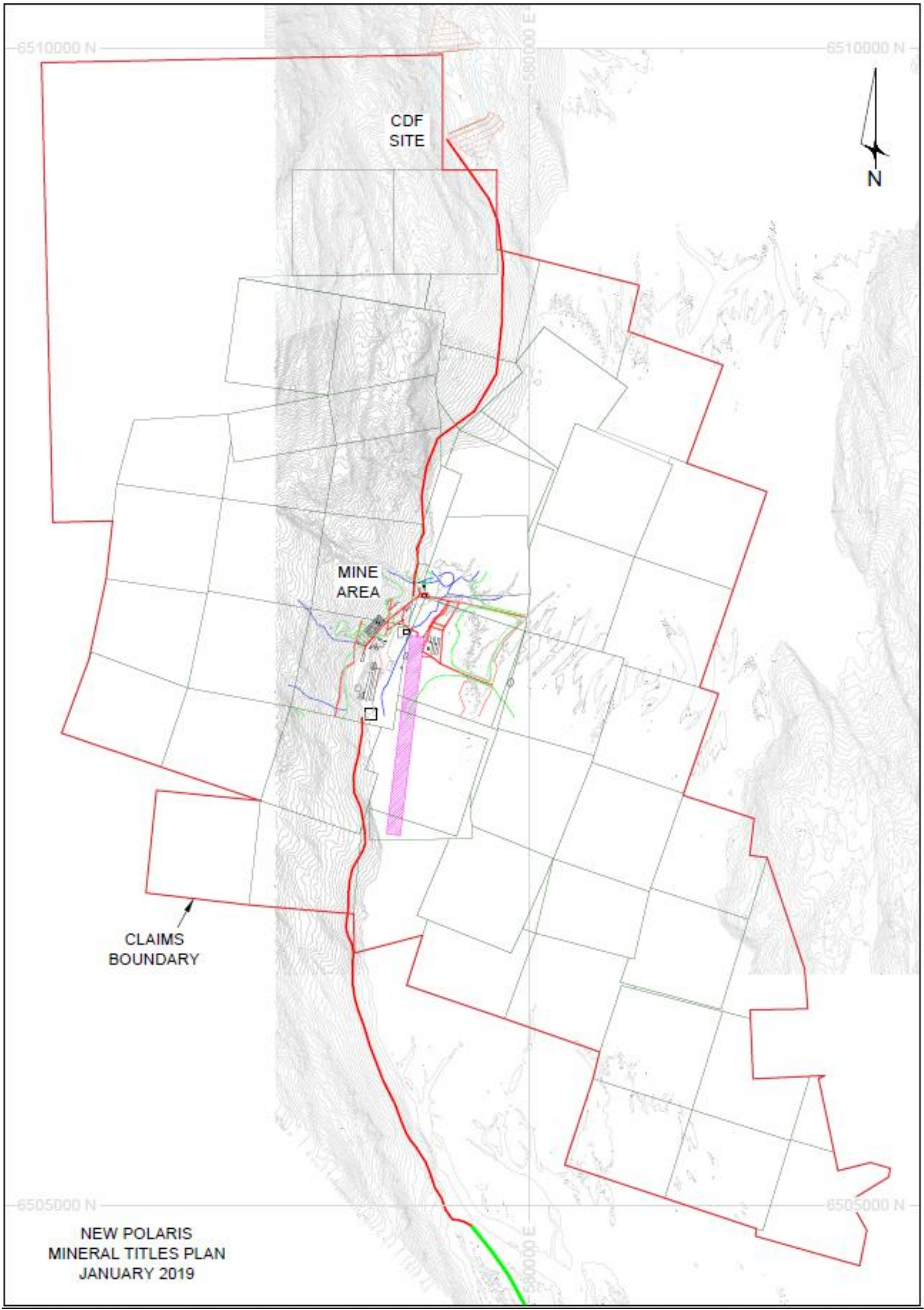

Figure 1-3 New Polaris General Arrangement Map with Claim Boundaries

1.13 Markets and Contracts

A long-term gold price of US$1,300/oz is considered reasonable with respect to the prevailing market and has been used in the study.

Terms, conditions and charges for refining and treatment of doré will be typical of similar contracts in the industry. Offsite costs and payable percentages are applied to the gold market price.

1.14 Environmental, Permitting and Social Considerations

The Project is located within the land claim and traditional territory of the Taku River Tlingit First Nation (TRTFN). No formal agreements are in place with the TRTFN, however an open communication with the Project proponent has been established and used for some time. TRTFN have indicated they support industrial projects within their traditional lands assuming Management Plans comply with their accepted guidelines. Additional baseline data collection work required to meet regulatory requirements for Archeology and Heritage Resources will be completed during the Feasibility and Permitting stages of the Project. The proponent will continue to engage and consult with the TRTFN on all Project components and potential impacts.

A program of Consultation and Engagement is also required for potentially affected Communities and other stakeholders.

The Project will be subject to permitting under BCEAO and CEAA as its production exceeds the threshold requirements for mining activities. Additional consultations and input may also be required from Alaska, United States, authorities as the rivers in the Project area drain into US waters (Transboundary Water).

Some environmental studies have been initiated; however more work is required to meet regulatory requirements. Due to the time elapsed since collection; it is possible that a large portion of the current dataset will need to be refreshed to reflect existing conditions. Critical components of impact mitigation include Management Plans for land, water, air and groundwater.

It is not unreasonable in this jurisdiction to expect approvals will be received conditional upon acceptance of respective Management Plans and commitments to mitigate impacts from operations.

The project will be designed to minimize environmental impacts during the construction and operating phases of the mine to minimize any long-term environmental impacts and facilitate reclamation and closure. Future work will incorporate and complement all previously completed aquatic, hydrogeology and terrestrial baseline study data collected at the site.

Previous geochemical sampling shows the waste rock to be non-acid generating in nature, due to high carbonate content. More static testing and the introduction of kinetic tests are needed to more accurately characterize waste rock, tailings and effluent from leaching operations.

For tailings, Acid Base Accounting tests show potential for acid generation and leaching of copper, lead, zinc, and iron. Tailings are planned to be disposed of in both underground and surface facilities. Tailing disposed of on surface will be thickened and disposed of in a Co-Disposal Facility (CDF) with mine waste rock. The CDF does not store water. Underground disposal of tailings and paste fill will be directed to mined out areas to minimize the surface disturbance footprint.

The Project is located in an historic mining area that currently has low usage for mining, exploration, hunting, fishing, and trapping activities. The New Polaris site was previously mined between 1938 and 1956 with remnants of the old activities still present at the site.

1.15 Capital Cost Estimates

A Project capital cost estimate has been completed at a Scoping level of accuracy. The base date of the estimate is first quarter 2019. The scope of work includes direct costs, indirect costs, owner's costs and contingency. Sustaining capital for the Project is estimated at $56.4 million over the 8.7-year mine life.

Table 1-3 Capital Cost Summary

Area	Capital Estimate (M$)
Overall Site	$12.8
Mining	$20.0
Plant	$39.8
Waste/Tailings	$6.8
Site Services	$4.3
Surface Mobile Fleet	$2.5
Buildings and Facilities	$8.4
Total Direct Costs	$94.5
Total Indirect Costs	$25.2
Contingency (20%)	$23.9
Total	**$143.7**

1.16 Operating Cost Estimates

Operating costs were calculated based on mine manpower, supplies and equipment costs, process manpower supplies and consumables, and general and administrative costs (G&A). Operating costs and any revenue from production incurred during the pre-production period until commercial production is achieved were capitalized.

Table 1-4 Operating Cost Summary

Area	Operating Estimate	Units
Mining	$65.25	$/t milled
Processing	$62.70	$/t milled
G&A	$37.00	$/t milled
Total	**$164.95**	**$/t milled**

1.17 Economic Analysis

The results of the economic analyses discussed in this section represent forward looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected recovery rates;
- Sustaining capital costs and proposed operating costs;
- Assumptions of closure costs and closure requirements;
- Assumptions of environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade, or recovery rates;
- Geotechnical and hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment, or processes to operate as anticipated;
- Accidents, labour disputes and other risks of the mining industry.

The results of the Preliminary Economic Assessment are shown in Table 1-5.

Table 1-5 PEA Summary

Mill Feed Production	2,306,000 tonnes (subset of Mineral Resource)	
Production Rate	750 tonnes per day	
Production Grade	10.3 grams per tonne	
Recoveries	90.5% gold into doré	
Average Output	80,000 oz gold per year	
Mine life	8.7 years	
The Base Case Financial Parameters:		
Gold Price	US$ 1300 per oz	
Exchange Rate	US$ 1.00 = C$ 1.30	
Capital Cost	$143.7 million	
Cash Cost **All -in-Sustaining Costs**	US$ 433 per oz US$ 510 per oz	
	Pre-Tax	After Tax
Cash Flow (LOM)	$554.0 million	$414.4 million
NPV (5%)	$384.8 million	$280.4 million
	Pre-Tax	After Tax
Internal Rate of Return	47.2%	37.7%
Payback Period	2.3 years	2.7 years

1.18 Interpretations and Conclusions

The PEA indicates that the New Polaris Project has positive results and therefore further work is recommended to gather additional site baseline data, optimize the project, and complete a Preliminary Feasibility Study.

2 Introduction

Canarc Resource Corp. (Canarc) is engaged in the exploration and advancement of the New Polaris Gold Project (the Project) in northwestern British Columbia, Canada.

Mr. Marc Schulte, P.Eng. Mr. Robert J. Morris, P.Geo., Mrs. Sue Bird, P.Eng., Mr. Michael A Petrina, P.Eng., Mr. Tracey Meintjes, P.Eng., have prepared an NI 43-101 Technical Report (the Report) on the Project for Canarc. The Report is based on an updated Mineral Resource Estimate and Preliminary Economic Assessment (PEA) on the Project.

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The Report has been prepared in support of disclosures in Canarc's news release dated 4 March 2019, entitled "Canarc Announces Robust Preliminary Economic Assessment on the New Polaris Gold Mine Delivering Post-Tax IRR of 38%".

A Mineral Resource Estimate and a PEA Technical Report on the Project was completed in 2007. The PEA Technical Report was updated in 2009 and 2011, based on the same Mineral Resources from 2007, but updated gold prices and capital and operating cost estimates.

An updated Mineral Resource Estimate and PEA has been completed in 2019. Information from this study has been summarized into this Report in the relevant sections.

The overall Report effective date is February 28, 2019, marking the completion of the financial analysis that supports the PEA. The effective date of the Mineral Resource Estimate is also February 28, 2019.

The Preliminary Economic Assessment is preliminary in nature; that it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Mineral Resource estimates were performed in accordance with the 2003 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and reported in accordance with the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

The quality of information, conclusions and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project. The Report is based on: i) information available at the time of preparation, ii) data supplied by outside sources, iii) engineering, evaluation, and costing by other technical specialists and iv) the assumptions, conditions and qualifications set forth in this Report. These assumptions include future gold prices, refinery terms, shipping terms, supply costs, and labour rates, all of which are forward looking estimates.

This Report is intended to be used by Canarc as a scoping study, subject to the terms and conditions of its contract with MMTS and is intended to show the potential of the property. Further level of detailed resource modeling, engineering and other technical studies are required to advance the project to a production decision. Certain cost and price prediction used in this Report are forward looking and as such cannot be relied upon as indicative of what will actually occur in the future if the property is developed and production commences. Readers of this report should update and evaluate the key economic parameters in this study at such time as they are evaluating the property potential or request such from MMTS.

Units used in the report are metric units unless otherwise noted. Monetary units are in Canadian dollars (C$) unless otherwise stated.

Figures throughout the Report are plotted on two different coordinate systems, a UTM set and a mine specific set. The UTM coordinate system is WGS 84 Zone 8V. Resource estimation and mine planning have been run within the mine specific coordinate system, and all Figures related to these portions of the Report are shown within this space.

Sources of information are listed in Section 27.

2.2 Qualified Persons

The following serve as the qualified persons (QP) for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Mr. Marc Schulte, P.Eng., Associate, Mine Engineering, Moose Mountain Technical Services (MMTS)
- Mr. Robert J. Morris, P.Geo., Principal, Geology, MMTS
- Mrs. Sue Bird, P.Eng., Principal, Resource Engineering, MMTS
- Mr. Michael A Petrina, P.Eng., Principal, Mine Engineering, MMTS
- Mr. Tracey Meintjes, P.Eng., Principal, Metallurgist, MMTS

2.3 Site Visits and Scope of Personal Inspections

Mr. Robert J. Morris of MMTS conducted a site visit and detailed examination of the property August 22 through to August 23, 2006. During the site visit, sufficient opportunity was available to examine core logging procedures, drill core from the 2006 program as well as conduct a general overview of the property, including selected drill sites, historic core, an underground tour, and the condition of existing project infrastructure. Based on his experience, qualifications and review of the site and resulting data, the author, Mr. Morris, is of the opinion that the programs have been conducted in a professional manner and the quality of data and information produced from the efforts meet or exceed acceptable industry standards. It is also believed that for the most part, the work has been directed or supervised by individuals who would fit the definition of a Qualified Person in their particular areas of responsibility as set out by the Instrument.

Mr. Michael A Petrina of MMTS conducted two site visits in October and December of 2006. During those visits he was able to inspect most of the underground workings on the Polaris level, and some of the underground workings on the AJ level, in addition to seeing the Polaris shaft. Mr. Petrina also inspected diamond drill core and observed it from a geotechnical perspective. Mr. Petrina supervised the dewatering of the Polaris shaft until the end of his site visit.

The remaining QPs have not conducted a site visit.

There has been no additional exploration activity and there is no new material scientific or technical information about the property between the QP site visits and the filing date of this Technical Report. The QP have checked with the Northwest Regional Office of the British Columbia Ministry of Energy, Mines and Petroleum Resources for any Notice of Work applications related to the project, and no activity has been completed at site since 2007. Since the underground workings and deposit are currently flooded and have been since the last exploration activity was completed in October 2006, a more recent inspection would not increase the confidence in the data being used for the Mineral Resource Estimate or PEA.

3 Reliance on Other Experts

MMTS has relied upon the expertise of Roger Berdusco, B.S.F., R.P.F, Principal, Sustainability at MMTS for the analysis, opinion and advice summarized in Section 20 of the Report, as well as recommendations summarized in Section 26.5 of the Report.

MMTS reviewed the status of Canarc's ownership of the mineral claims which appeared to be in order in February 2019. This status can change and should be re-evaluated as part of any future due diligence reviews.

No other experts were relied upon in the preparation of this Technical Report.

4 Property Description and Location

The New Polaris property consists of a group of 61 contiguous crown grants, and one modified grid claim totaling, 1,196 ha (2,956 acres) located 96 km (60 miles) south of Atlin, BC and 64 km (40 miles) northeast of Juneau, Alaska. Located at approximately 133º37'W Longitude and 58º42'N Latitude, the deposit lies on the eastern flank of the Tulsequah River Valley (Figure 1-1).

The claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canarc Resource Corp. (Canarc), and subject to a 15% net profit interest held by Rembrandt Gold Mines Ltd. (Rembrandt), which Canarc has the right to reduce to 10% by issuing 150,000 shares to Rembrandt. Table 4-1 summarizes the claims and the locations are shown on Figure 4-1. Apart from the W.W.1 claim, the claims are crown granted and are kept in good standing through annual tax payments. The W.W.1 is a modified grid claim. The claim has sufficient work filed on it to keep it in good standing until February 4, 2020. The crown granted claims were legally surveyed in 1937. The mineralized areas are shown on Figure 4-2 and Figure 7-2, which shows the geology of the property on the mineral showings.

The Polaris No. 1, Silver King No. 1, Silver King No. 5, Black Diamond, Lloyd and Ant Fraction crown grants include the surface rights. Surface rights for the remainder of the property lie with the Crown, including the areas covered by the Co-Disposal Facility (CDF) and access road to the CDF, and will need to be obtained from the Province of British Columbia.

Mining of the AB Vein system and to a lesser extent the Y and C veins was carried out during the 1930s to early 1950s. Much of the former infrastructure has been reclaimed. A $249,000 reclamation bond is in place and it is the writer's opinion that this adequately covers the cost of reclaiming the original mill site and infrastructure. Currently there is no legal or regulatory requirement to remove or treat the tailings on the property.

Prior to commencing further exploration on the property, a Notice of Work is required to be submitted to the Mining and Minerals Department of the BC Ministry of Energy and Mines. Work can only commence once approval has been received.

Exploration work carried out in 2006 was covered by:
Mines Act Permit MX-1-208
Approval #06-0100054-0808

Water rights will need to be acquired from the Province of British Columbia for sources of water for mining, processing and potable water during operations. Since a positive water balance prevails in the area, and plans include maximizing water recycling to minimize fresh water requirements, such approvals are generally granted, subject to acceptable conditions.

To the extent known, there are no other significant factors and risks that may affect access, title or right, or ability to perform proposed work on the Project.

Table 4-1 List of Claims

Claim Name	Lot No.	Folio No.		Claim Name	Lot No.	Folio No.
Polaris No. 1	6109	4472		Snow	3497	4545
Polaris No. 2	6140	5223		Snow No. 2	3495	5088
Polaris No. 3	6141	5223		Snow No. 3	3494	5495
Polaris No. 4	3498	4545		Snow No. 4	3499	5495
Polaris No. 5	6143	5223		Snow No. 5	6105	4472
Polaris No. 6	6144	5223		Snow No. 8	6107	4472
Polaris No. 7	6145	5223		Snow No. 7	3500	4472
Polaris No. 8	6146	5223		Snow No. 6	6106	4472
Polaris No. 9	6147	5223		Snow No. 9	6108	4472
Polaris No. 10	6148	5290		Black Diamond	3491	4472
Polaris No. 11	6149	5290		Black Diamond No. 3	6030	4944
Polaris No. 12 Fr	6150	5290		Blue Bird No. 1	5708	4545
Polaris No. 13 Fr	6151	5290		Blue Bird No. 2	5707	4545
Polaris No. 14	6152	5290		Lloyd	6035	5010
Polaris No. 15	6153	5290		Lloyd No. 2	6036	5010
Silver King No. 1	5489	4804		Rand No. 1	6039	5010
Silver King No. 2	5490	4804		Rand No. 2	6040	5010
Silver King No. 3	5493	4804		Minto No. 2	6033	4944
Silver King No. 4	5494	4804		Minto No. 3	6034	4944
Silver King No. 5	5491	4804		Jumbo No. 5	6031	4944
Silver King No. 6	5492	4804		Ready Bullion	6032	4944
Silver King No. 7	5495	4804		Roy	6042	5088
Silver King No. 8	5717	4545		Frances	6041	5010
Silver Queen No. 1	6026	4545		Eve Fraction	6170	5495
Silver Queen No. 2	6027	4545		Eve No. 1 Fraction	6171	5495
Silver Queen No. 3	6028	4944		P.T. Fraction	3493	5495
Silver Queen No. 4	6029	4944		Ant Fraction	3492	5088
Silver Strand No. 1	6037	5010		Atlin Fraction	3496	5088
Silver Strand No. 2	6038	5010		Powder Fraction	6043	5088
F.M. Fraction	6044	5088		Jay Fraction	6045	5088
Par Fraction	6154	5290				

W.W.1 Tenure No. 353540 Issue date February 4, 1997. Expiry date: February 4, 2020.

Figure 4-1 Claim Location Map

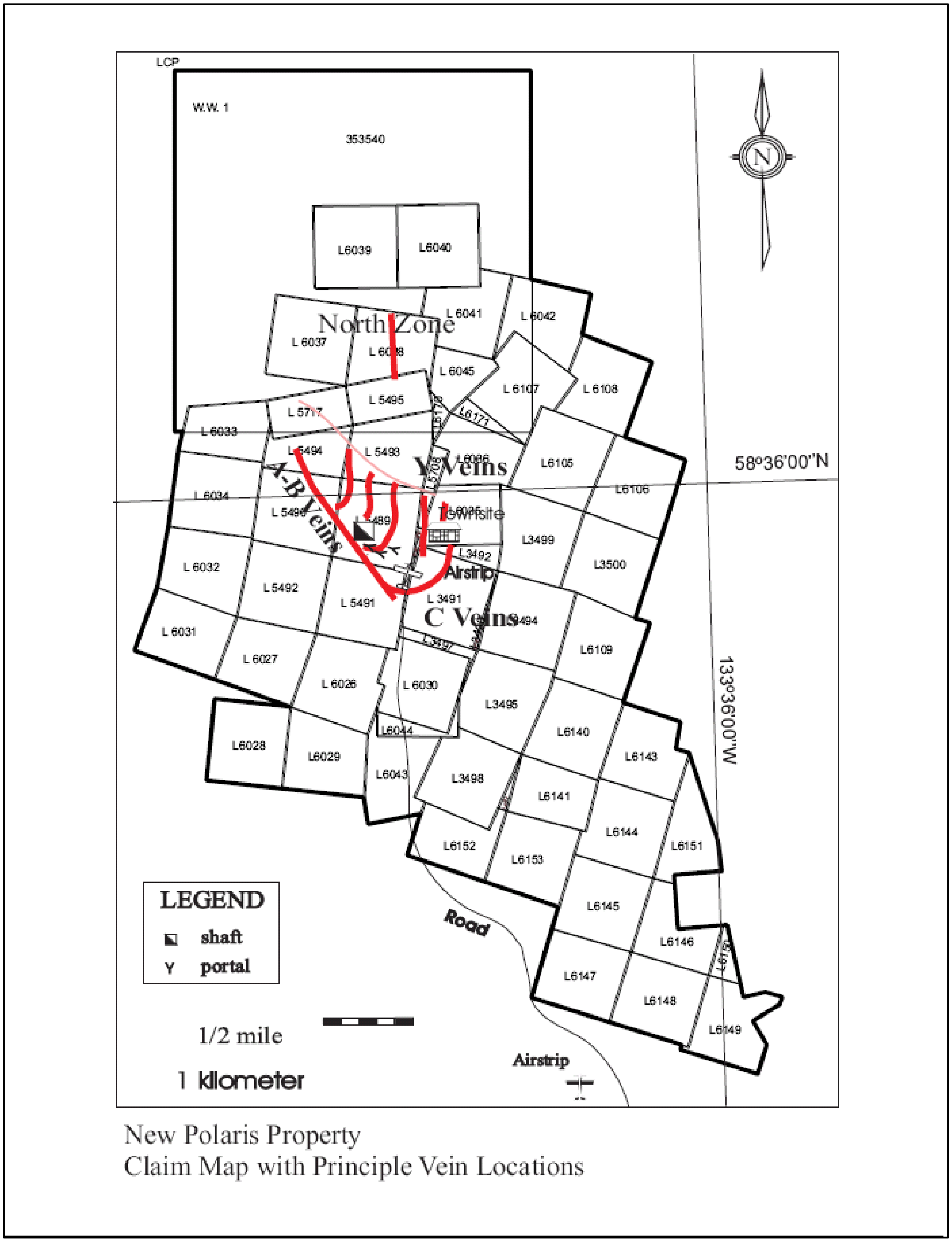

Figure 4-2 **Claim Map with Principle Vein Locations**

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The New Polaris project area lies on the eastern flank of the steep, rugged, Coast Range Mountains, with elevations ranging from the sea level to 2,600 metres.

Extensive recent glaciation was the dominant factor in topographic development. The Taku and Tulsequah Rivers are the most prominent topographic features: broad valleys bounded by steep mountains. Numerous tributary streams flow from valleys filled with glaciers. Most of the glaciers are fingers branching from the extensive Muir ice cap, lying to the northwest of the Taku River. The Tulsequah glacier, which terminates in the Tulsequah valley about 16 km north of the New Polaris mine site, is one of the largest glaciers in the immediate area. It forms a dam causing a large lake in a tributary valley that breaks through the ice barrier (Jakülhlaup) during the spring thaw every year, flooding the Tulsequah and Taku valleys below for three to five days.

Small aircraft provide site access from the nearest population centers in Atlin, BC, 100 km north of the Property, or Juneau, Alaska, 60 km southwest of the Property. A short airstrip for light aircraft exists on the property. The nearest roads in the area terminate 20 km due south of Atlin and 10 km southeast of Juneau. Shallow draft barges have been used in the past to access the site via the Taku River to transport bulk supplies and heavy equipment to site, as well as ship flotation concentration from site. The property can be operated year-round.

The climate is one of heavy rainfalls during the late summer and fall months, and comparatively heavy snowfall, interspersed with rain during the winter. The annual precipitation is approximately 1.5 m of which 0.7 m occurs as rainfall. The snow seldom accumulates to a depth greater than 1.5 m on the level. Winter temperatures are not severe and rarely fall below –15ºC. Summer temperatures, in July, average 10ºC with daytime temperatures reaching the high 20’s on occasion. The vegetation is typical of northern temperature rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley.

There is sufficient land available within the mineral tenure held by Canarc for installations such as the process plant and related mine infrastructure. Surface rights for the areas covered by the CDF, and access road to the CDF, lie with the Crown and will need to be obtained from the Province of British Columbia.

6 History

From 1923 to 1925 the Big Bull and Tulsequah Chief properties were discovered along the east side of the Tulsequah River and opened up the Taku River district. In 1930, Noah A. Timmins Corporation optioned some of the claims that make up the New Polaris property and conducted trenching and diamond drilling in 1931. The trenching exposed several veins, of which 10 showed promising grades. A short exploration adit about 9 m long (30 feet) was also driven into the side of the hill and Timmins drilled 19 holes for a total of 1,615 m (5,297 feet) but was unable to correlate the intersections and elected to drop the option in September 1932.

The Alaska Juneau Gold Mining Company then optioned the property and conducted underground exploration from the "AJ" (Alaska Juneau) adit. Alaska Juneau drove a total of 190 m of drifting (625 feet) and, although they intersected "ore grade" mineralization, they too had problems with correlation and dropped the property in the fall of 1934.

H. Townsend and M.H. Gidel of the Anaconda Corporation examined the property in 1934, carefully mapping the showings. They concluded that commercial ore bodies existed even though these showed irregularity due to faulting. Samples were sent to Geo G Griswold in Butte, Montana, who obtained gold recoveries from flotation tests in the order of 88%.

D.C. Sharpstone then secured an option on the property on behalf of Edward C. Congdon and Associates of Duluth, Minnesota. Congdon conducted 236 m (775 feet) of underground exploration in the "AJ" tunnel and collared 26 m (85 feet) into the Canyon adit. The Polaris-Taku Mining Company was then incorporated in 1936 to take over the property from Congdon. Polaris-Taku erected a 150-ton per day flotation mill in 1937 and mined underground continuously until it was closed down in April 1942 due to labor restrictions brought on the Second World War. Mining operations resumed in April 1946 and continued until 1951 when the mine was closed due to high operating costs, a fixed gold price, and the sinking of a concentrate barge shipment during a storm in March 1951. Up to date, 231,604 oz. of gold was produced at a recovered grade of 0.3 oz/ton.

An Edwards Roaster and a cyanide plant to produce bullion were installed and tested in 1949 in order to improve recovery and reduce shipping cost of concentrates to the Tacoma smelter. The addition of the roaster helped improve milling economics, but its capacity was somewhat limited as it could treat only about 45% of the concentrates produced from the flotation plant. After closure, the mill was leased to Tulsequah Mines Ltd. (owned by Cominco) who modified it to process 600 tpd of massive sulphide polymetallic ore (containing gold, silver, copper, lead and zinc) from the Tulsequah Chief and Big Bull Mines. Tulsequah Mines Ltd. used the mill from 1953 to 1957.

Numalake Mines acquired the property in 1953, changed their name to New Taku Mines Ltd. (New Taku) and undertook rehabilitation work of the mine's plant. A negative feasibility study in 1973 halted this work. New Taku changed its name to Rembrandt Gold Mines Ltd. in 1974. The property lay idle until Suntac Minerals Corp. (Suntac) optioned the property in 1988 and started surface exploration. Canarc merged with Suntac in 1992 and acquired a 100% interest from Rembrandt in 1994, subject to a 15% net profit interest, which Canarc can reduce to 10%. Canarc's subsidiary, New Polaris Gold Mines Ltd. (formerly Golden Angus Mines Ltd.), currently operates the property.

Exploration restarted on the Property in 1988. During the period 1988 to the end of 2006, a total of 74,228 m in 292 holes were drilled on the AB, C and Y vein systems. Individual annual metres are provided in Table 6-1.

Table 6-1 Summary of Exploration Drilling to 2006

Year	Zone	No. of Holes	Metres
1988	Y vein	8	1028
1989	Y vein	19	4078
1990	C vein	10	2862
1991	Y & C veins	11	3333
1992	Y& C veins	23	6378
1993	C vein	8	1301
1994	C & Y veins, North Zone	30	5235
1995	North Zone	20	7600
1996	Underground	24	3205
1997	Underground	49	8869
1998	No drilling	0	0
1999	No drilling	0	0
2000	No Drilling	0	0
2001	No drilling	0	0
2002	No drilling	0	0
2003	C & AB veins	3	1530
2004	C vein	7	1651
2005	C vein	8	2357
2006	C vein	72	24801
Total		**292**	**74,228 m**

A general distribution of this drilling can be seen in Figure 7-3. Initial efforts were confined to the lower elevations of the property due to limited availability of road building equipment and were designed to test the "Y" Vein system either down dip or along strike from old workings. Discovery of the "C" Vein system in 1989 resulted in a refocusing of efforts towards defining this Zone. Drilling during 1994 and 1995 has been designed to test the North Zone and the downward continuity of the "C" Zone. Drilling on the North Zone cut low-grade gold mineralization in a gently dipping shear zone. Drilling at 60 m (200 foot) centres showed the mineralization to be of limited extent and bounded down dip by a post mineralization fault. No additional drilling of the North Zone is warranted.

Diamond drilling from underground workings in 1996 was focused from the AJ level and targeted both the AB and Y vein systems. This work showed that the AB system did not continue to depth and appears at its south east end to bend from a south east strike to an easterly strike direction and become part of the C vein system. As there appears to be little potential for significant additional mineralization on the AB vein system, little exploration of the AB vein has been carried out since 1997.

Diamond drilling from underground workings in 1997, was focused from the AJ, Polaris and 150 levels and targeted the AB, Y, and C vein systems. Due to the location of the workings relative to the orientation of the veins, many of the holes were drilled sub parallel to the dip and strike of the veins. For this reason, since 1997 drilling has been carried out from surface to allow holes to test the veins obliquely to strike and dip.

Drilling to the end of 1997 identified the C vein system as having the most potential for extensive gold mineralization with gold grades and thicknesses comparable to that mined in the 1930s to early 1950s. Mineralization was encountered in drillholes over a 250 m by 300 m area, which remained open to depth. Although the mineralization appears to be continuous between drillholes, the spacing between vein pierce points was too great to give the confidence to calculate a resource. Drilling from 2003 to 2005 focused on closing the drillhole spacing in order to determine the continuity of the grade and thickness of the C vein system.

Drilling to the end of 1997 on the Y vein indicates they are relatively narrow and less continuous along strike than the C veins. Gold grades are comparable to the C vein and these veins have remaining potential for the discovery of additional gold mineralization at depth. Further drilling is required to prove the continuity, gold grades and thicknesses of the veins. The smaller size potential of the Y vein system makes it a second order priority for future drilling.

Drilling has also confirmed the existence of a new "North" Zone, which, although it appears to be low grade (5.6 g/t gold), has exhibited possible significant widths in the order of 6.5 m.

6.1 Historical Resource Estimates

Since the closure of the Taku Polaris Mine in 1951, several resource and reserve estimations have been made with the goal of identifying the probable order of magnitude of "reserves" that may be defined over time. **None of these estimations meet the definition requirements of NI 43 – 101 for a resource or reserve.** The terms resource and reserve are used below, as they were defined at the time, but they do not meet the definition of resources or reserves using today’s CIM standards.

An estimate of Polaris-Taku “reserves” was made prior to closure in 1951 based on stringent precepts. “Reasonably Assured" ore was projected 25 feet (7.6 m) in the plane of the vein above and below sampled drift sections of minable grade while "possible" ore was projected an additional 25 feet (7.6 m) beyond these confines (Parliament 1949). These “reserves” were apparently based solely on underground sampling without using underground diamond drill intercepts (WGM 1992).

Adtec Mining Consultants (1972) re-estimated these “reserves” in contemplation of reopening the mine. Based on similar definitions and existing mine drawings and assay plans, Adtec Consultants (1983) re-estimated the remaining "reserves" within the mine workings using a minimum mining width of 4 feet (1.2 m).

Beacon Hill re-estimated these reserves in 1988 for Suntac using a minimum mining width of 5 feet (1.5 m) with similar results. Their “reserve estimate was limited to those areas where continuous sampling data was available along drifts, raises and stope backs, etc. and where it appears that minimal development work would be required to access the reserves". In 1989, Beacon Hill added further “probable and possible mining reserves” from 27 new drillholes completed by Suntac.

Montgomery Consultants were commissioned to conduct a “Geostatistical Study of the Geological Resource” for the Polaris-Taku Deposit in 1991. The estimate, by G.H. Giroux, discounted much of the reserves around the old workings and did not include dilution and minimum mining width provisions. These estimates were based on both old and new drilling and extended the resource base down to roughly 1,200 feet (366 m) BSL.

Watts, Griffis, and McQuat were contracted to review the previous “reserves” in August 1992. Their review incorporated the residual reserves within the mine workings, as estimated by Beacon Hill in 1989, into their overall estimate. Their estimations were based upon a minimum mining width of 5 feet (1.5 m) or 15 % dilution and a cutoff grade of 0.25 oz/ton gold.

Giroux was further contracted to provide “resource” updates throughout 1992 and in February 1995 he re-estimated the “resources” for the newly drilled portions of the "C" Zone. Drilling also confirmed the existence of a new "North" Zone which, although it appears to be relatively low grade (0.18 oz/ton gold) has exhibited possible significant widths in the order of 6.5 m. Most of the C vein “resource” lies above 800 feet (244 m) BSL and within 200 feet (60 m) of the existing shaft bottom. Giroux’s estimates were in situ based on a 0.25 oz/ton gold cutoff and did not include dilution provisions as described below and considered to be relevant as they are based on a significant amount of data and were independently calculated.

Table 6-2 summarizes the variety of estimations identified above by the following: Beacon Hill’s 1988 estimation of residual “reserves” within and around the workings were totaled. To this total, the geostatistical resource estimation of Giroux was added after applying a general dilution factor of 25 % at zero grade to Giroux's figures for the " Y " Zone and 15% at zero grade for the "AB" and "C" Zones. The dilution factors were estimated based on vein characteristics. The "Y" Veins are described as being high grade, but narrow, which makes them prone to high dilution from over-break during mining as well as over mining. The "AB" veins in situ grade, as estimated by Giroux, already contains internal dilution from a parallel dike. To this total, an overall additional dilution of 15 % was added which is appropriate as the "C" vein would not experience much dilution since it is generally thought to be fairly thick. **This estimate does not meet the definition requirements of NI 43 – 101 for a resource or reserve.** The Author has not done sufficient work to classify them as current reserves or resources and is not treating them as current. This estimate therefore should not be relied upon but is included for historical purposes.

Table 6-2 Historic Resource Estimates, 1988 to 1995

Polaris Takus Geostatistical Resources								
	PROBABLE RESOURCES				POSSIBLE RESOURCES			
	In-Situ		Diluted		In-Situ		Diluted	
	Tons	Grade	Tons	Grade	Tons	Grade	Tons	Grade
Zone	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)	(SDI)	(oz/SDT)	(SDI)	(oz/SDI)
GIROUX (1995)								
Y Zone	210,000	0.461	262,500	0.369	987,000	0.469	1,234,000	0.375
AB Zone	78,000	0.403	89,700	0.35	508,000	0.387	584,000	0.337
C Zone	85,700	0.426	98,500	0.37	595,000	0.425	684,000	0.37
Sub-total	373,000	0.441	450,700	0.365	2,090,000	0.437	2,502,000	0.365
BEACON HILL (1988)								
Upper Levels	53,440	0.37	67,800	0.29	41,560	0.35	53,450	0.27
Lower Levels	50,170	0.5	64,410	0.39	45,000	0.48	58,760	0.37
Sub-total	103,610	0.43	132,210	0.33	85,560	0.42	112,210	0.32
TOTAL	**476,610**	**0.439**	**582,910**	**0.359**	**2,175,560**	**0.436**	**2,614,210**	**0.363**

Note: With NI 43-101 guidelines, the terms Probable Resources and Possible Resources have been replaced with Indicated Resources and Inferred Resources.

Subsequent to the above estimates, additional exploration has been done, particularly 90 holes drilled from 2003 to 2006 by Canarc. A revised Mineral Resource Estimate was made by G.H. Giroux and included in the NI 43-101 Technical Report entitled “Resource Potential, New Polaris Project” March 5, 2007, using ordinary kriging of 192 drillholes and 1,432 gold assay intervals constrained within four main vein segments as modeled in 3D by Canarc geologists.

These historic estimates are presented as a series of grade-tonnage tables, reporting all material within undiluted mineralized solids below -95 m elevation in the CWM vein and below -140 m elevation in the CLOE and CHIE veins (the lowest elevations of the historic mine workings). All material below these elevations is based on verified drilling information and does not use any historical information from the old mined out levels and excludes resource calculation previous to March 2007.

Table 6-3 Historic Resource Estimate, 2007

	MEASURED RESOURCE			INDICATED RESOURCE		
Au Cutoff	Tonnes > Cutoff	Grade>Cutoff	Contained Metal	Tonnes > Cutoff	Grade>Cutoff	Contained Metal
(g/t)	(tonnes)	Au (g/t)	Au (ozs)	(tonnes)	Au (g/t)	Au (ozs)
6.00	271,000	11.89	104,000	1,017,000	12.71	416,000
7.00	233,000	12.77	96,000	910,000	13.45	393,000
8.00	203,000	13.54	88,000	806,000	14.22	368,000
9.00	173,000	14.42	80,000	696,000	15.11	338,000
	MEASURED PLUS INDICATED RESOURCE			INFERRED RESOURCE		
Au Cutoff	Tonnes > Cutoff	Grade>Cutoff	Contained Metal	Tonnes > Cutoff	Grade>Cutoff	Contained Metal
(g/t)	(tonnes)	Au (g/t)	Au (ozs)	(tonnes)	Au (g/t)	Au (ozs)
6.00	1,288,000	12.54	519,000	1,628,000	12.15	636,000
7.00	1,143,000	13.31	489,000	1,473,000	12.74	603,000
8.00	1,009,000	14.08	457,000	1,340,000	13.27	571,000
9.00	870,000	14.97	419,000	1,149,000	14.06	519,000

Technical Reports from October 4, 2007, December 23, 2009, and April 10, 2011 entitled "New Polaris Project - Preliminary Assessment" contain the same resource estimates as described above and are based on the March 5, 2007 Technical Report.

7 Geological Setting and Mineralization

7.1 Regional Geology

The New Polaris Mine lies on the western edge of a large body of Upper Triassic Stuhini Group volcanic rocks, which has been intruded by a Jurassic-Cretaceous granodiorite body north of the mine. Older Triassic volcanic rocks and earlier sediments underlie the Stuhini volcanic rocks. The granodiorite is part of the Coast Plutonic Complex (Figure 7-1).

The structural trend in the area is northwest-southeast, paralleling major faults and folds to the east and intrusive alignment to the west. The Triassic volcanic rocks and older sedimentary rocks have been folded and sheared with the Stuhini Group rocks being deformed into broad to isoclinal, doubly plunging symmetrical folds with large amplitudes.

7.2 Property Geology

Canarc has carried out extensive mapping of the Polaris-Taku property since the early 1990's. The work has been done by several employees and contractors and is shown in Figure 7-2. The gold deposit is hosted within an assemblage of mafic (basalt and andesite units) volcanic rocks altered to greenschist metamorphic facies. The orientation of these units is inconclusive because there are no marker beds in the sequence. It is thought that the units are steeply dipping (70º to 80º) to the north based on the orientation of the limestone/basalt interface at the southern portion of the property.

A serpentinite unit is located to the northeast, which was identified in recent (1996/97) drilling and underground mapping. This unit appears to form the eastern extent of the mineralization. The age relationship is unclear, but it is assumed that the serpentinite is a later stage feature possibly associated with tectonism in the area.

The 'vein' zones are structurally controlled shear zones and are typified by silicification and carbonatization cross cutting actual quartz-carbonate veins. These zones have sharp contacts with the wall rock and form anastamosing ribbons and dilations. These zones have been deformed several times, which makes original textures difficult to determine. The zones are generally tabular in geometry forming en-echelon sheets within the more competent host lithologies.

All of the strata within the property have been subjected to compression, rotation and subsequent extension. The plunge of folds appears to be variable though generally shallow. Small-scale isoclinal folds strike north-northwesterly and plunge moderately to the north. Numerous faults are found on the property, the more significant of which are discussed below.

The possible extension of the Llewellyn fault, termed the South Llewellyn fault, continues south from the Chief Cross fault along mine grid coordinate 4400 East. Slightly north of Whitewater Creek it is offset to the west by an east-west fault, the 101 fault, to continue in a more southeast orientation of the opposite side of Whitewater Creek. This northwest-southeast orientation structure was named the Limestone Fault due to its bedding parallel attitude within a discontinuous limestone/marble horizon. It marks the southwest boundary of the "mine wedge": the wedge-shaped package of rock within which all past production took place. The northern boundary of the "mine wedge" is further defined as mentioned above by the Whitewater Creek Schist Zone, a zone of schistose chlorite-amphibolite-serpentinite less than 100 m thick. A complex network of brittle faults is also found within this zone.

Three major faults, Numbers 1 and 5, and an unnamed fault, lie within the mine wedge. The No.1 and No.5 faults strike northwest-southeast, dipping approximately 45º to the northeast, and are sub-parallel to the unnamed fault, which dips steeply to the southwest. The No.1 fault has reverse displacement of up to 30 m while the displacement of the No.5 fault is poorly defined. The southwest dipping, unnamed fault showed no displacement, as it apparently parallels the A-B vein system. The mined-out areas indicate the wedge shape, the predominant orientations and continuity of the zones, and the overall plunge of the system to the southeast. An early interpretation of the structure showed that various veins appear to meet and form "junction arcs" where both thickness and grade improve.

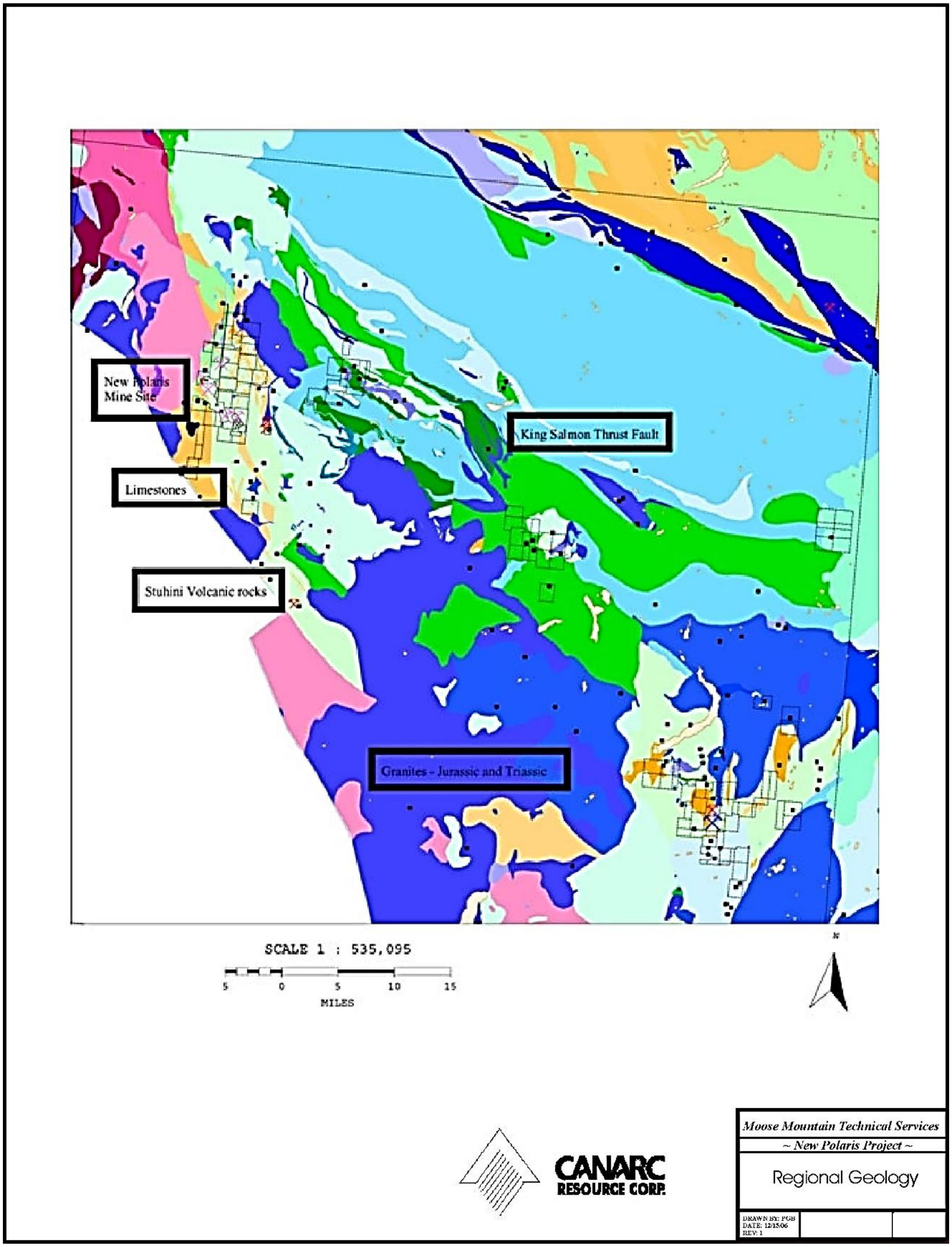

Figure 7-1 **Regional Geology**

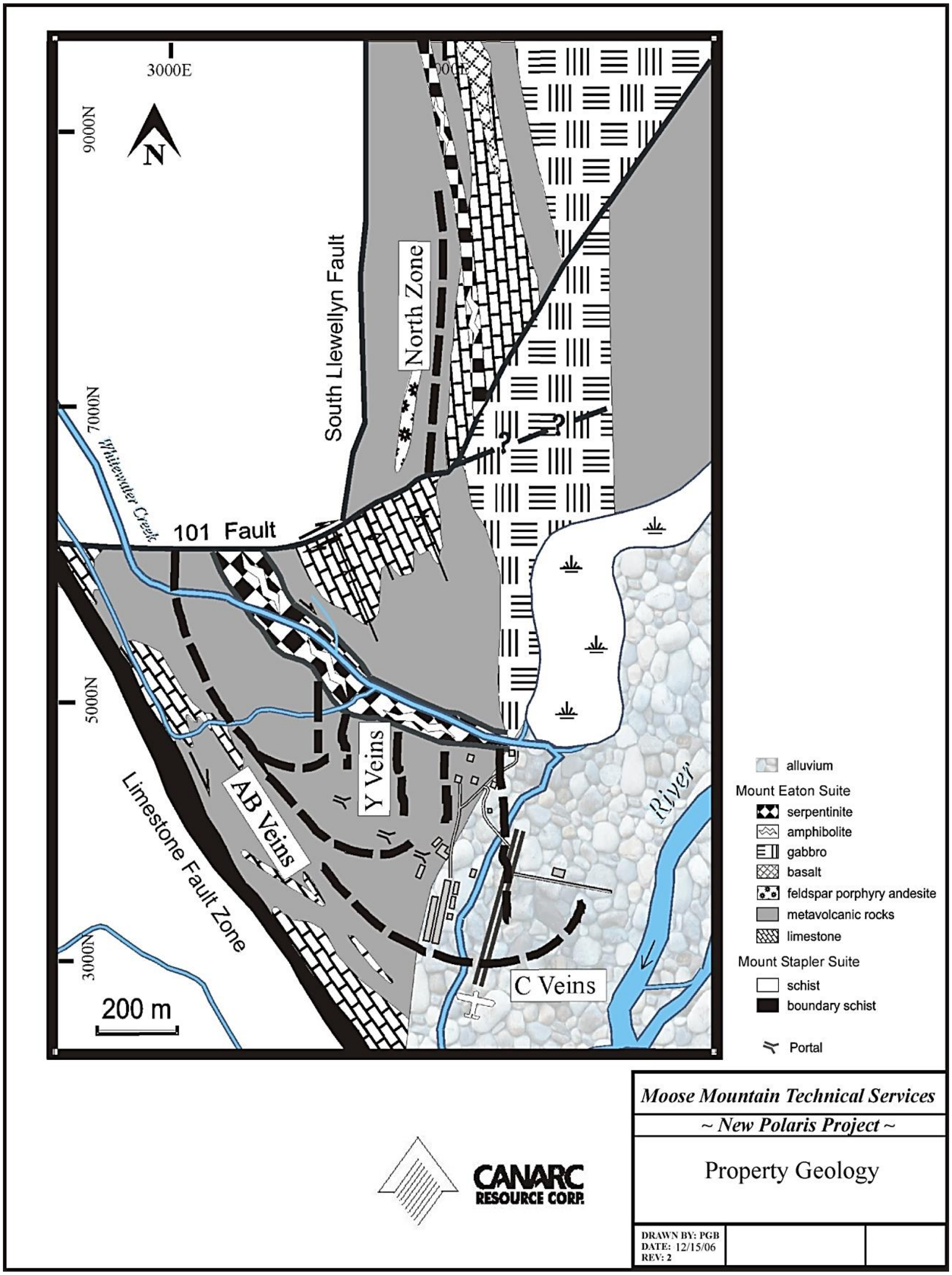

Figure 7-2 Property Geology

7.3 Mineralization

Mineralization of the New Polaris deposit bears strong similarities to many Archean lode gold deposits such as the arsenical gold camp of Red Lake, Ontario where the gold-bearing arsenopyrite is disseminated in the altered rock and in quartz-carbonate stringers.

The vein mineralization consists of arsenopyrite, pyrite, stibnite and gold in a gangue of quartz and carbonates. The sulphide content is up to 10% with arsenopyrite the most abundant and pyrite the next important. Stibnite is fairly abundant in some specimens but overall comprises less than one-tenth of 1% of the vein matter. Alteration minerals include fuchsite, silica, pyrite, sericite, carbonate and albite.

In general, the zones of mineralization ranging from 15 to 250 m in length with widths up to 14 m appear to have been deposited only on the larger and stronger shears. Their walls pinch and swell showing considerable irregularity both vertically and horizontally. Gold values in the veins have remarkable continuity and uniformity and are usually directly associated with the amount of arsenopyrite present. The prominent strike directions are north-south and northwest-southeast, which is interpreted to be within a major shear zone. Up to 80% of the mine production was from "structural knots" or what is now known as "C" zones. In detail the "C" zones are arcuate structures. Figure 7-3 shows a 3D view of the "C" vein system.

The vein mineralization has well marked contacts with the wall rock. The transition from mineralized to non-mineralized rock occurs over a few centimeters. The mineralization consists of at least three stages of quartz veining. The initial stage of quartz-ankerite introduced into the structure was accompanied by a pervasive hydrothermal alteration of the immediately surrounding wall rock. Arsenopyrite, pyrite and lesser stibnite were deposited with the alteration. Later stages of quartz-ankerite veining are barren and have the effect of diluting the gold grades in the structure. The sulphide minerals are very fine-grained and disseminated in both the wall rock and early quartz and ankerite veins. Free gold is extremely rare and to the end of 2005 had not been recognized in core samples. The majority of the gold occurs in arsenopyrite and to a lesser extent in pyrite and stibnite. Because there is no visible gold and the host sulphides are very fine-grained and disseminated there is little nugget effect and gold values even over short intervals rarely exceed 1 oz/ton.

Mineralization was observed by Morris during the site visit both in drill core and underground. The description of the regional setting, local geology, and mineralization appears applicable to the New Polaris project and is sufficiently well understood to support the estimation of Mineral Resources.

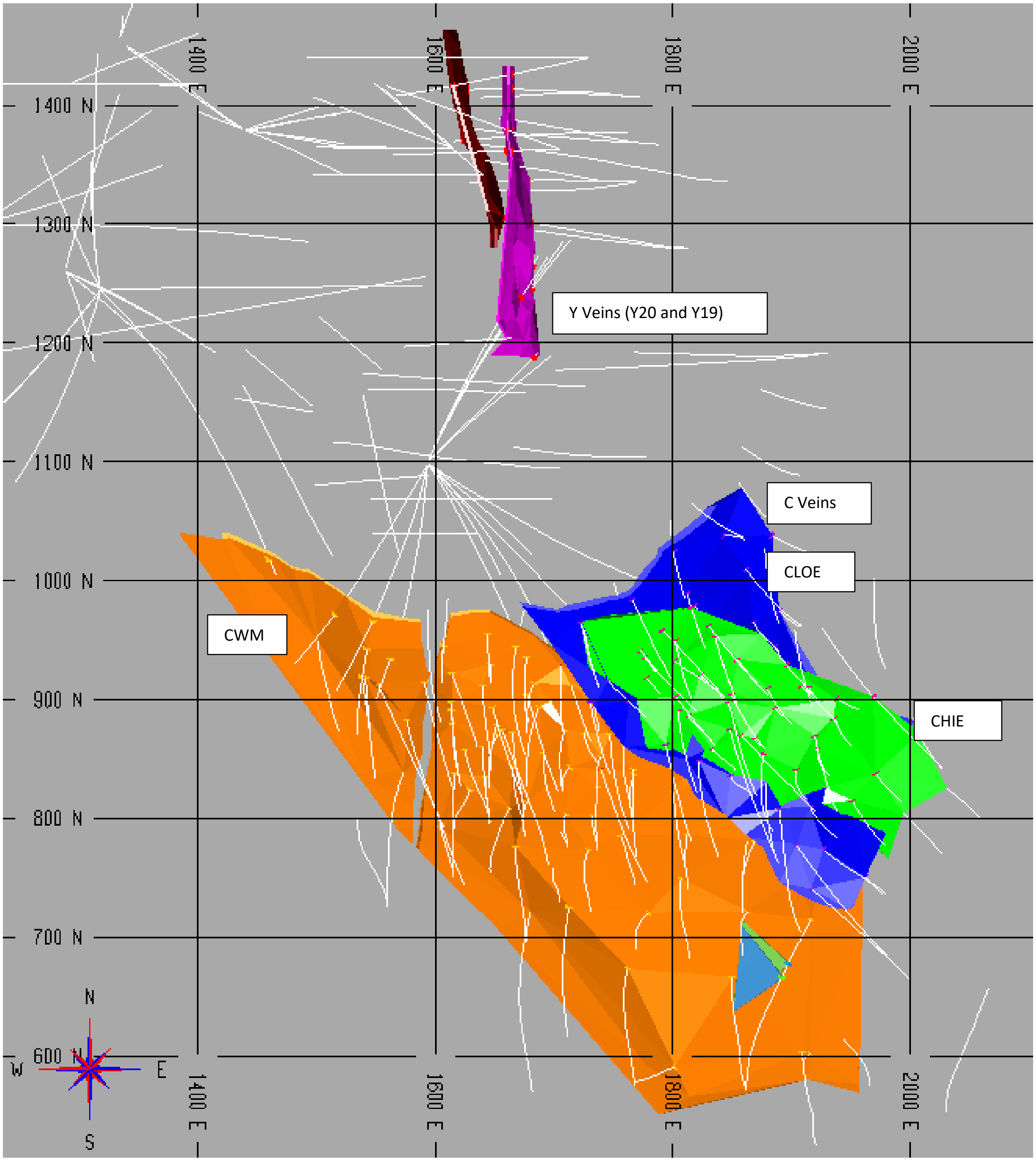

Figure 7-3 3D Model C Vein; plan view in mine specific space

(Note: Shows only drillholes since 1988)

8 Deposit Types

The New Polaris deposit is classified as a mesothermal lode-gold deposit (Hodgson, 1993).

In general, it is quartz-vein-related, with associated carbonatized wall rocks. The deposits are characterized by a high gold/silver ratio, great vertical continuity with little vertical zonation, and a broadly syn-tectonic time of emplacement. They are commonly associated with pyrite, arsenopyrite, tourmaline and molybdenite. Mineralization may occur in any rock type and ranges in form from veins, to veinlet systems, to disseminated replacement zones. Most mineralized zones are hosted by and always related to steeply dipping reverse- or oblique-slip brittle-fracture to ductile-shear zones.

The exploration target on the New Polaris project is orogenic lode gold deposits also known as Mesothermal vein deposits. Numerous examples of this type of deposit are known throughout the work including the Campbell Red Lake deposits in Ontario and the Bralorne deposit in British Columbia. Past exploration studies have demonstrated that the New Polaris vein systems have all the attributes of the orogenic vein gold deposit including, but not limited to association with major structural break, quartz-carbonate vein association, low-sulphide assemblage of pyrite and arsenopyrite, chloritic and sericitically altered wall rocks and persistent gold mineralization over a vertical distance of nearly 1 km.

The deposit type and model are considered by the QP as appropriate for a Mesothermal lode-gold deposit.

9 Exploration

The New Polaris property represents an advanced exploration project on a former gold producer. The early exploration work in the area located gold mineralization on surface and subsequent exploration led to mining of approximately 689,500 tonnes of material grading 10 g/t gold. More recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones.

Geological mapping, geochemical surveys, geophysical techniques, and drilling have added considerable value to the project. Table 9-1 lists the relevant exploration work on the property along with contractor name and supervisor.

Surface mapping, geochemistry and geophysics over the “mine wedge” were completed by Orequest in 1988. Surface mapping and geochemistry, on the “north grid”, were completed in 1993.

Underground exploration included the rehabilitation of the AJ Level in 1988 and the rehabilitation of all the other levels, including the Polaris Portal, in 1996 and 1997. The underground rehabilitation also included a re-survey of the old workings so that the more recent surface work could be aligned with the old underground workings.

Only drillhole data has been used in the current resource estimate. In total the database includes 1,056 drillholes with a total of 31,514 samples, of which 1,432 are within the mineralized zones (see Section 14 for more details).

The procedures followed in the field and through the interpretation stage of exploration have been professional and are appropriate to the style of mineralization and current degree of geological knowledge and understanding of mineralization control.

Various crews under the supervision of professional geologists carried out the exploration work. It is considered that the reliability of the data obtained with exploration is very high.

Table 9-1 Exploration Employees / Contractors

Year	Supervisor	Drilling Contractors	Laboratory	Underground
1988	Cloutier		TSL	
1989	Cloutier		TSL	
1990	Cloutier		TSL	
1991	Marriott	Arctic	Min En/Chemex	
1992	Marriott	Arctic	Chemex	
1993	Marriott/Moors	Arctic	Chemex	
1994	Moors	Arctic/Falcon	Chemex	
1995	Moors	Arctic/Falcon	Chemex	
1996	Karelse/Watkinson	Advanced	Northern	Main Street
1997	Karelse/Watkinson	Advanced	Northern	Main Street
1998	-			
1999	-			
2000	-			
2001	-			
2002	Moors			
2003	Moors	Hy-Tech	ALS Chemex	
2004	Moors/Aspinall	Hy-Tech	ALS Chemex	
2005	Moors/Aspinall	Hy-Tech	ALS Chemex	
2006	Moors/Cote	Hy-Tech	ALS Chemex	

10 Drilling

Diamond drill programs were carried out on the New Polaris Project when the project was reactivated in 1988 until 2006 (Table 10-1). Initially, the drilling focused on the down dip and along strike extensions of the Y veins. This work showed that the Y veins, while good grade were narrow and less continuous than the AB vein system. It also showed that the Y vein system is comprised of about 12 separate veins all of which are narrow and of short strike length.

In 1990, drilling shifted to the area beneath the lowest most C vein stopes. This drilling found that the vein system continued to depth and that gold grades in the 0.30 to 0.45 oz/ton range over an average true thickness of 3 m were present. From 1991 to 1993 most of the drillholes tested the C veins with fewer drilled on the Y vein system.

In 1994, the North Zone was discovered and was tested with a total of 30 drillholes during the 1994 and 1995 period. Although thicknesses of the North Zone are up to 6.7 m, the grades are relatively low compared to the C vein (less than 0.2 oz/ton). This combined with the limited extent due to structural termination of the zone by a fault resulted in a decision to terminate exploration of the North Zone.

Encouraging drill results from the C veins and to a lesser extent from the Y vein system led to further drilling on these two vein systems. Drilling on the C vein showed the veins to be open to depth and to have gold grades that ranged from 0.2 to 0.6 oz/ton over true thicknesses of 3 m. The increased interest in the C vein system was due to its greater continuity and thickness compared to the Y vein. The narrow width and lesser continuity of the Y vein system made it a secondary exploration target.

In 1996 and 1997 the Y, C and AB veins were explored from underground. The plan was to closely test the upper portions of the Y, C and AB veins in order to allow calculation of a resource that might form the basis for resumption of mining. The results of the underground drilling program were mixed. The underground workings were for the most part driven along the vein structures with few crosscuts from which holes could be drilled to cut the down dip and along strike extension of the veins. As a result, except for those holes that tested the area immediately below the workings, most cut the veins at shallow angles. The very shallow angles that in places approach parallel to the vein make the use of these intersections inappropriate for a resource calculation (An example is hole 97-44 that cut 34.1 m grading 0.42 oz/ton). Despite the number of holes drilled during 1996 and 1997, the work did little to expand the extent of the mineralization in the AB, C or Y vein systems. The work did confirm that the mineralized shoots in the lower most stopes on the Y and C veins were open to depth. The Table below summarizes the locations of the 1988 to 2005 drilling. Composites of assay results are listed for the C vein system holes that intersected significant mineralization.

Poor market conditions after 1997 made financing of the New Polaris Project difficult. Drilling restarted on the property in 2003 with the objective of testing the extent of the C vein mineralization.

Godfrey Walton, P. Geo., at the request of Canarc, undertook a review of the Polaris Project and recommended additional drilling in order to test the continuity of the “C” vein zone mineralization at depth below the lower most mine workings. To this end, limited drill programs were carried in 2003 to

2006. If successful, these programs would lead to the expanded drill program, to allow a higher confidence level for the mineral resource estimation, as recommended as a second phase by Walton.

The 2003 to 2006 exploration programs targeted the “C” vein extensions below the existing mine workings. Collar locations for the holes drilled during the 2003 to 2006 programs, as well as relevant holes from earlier drilling are plotted on Figure 10-2. Representative plan and section plots showing the pierce points of drillholes and grades of the 1988 to 2006 drilling is presented in Figure 10-2 to Figure 10-3. Cross-sections of the C vein to assist the reader in understanding the attitude, grades and thickness of the C vein system are presented in Figure 10-4 to Figure 10-6.

The results of the 2003 to 2006 drilling of the C vein system confirmed the continuity of gold mineralization and the vein structure between the earlier drilled holes. As can be seen in the sections below, drill results show the C vein system to be an arc-like structure oriented east-west in the west swinging to a northeastern strike in the east. The change in strike occurs across the No.1 fault. To the east of the No.1 fault, the vein splays into two or more branches. The dip of the vein system is to the south and southeast and has an average dip of about 50º, although east of the No.1 fault the vein appears to flatten and thicken in a simoid-like feature. The exact nature of the apparent flattening of the vein’s dip is not clear and requires additional drilling to be resolved.

The thickness of the C veins varies from 0.30 m to a maximum of 15.2 m. The thicker parts of the vein occur to the east of the No. 1 fault where the dip of the vein flattens due to an apparent folding of the vein.

Table 10-2 lists the core length of the vein material cut in the drillhole. Depending upon the angle of intersection, the true thickness ranges from 100% to about 70%. The average core length thickness of the intersections is approximately 4.5 m and the average grade is 14.4 g/t (0.4 oz/ton) gold. The estimated average true thickness of the vein is 3.0 m.

All of the holes in this period were drilled from surface and intersected a similar geologic sequence. From the collar, the holes penetrated from 15.2 m to 79.2 m of overburden followed by inter-layered ash and lapilli tuff, volcanic wacke, and foliated andesite. The C vein system crosscuts the strike of the volcanic and volcaniclastic rocks at steep angles.

Table 10-1 Diamond Drillholes (1988 to 2005)

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
03-P01	1569.72	1021.08	365.46	-70	345
03-P02	1513.33	1021.08	426.72	-70	330
03-P03	1591.06	1255.78	768.71	-60	260
04-1707E1	1706.88	873.25	259.08	-75.4	358.8
04-1707E2	1707.44	873.25	280.11	-83	359
04-1737E1	1737.63	873.16	287.43	-83	359

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
04-1737E2	1737.33	845.82	289.26	-83	320
04-300SW1	1842.61	902.79	224.03	-66.1	320
04-300SW2	1841.57	901.6	246.89	-75	320
04-300SW3	1842.7	902.73	248.41	-83	320
04-330SW1	1785.71	922.54	213.06	-81.7	323.8
04-330SW2	1789.62	904.06	237.44	-83.8	324
04-360SW1	1777.39	884.53	237.44	-75	318.5
04-360SW2	1777.39	884.53	243.84	-83	318
05-1676E1	1676.4	872.49	210.92	-71	359
05-1676E2	1676.4	872.49	240.49	-78	359
05-1676E3	1676.4	872.49	94.49	-83	359
05-300SW4	1875.64	860.29	289.86	-77.5	322
05-300SW5	1877.2	858.36	315.16	-85	318
05-300SW6	1911.06	816.86	347.17	-84.5	320
05-330SW3	1835.64	860.95	277.98	-77	320
05-330SW4	1835.69	860.88	298.4	-83	320
05-330SW5	1853.22	839.92	315.16	-83	320
L52	1465.17	1330.15	248.41	-25	85
L55	1510.89	1361.54	274.32	-17	40
L56	1498.7	1313.69	161.85	-25	92
L57	1464.26	1330.45	215.49	-25	51
L58	1473.4	1278.79	189.28	-25	112
L59	1477.67	1279.25	343.2	-25	95
L62	1463.04	1233.83	306.32	-25	149
L84	1342.64	1237.79	229.21	-2.5	247.5
P-9612	1289.97	1259.92	174.65	-35	119
P8918A	1819.79	889.97	367.89	-70	315
P91C01	1888.57	762.3	322.17	-65	338
P91C02	1888.57	762.3	331.62	-77	338
P91C03	1937.98	805.62	320.04	-75	339
P91C04	1937.98	805.62	324.92	-87	339
P91C05	1827.28	942.78	199.19	-73	339
P91C06	1672.83	818.27	263.96	-71	339
P91C07	1907.53	704.18	320.65	-75	335
P91Y01	1767.03	1369.25	330.1	-66	270
P91Y02	1769.59	1336.46	305.1	-64	266

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
P91Y03	1705.02	1095.33	274.93	-49	268
P91Y04B	1845.38	1336.03	444.09	-68	274
P92C08	1810.12	984.99	199.03	-71	342
P92C09	1889.24	964.2	210.31	-72	340
P92C10	1917.31	894.37	243.84	-76.5	340
P92C11	1978.82	925.86	251.16	-71.5	337
P92C12	1870.25	819	282.55	-71.5	337
P92C13	1635.16	860.39	223.72	-70	345
P92C14	1614.98	823.9	262.13	-70	343
P92C15	1702.92	823.57	258.17	-71.5	338
P92C16	1816	780.29	254.2	-71.5	338
P92C17	1730.41	865.11	264.26	-76	339
P92C18	1606.63	792.6	350.22	-70	337
P92C19	1694.87	761.73	381.3	-70	350
P92C20	1549.57	832.74	267.31	-70	350
P92C20A	1549.57	832.74	273.71	-70	350
P92C21	1534.49	871.36	213.97	-71	358
P92C22	1624.58	743.71	413.61	-71	336
P92C23	1518.98	873.06	274.62	-72	337
P92C24	1798.02	918.21	149.66	-61	331
P92Y05	1769.67	1335.82	346.56	-70	259
P92Y08	1787.35	1367.03	326.96	-67	274
P92Y09	1745.28	1336.55	248.11	-61	269
P93C25	1737.36	911.35	102.41	-65.5	340
P93C26	1858.98	1036.32	144.17	-70	295
P93C27	1860.13	1034.89	166.73	-85	295
P93C28	1882.41	1050.04	165.51	-75	310
P93C29	1906.65	1088.81	179.83	-75	280
P93C30	1928.77	1144.68	168.25	-75	280
P93C31	1815.61	901.17	215.49	-74	335
P94C32	1908.04	962.37	249.02	-72	303
P94C33	1908.37	961.28	279.81	-86	284
P94C34	1937.31	1015.9	258.17	-72	287
P94C35	1937.31	1015.9	339.55	-83	288
P94C36	1957.12	891.54	371.86	-80	288
P94C37	1883.66	789.74	279.5	-74	327

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
P94C38	1699.87	806.2	287.73	-82	348
P94C39	1769.36	768.1	290.17	-77	351
P94N01	1469.75	2124.76	84.73	-55	85
P94N02	1469.75	2124.76	84.73	-85	85
P94N03	1488.95	2194.26	60.35	-55	85
P94N04	1488.95	2194.26	106.07	-90	0
P94N05	1495.96	2258.26	118.26	-56	85
P94N06	1495.96	2258.26	124.36	-87	85
P94N07	1488.34	2332.63	224.94	-55	85
P94N08	1488.34	2332.63	273.71	-85	85
P94N09	1490.17	2438.1	264.57	-50	70
P94N10	1481.33	2438.1	96.93	-85	70
P94N11	1492.3	2499.36	75.59	-50	70
P94N12	1492.3	2499.36	71.02	-85	70
P94N13	1482.85	2384.76	66.45	-50	70
P94N14	1482.85	2384.76	66.45	-85	70
P94N15	1187.2	1822.7	51.21	-60	70
P94N16	1187.2	1822.7	57.3	-60	100
P94Y10	1663.9	1203.96	236.83	-63	266
P94Y11	1813.56	1280.16	251.76	-55	270
P94Y12	1813.56	1280.16	276.15	-63	265
P94Y13	1929.69	1191.46	371.86	-62	265
P94Y14	1929.69	1191.46	400.81	-55	262
P95C40	1965.96	575.77	786.69	-82.5	355
P95C41	2029.97	553.21	769.01	-80	360
P95C42	1861.72	571.5	673	-77.5	0
P95C43	1861.72	571.5	751.94	-80	340
P95C44	1920.24	525.78	792.48	-83	340
P95N17	1347.22	2122.93	264.57	-69.5	85
P95N18	1370.69	2183.89	270.66	-72	85
P95N19	1350.26	2243.33	315.47	-68	85
P95N20	1348.74	2302.76	335.28	-70	85
P95N21	1348.13	2329.59	215.8	-70	70
P95N22	1353.92	2387.8	227.99	-72	70
P95N23	1338.07	2449.07	261.52	-68	70
P95N24	1336.24	2449.07	294.74	-85	70

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
P95N25	1502.66	2804.16	255.42	-55	100
P95N26	1325.88	2804.16	273.71	-80	100
P95N27	1325.88	2804.16	334.67	-80	80
P9705A	1055.52	1545.03	231.34	-21	52
PC25A	1738.27	909.83	159.41	-68	336
PC44A	1920.24	525.78	749.81	-83	340
PC8801	1554.16	1178.32	106.68	-50	310
PC8802	1554.16	1178.32	38.71	-90	0
PC8803	1496.69	1146.74	133.2	-60	310
PC8804	1496.21	1141.91	119.48	-60	280
PC8805	1614.15	1363.59	149.35	-45	271
PC8806	1614.68	1363.74	189.28	-60	270
PC8807	1614.14	1364.2	180.14	-48	290
PC8808	1614.26	1365.03	110.95	-48	310
PC8901	1573.08	1149.96	142.34	-90	0
PC8902	1594.63	1088.77	128.32	-86	275
PC8903	1593.56	1088.51	167.64	-55	265
PC8904	1546.21	1068.5	50.9	-52	90
PC8904A	1546.21	1068.5	230.73	-52	90
PC8905	1546.73	1039.04	167.94	-60	90
PC8906	1705.42	1363.04	220.98	-55	268
PC8907	1704.52	1363.2	236.83	-70	290
PC8908	1704.52	1363.2	243.23	-50	290
PC8909	1639.71	1341.13	185.32	-45	270
PC8910	1639.14	1341.19	126.34	-59	270
PC8911	1639.77	1342.07	187.45	-53	310
PC8912	1726.56	1440.13	307.85	-45	270
PC8913	1727.76	1440.48	244.75	-60	270
PC8914	1726.95	1439.69	253.59	-56	256
PC8915	1726.18	1163.72	413.31	-65	270
PC8916	1762.08	1377.37	233.78	-55	270
PC8917	1676.68	1157.69	171.3	-45	270
PC8918	1819.79	889.97	191.11	-70	315
PC8919	1777.26	1103.56	219.15	-60	270
PC9001	1820.17	822.27	400.51	-70	340
PC9002	1838.2	767.83	388.92	-70	340

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
PC9003	1746.57	837.7	305.1	-70	340
PC9004	1853.47	897.63	305.71	-70	340
PC9004A	1853.47	897.63	108.91	-70	340
PC9005	1675.88	852.91	265.48	-60	340
PC9006	1762.29	792.22	404.16	-70	340
PC9007	1605.02	880.79	242.62	-68	340
PC9008	1778.45	746.71	441.05	-70	330
PT23A	1317.96	1244.5	206.65	-32	195
PT29A	1317.96	1244.5	325.53	-31	134
PT30A	1317.96	1244.5	279.81	-15	90
PT31A	1334.41	1456.94	406.3	0	20
PT37B	1595.05	1102	179.53	-52	45
PT38A	1595.32	1097.49	260.3	-35	124
PT9601	1444.26	1379.06	106.07	-36	79
PT9602	1441.21	1379.06	120.4	-51	79
PT9603	1441.22	1378.72	102.11	-36	96
PT9604	1441.22	1378.72	130.76	-51	96
PT9605	1441.09	1379.36	101.8	-49	68
PT9606	1441.33	1378.29	115.82	-38	110
PT9607	1436.92	1380.33	43.59	21.5	291
PT9608	1436.92	1380.62	17.98	21.5	299
PT9609	1437.02	1380.72	119.48	21.5	304
PT9610	1437.02	1380.73	119.48	-4.5	304
PT9611	1436.91	1380.62	125.88	-22	299
PT9613	1289.76	1259.71	149.05	-41	134
PT9614	1289.43	1258.88	121.01	-52	164
PT9615	1289.76	1259.71	147.52	-52	134
PT9616	1289.97	1259.92	176.78	-46	119
PT9617	1288.85	1263.64	152.1	0	15
PT9618	1055.52	1545.03	146	18	88
PT9619	1055.52	1545.03	127.41	18	108
PT9620	1055.52	1545.03	173.74	-10	118
PT9621	1055.52	1545.03	141.43	-8.5	80
PT9622	1055.52	1545.03	152.1	22	108
PT9623	1055.52	1545.03	167.34	-14	110
PT9624	1055.52	1545.03	272.19	-10	52

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
PT9701	1055.52	1545.03	9.75	-5	52
PT9702	1055.52	1545.03	287.73	-7	52
PT9703	1055.52	1545.03	220.07	-13	60
PT9704	1055.52	1545.03	205.44	-10	60
PT9706	1055.52	1545.03	255.42	-9.5	49
PT9707	1055.52	1545.03	240.18	10	52
PT9708	1049.12	1542.59	144.48	0	242
PT9709	1217.37	1301.04	69.8	-22	0
PT9710	1217.98	1301.19	69.49	-22	340
PT9711	1218.9	1301.04	163.37	-15.5	78
PT9712	1218.9	1301.04	171.3	4	60
PT9713	1222.1	1299.97	281.33	-15	90
PT9714	1219.2	1297.84	47.61	-42	0
PT9715	1218.9	1301.04	59.74	-46	330
PT9716	1217.07	1296.62	135.33	-15	270
PT9717	1217.07	1296.62	154.84	-30	270
PT9718	1217.07	1296.62	118.26	-18	249
PT9719	1290.83	1417.62	112.47	-5	90
PT9720	1290.83	1417.62	133.81	-35	90
PT9721	1290.52	1417.62	153.92	-44	71
PT9722	1289.3	1418.23	219.15	-5	270
PT9723	1317.96	1244.5	17.98	-32	195
PT9724	1317.96	1244.5	192.94	-30	217
PT9725	1317.96	1244.5	244.14	-47	218
PT9726	1317.96	1244.5	124.66	-10	334
PT9727	1317.96	1244.5	110.34	-12	351
PT9728	1317.96	1244.5	228.6	-31	351
PT9731	1334.11	1456.94	19.2	0	20
PT9732	1594.99	1102.89	229.51	-22	26.5
PT9733	1594.65	1102.28	272.49	-52	23.5
PT9734	1595.23	1101.76	155.14	-32	50
PT9735	1594.96	1102.83	198.73	-10	26
PT9736	1594.87	1102.95	237.13	-29	24
PT9738	1595.35	1097.49	14.33	-35	124
PT9739	1594.96	1097.43	204.52	-29	136
PT9740	1594.53	1097.25	180.75	-34	146

HOLE-ID	Easting (m)	Northing (m)	Length (m)	Dip	Azimuth
PT9741	1594.93	1097.74	155.14	-42	130
PT9742	1593.46	1096.88	247.65	-37	188
PT9743	1592.92	1096.85	246.58	-36	210
PT9744	1594.71	1097.4	290.17	-38	134
PT9744A	1594.71	1097.4	136.55	-35.5	141.5
PT9745	1593.1	1096.85	219.46	-26	193
PY06B	1769.36	1406.96	295.05	-60	273
PY07A	1772.11	1406.96	387.1	-67	275

Table 10-2 Assay Composites C Vein System

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
04-1707E1	201.93	203.03	1.10	10.20
04-1707E1	209.46	213.42	3.96	5.10
04-1707E2	222.08	224.73	2.65	10.10
04-1707E2	246.64	248.72	2.07	11.80
04-1737E1	184.25	190.80	7.77	17.13
04-1737E1	200.25	207.57	7.32	4.00
04-1737E1	231.19	235.37	4.18	13.60
04-1737E2	233.66	233.96	0.30	30.90
04-1737E2	256.64	259.08	2.44	9.20
04-300SW1	180.75	181.72	0.98	11.20
04-300SW2	181.20	181.66	0.46	5.50
04-300SW2	206.53	208.94	2.41	14.80
04-300SW3	196.38	203.61	7.22	7.20
04-300SW3	223.88	230.73	6.86	14.50
04-330SW1	167.18	168.80	1.62	4.00
04-330SW1	172.36	174.49	2.13	35.30
04-330SW2	168.86	171.60	2.74	5.40
04-330SW2	194.22	202.27	8.05	31.90
04-360SW1	162.76	163.67	0.91	5.10
04-360SW1	179.10	193.43	14.33	11.60
04-360SW2	191.20	195.38	4.18	25.70
04-360SW2	198.24	201.87	3.63	5.20
05-1676E1	185.62	191.20	5.58	12.10
05-1676E2	210.01	213.27	3.26	8.90
05-300SW4	200.13	200.86	0.73	13.40
05-300SW4	261.34	269.05	7.71	17.40
05-300SW5	215.80	218.30	2.50	14.10
05-300SW5	241.10	246.74	5.64	21.60
05-300SW6	233.54	240.79	7.25	16.70
05-300SW6	260.91	266.09	5.18	18.20

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
05-330SW3	213.94	223.11	9.17	9.90
05-330SW4	221.89	233.00	11.89	19.88
05-330SW5	230.89	246.28	15.39	8.57
06-1676E-6	284.8	292.2	7.4	8.0
06-1707E-3	232.7	235.8	3.1	17.4
06-1707E-3	247.0	250.0	3.0	5.9
06-1707E-3	253.0	254.9	1.9	10.5
06-1707E-6A	312.0	313.4	1.4	18.8
06-1737E-3	242.3	244.3	2.0	24.6
06-1737E-3	252.7	254.7	2.0	10.6
06-240SW-8	263.5	270.0	6.5	7.7
06-240SW-8	297.7	299.2	1.5	8.8
06-240SW-8	316.1	317.55	1.45	8.3
06-270SW-4	263.1	268.85	5.75	12.3
06-270SW-4	310.8	313.7	2.9	4.9
06-1615E-8	346.4	352.6	6.2	44.7
06-1768E-2	259.2	262.7	3.5	17.9
06-1768E-3	329.6	332.1	2.5	9.1
06-270SW-2	201.6	207.5	5.9	23.0
06-270SW-3	239.5	252.15	12.65	7.4
06-270SW-3	262.0	264.0	2.0	17.5
06-330SW-11	361.0	363.5	2.5	6.6
06-1768E-1A	225.8	242.3	16.5	23.1
06-1646E-6	258.3	265.5	7.2	15.5
06-1676E-5B	257.2	271.2	14.0	10.9
06-300SW-8	303.3	338.3	35.0	8.9
06-330SW-9	326.4	333.0	6.6	8.3
06-330SW-10	330.5	333.25	2.75	12.9
06-330SW-10	373.85	376.45	2.6	17.7
06-1585E-8	391.2	392.05	0.85	7.7
06-1615E-2	142.1	147.3	5.2	25.0
06-1615E-7.5	351.15	354.2	3.05	22.5
06-1615E-9	439.5	451.9	12.4	16.1
06-1813E-1	300.1	303.1	3.0	16.7
06-1813E-2	313.3	314.9	1.6	12.7
06-1813E-2	331.5	333.0	1.5	6.4
06-1813E-2	336.3	343.5	7.2	18.2
06-1813E-2	338.2	339.1	0.9	38.4
06-1813E-3	332.6	333.6	1.0	6.3
06-1813E-3	392.5	394.35	1.85	6.0
06-1813E-3	405.7	407.25	1.55	7.3
06-1707DE-1	425.6	430.1	4.5	11.8
06-1768DE-1	473.3	474.0	0.7	12.6

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
06-1768DE-1	487.4	491.9	4.5	9.5
06-1768DE-2	538.4	548.6	10.2	7.1
06-1859E-2	297.5	299.1	1.6	14.9
06-1859E-2	336.2	338.3	2.1	15.6
06-1859E-2	417.4	420.8	3.4	11.0
P8918A	194.43	199.49	5.06	37.70
P91C01	251.83	258.17	6.34	7.80
P91C01	274.93	281.48	6.55	20.20
P91C02	263.35	268.10	4.75	25.70
P91C03	226.47	229.64	4.20	11.46
P91C04	282.55	288.74	6.19	4.80
P91C05	156.73	169.10	12.37	10.30
P91C06	228.66	231.31	2.65	22.40
P91C07	292.06	294.35	2.29	18.80
P92C08	124.05	127.35	3.29	8.90
P92C09	161.85	163.98	2.13	4.20
P92C12	216.26	218.08	1.83	4.20
P92C12	258.47	265.27	6.80	15.20
P92C13	190.56	192.39	1.83	15.00
P92C14	246.10	248.69	2.59	6.90
P92C15	231.77	233.17	1.40	4.80
P92C15	236.89	238.66	1.77	18.70
P92C16	229.36	231.65	2.29	4.20
P92C17	209.70	213.48	3.78	21.80
P92C18	281.18	283.98	2.80	9.20
P92C19	282.64	285.51	2.87	23.80
P92C20	243.35	245.97	2.62	8.90
P92C20A	233.08	235.82	2.74	12.50
P92C21	172.15	178.06	5.91	18.70
P92C24	136.52	139.39	2.87	23.70
P93C26	108.02	110.95	2.93	26.60
P93C27	148.89	151.49	2.59	12.50
P93C28	140.27	142.43	2.16	10.50
P93C31	184.80	186.99	2.19	8.00
P94C33	266.43	269.53	3.11	6.90
P94C34	176.72	180.14	3.41	7.20
P94C36	261.21	262.28	1.07	6.40
P94C37	255.24	261.27	6.04	22.20
P94C38	255.54	257.13	1.58	20.10
P94C38	274.53	279.23	4.69	22.90
P94C39	262.01	266.58	4.57	28.70
P95C40	493.01	498.93	5.92	8.47
P95C40	727.77	735.24	7.47	11.31

Hole-ID	From	To	Length (m)	Au_Comp (g/t)
P95C42	456.47	457.54	1.10	13.13
P95C43	482.04	484.51	2.47	15.89
P95C44	586.50	589.57	3.07	18.76
P95C44	640.08	641.70	1.62	12.41
P95C44	692.38	694.15	1.77	28.06
PC25A	137.31	141.00	3.69	5.60
PC44A	586.40	591.22	4.78	16.00
PC44A	726.40	730.60	4.18	6.00
PC9001	206.35	213.79	7.44	11.80
PC9002	277.31	279.96	2.65	29.30
PC9003	215.59	222.50	6.92	14.80
PC9003	227.17	232.87	5.70	31.20
PC9004	221.25	225.25	3.99	16.10
PC9005	189.43	190.96	1.52	18.90
PC9007	160.90	162.31	1.40	15.20

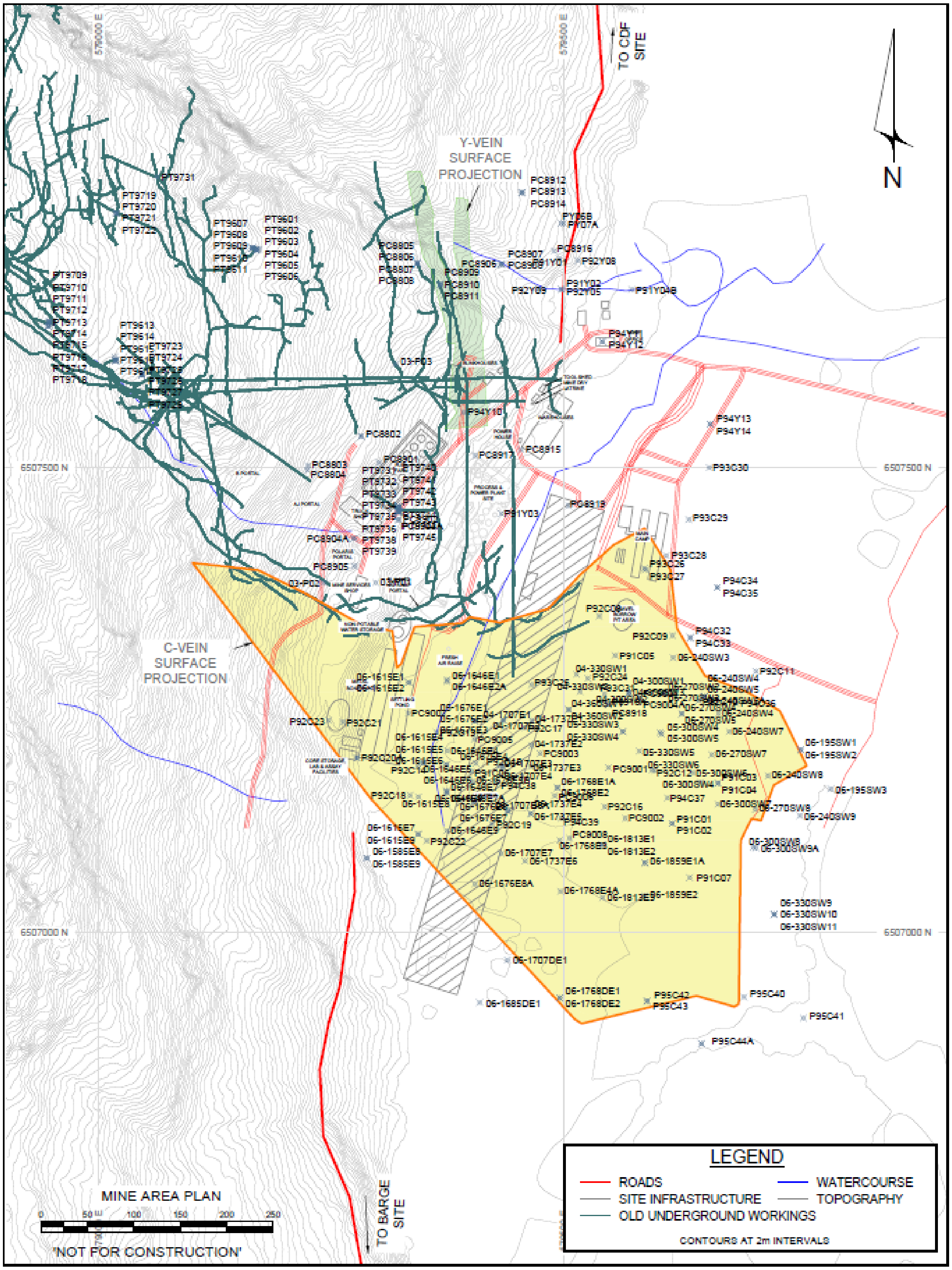

Figure 10-1 Drillhole Locations, Plan Map

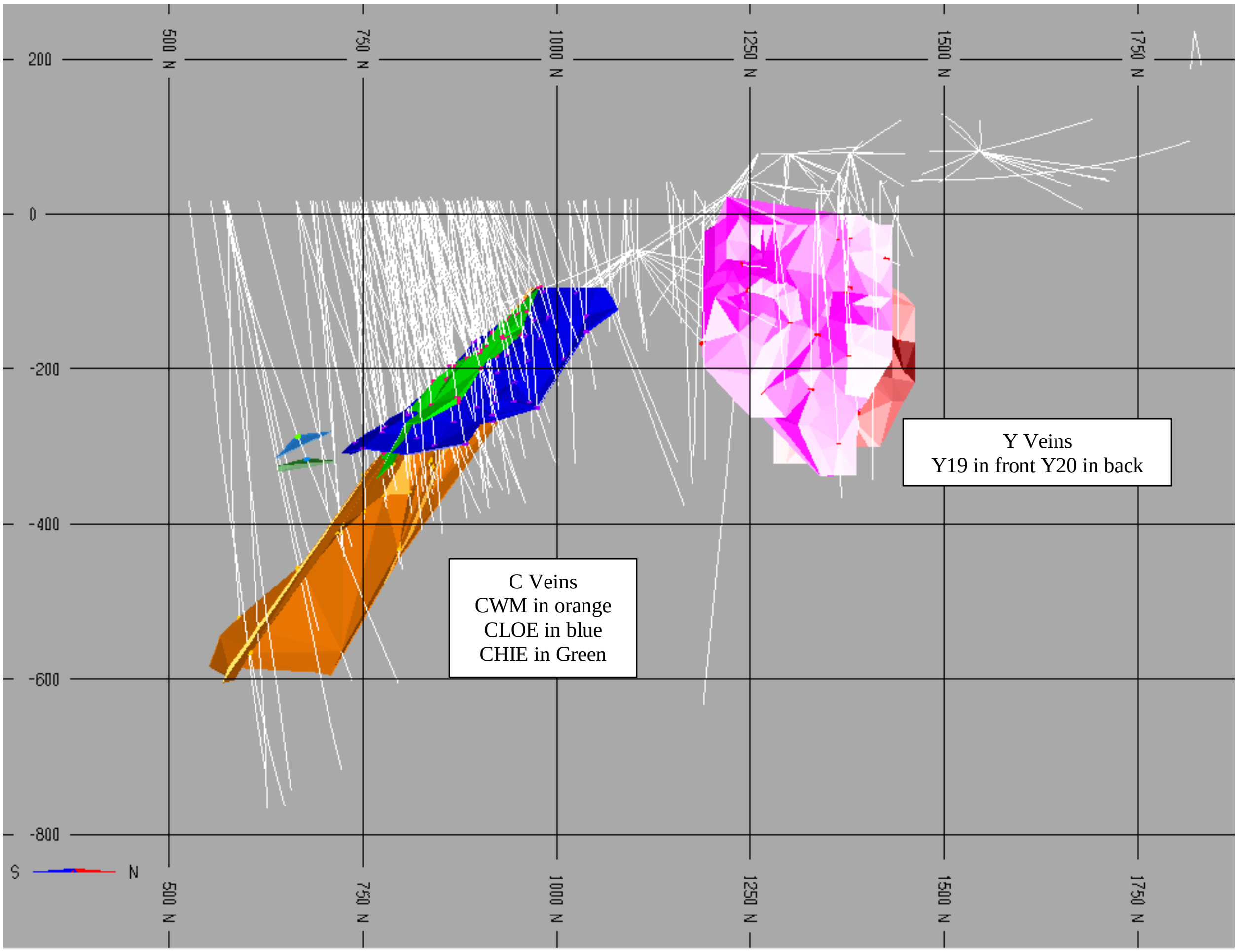

Figure 10-2 Drillhole Locations and Modelled 3D solids; Sectional View in mine specific space, looking east

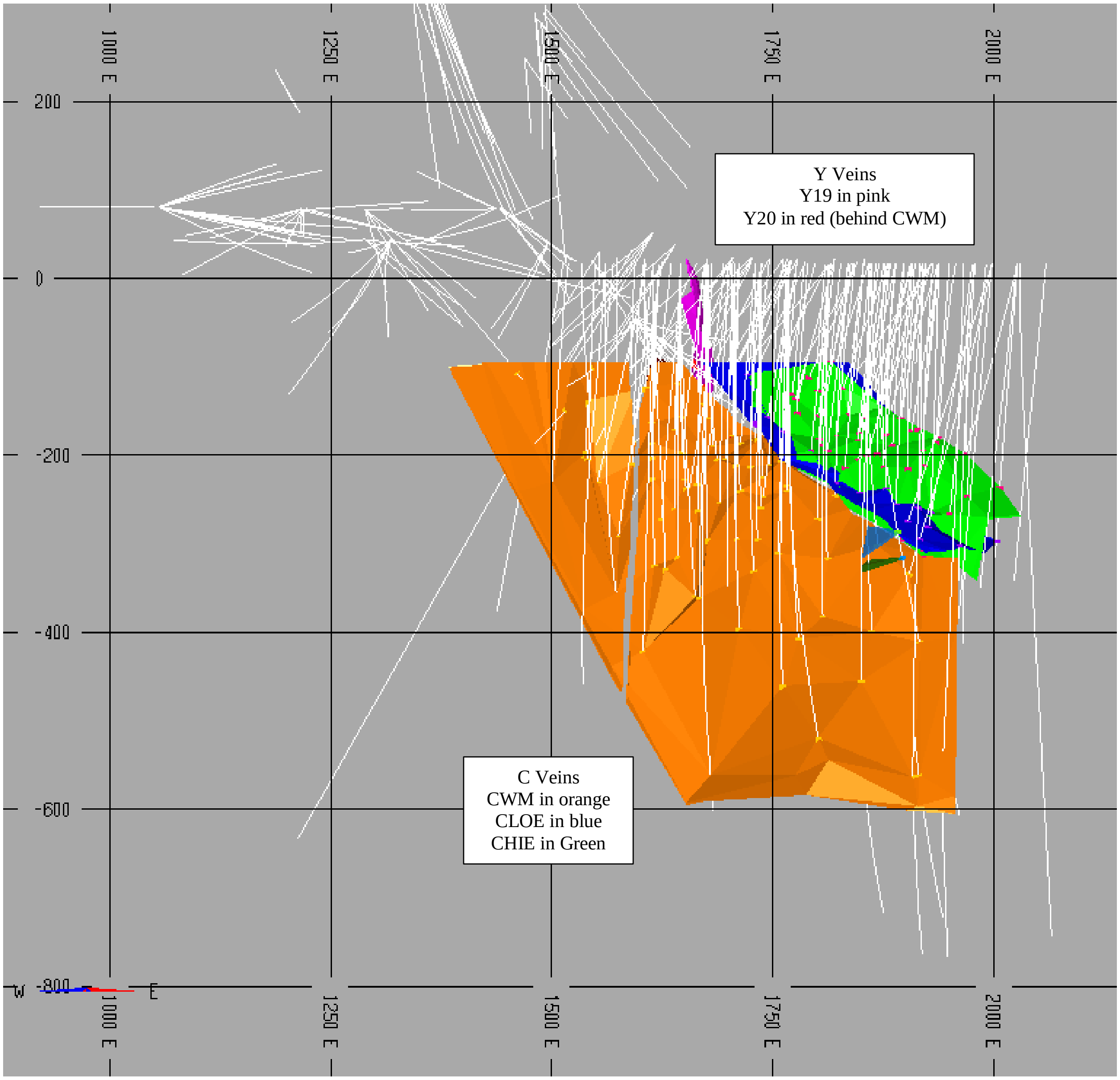

Figure 10-3 Drillhole Locations and Modelled 3D solids; Sectional View in mine specific space, looking north

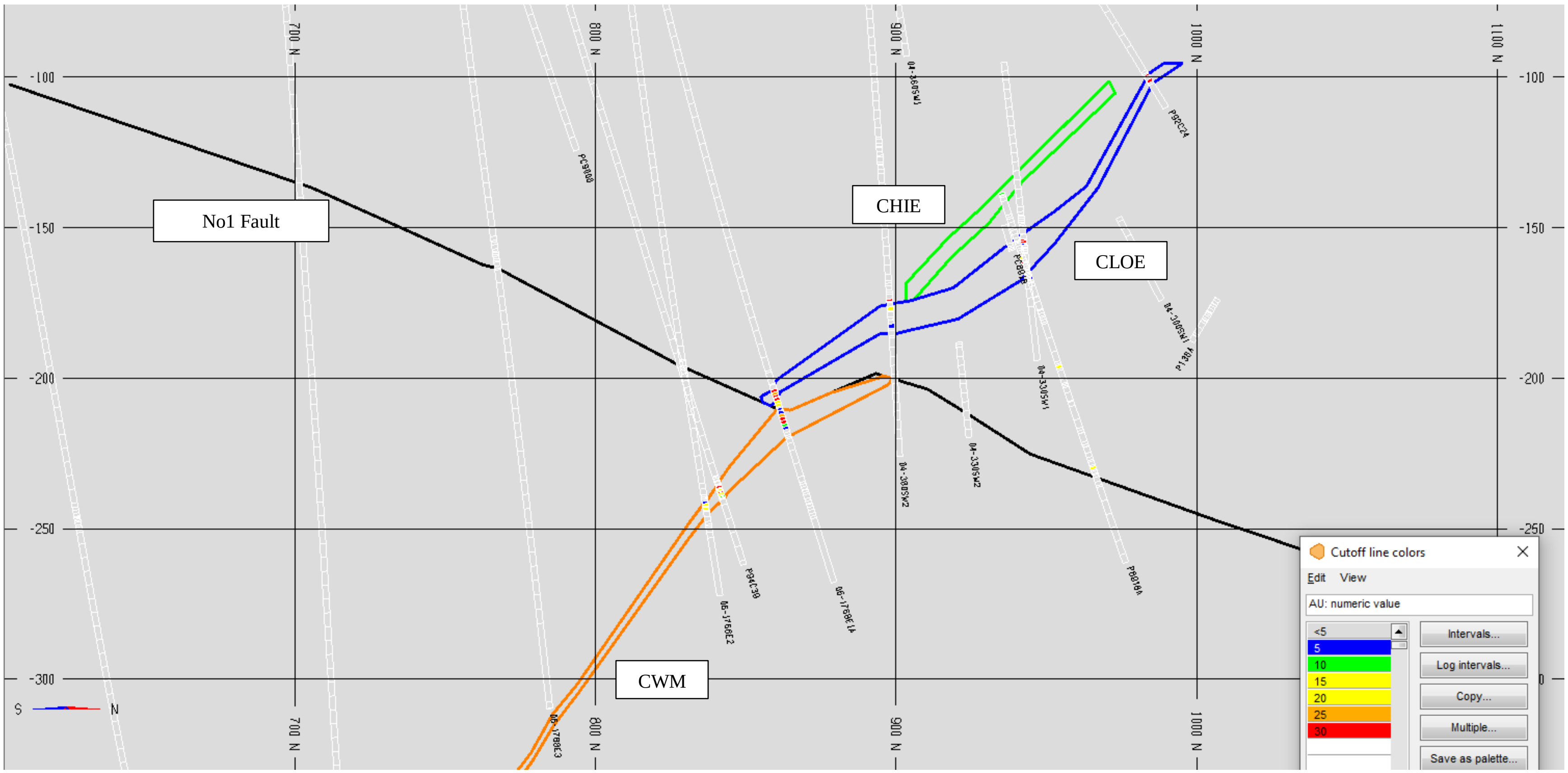

Figure 10-4 Drillhole Locations and Modelled 3D solids, North South Cross-section, East 1766 in mine specific space, looking west

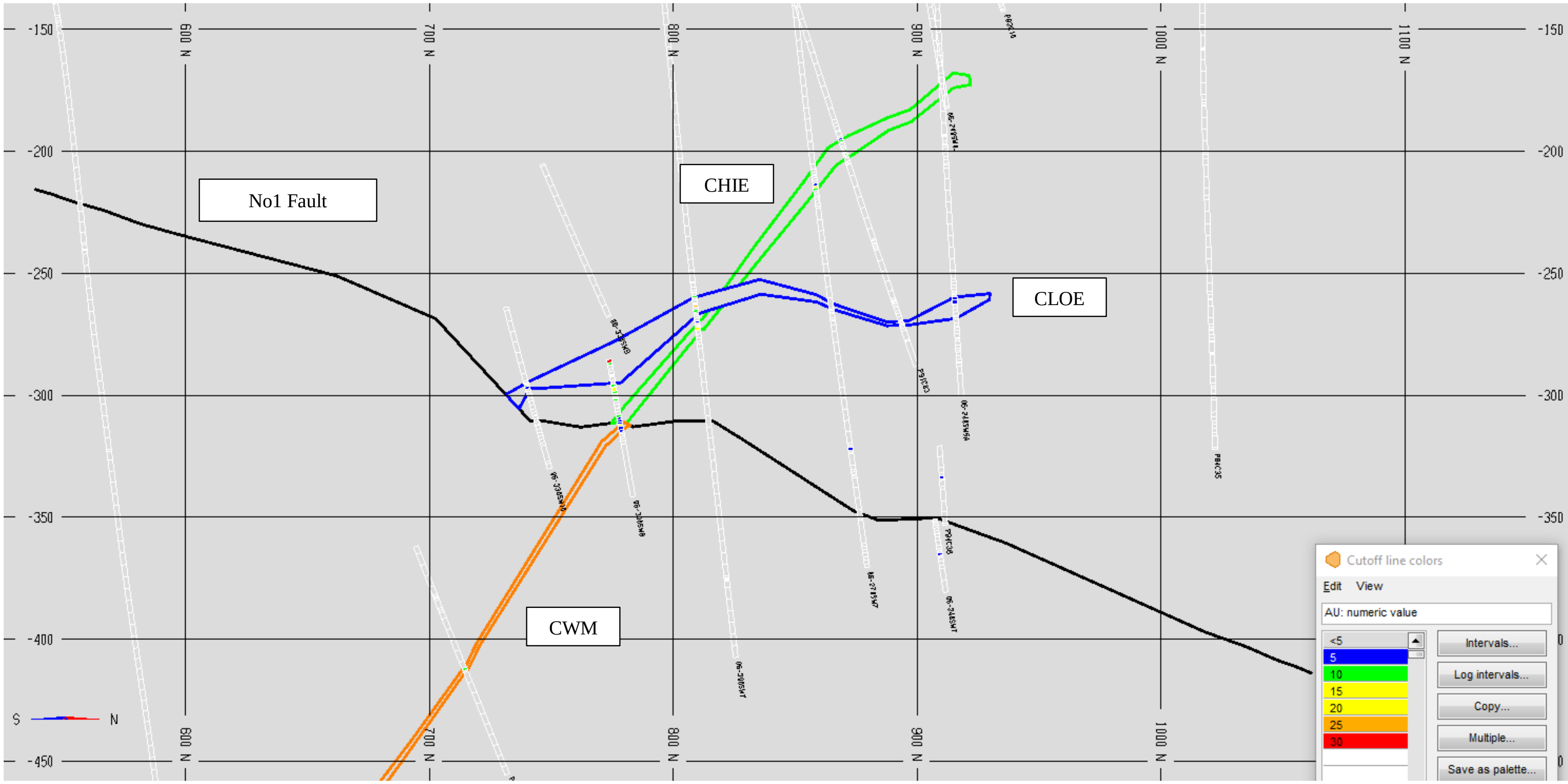

Figure 10-5 **Drillhole Locations and Modelled 3D solids, North South Cross-section, East 1916 in mine specific space, looking west**

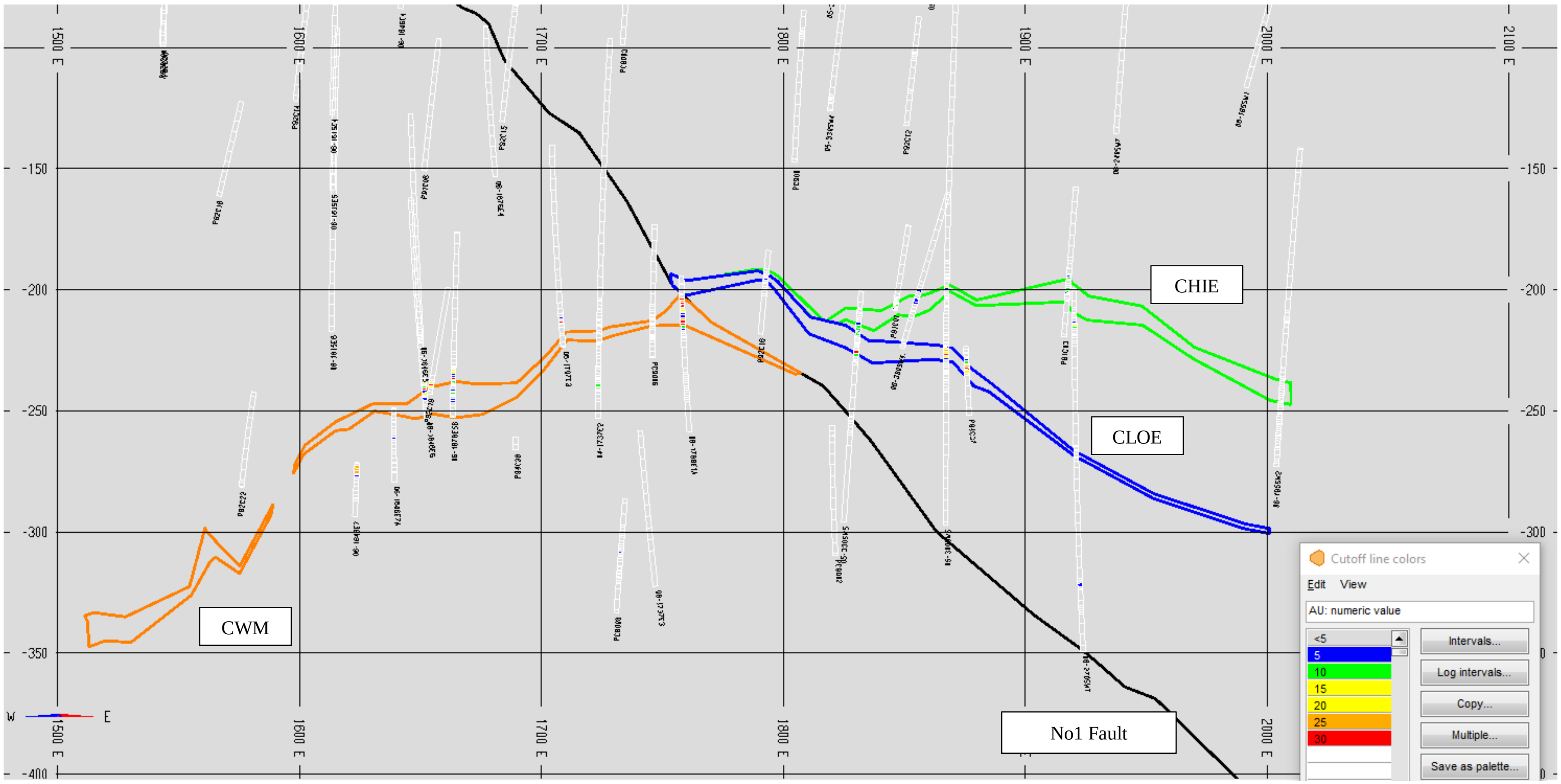

Figure 10-6 **Drillhole Locations and Modelled 3D solids , West East Cross-section, North 867.5 in mine specific space, looking north**

11 Sample Preparation, Analyses and Security

11.1 Sample Method and Approach

Drilling of the vein was done by wire line diamond drills using NQ-size rods. Drill collar locations were surveyed in by total station surveying method. Drilling azimuth and dip were set using a Brunton compass and inclinometer. Routine downhole measurements of azimuth and dip were not done on the three holes drilled in 2003 and prior. In 2004, three different downhole survey systems were tried before settling on a Reflex system. The Reflex system was also used in 2005. The downhole surveying was operated by the Hytech drill crew. This information was input to a GEMCOM program to plot the location of the collar and the pierce point of the veins.

Core recovery was very good and ranged from the low 90% to nearly 100% and is a fair sampling of the mineralization at the point where the drillhole pierced the vein.

Mineralization described in Section 7.3 suggests there is little nugget effect and gold values even over short intervals rarely exceed 1 oz/ton. Out of 4,700 samples with greater than 0.03 oz/ton gold collected from core and the underground workings, only 185 samples had a value greater than 1 oz/ton, the highest being 3.69 oz/ton. For this reason, no cutting of assays has been done in calculating composites nor are there many cases where a composite sample is carried by a single assay.

Determining intervals of core for sampling was done by the geologist during logging of the core. The mineralized vein structures were marked out. Selections of core intervals for sampling were based in the presence of veining and sulphide mineralization, particularly arsenopyrite. Within the defined vein structure sample intervals ranged from 0.3 m to 1.5 m. Divisions were based on intensity of mineralization and veining. Sampling of the core over several metres on either side of the mineralized vein structures was also done to the point where hydrothermal alteration disappeared. No sampling of core from the unaltered rock was done.

The core was logged and stored in the camp. Access to the core was only available to the geologists and the core sampler. The core was brought from the drill to the logging facility by the geologist at the end of each shift. The core was geologically logged, recoveries calculated, and samples marked out in intervals of 0.5 to 1.0 m. The core was handed to the sample cutter who cut it with a diamond saw. Each sample was individually wrapped in plastic bags for shipment. The sample intervals were easily identified and correlate well with the drill logs.

Core logging, sample layout, cutting and bagging procedures were observed by Morris during the site visit. The procedures for sample preparation, analysis and security procedures follow accepted engineering standards and the quality of gold analytical data collected by Canarc is sufficiently reliable to support Mineral Resource estimation.

Table 10-2 in the previous section, lists the relevant composite samples with gold values and sample length. True widths of the mineralized zone vary from 70% to 100%.

11.2 2006 Program Sampling

The 2006 QA/QC program was similar to the previous programs in that samples were collected by employees of Canarc on site and shipped to ALS Chemex laboratoty in Vancouver. For quality control and quality assurance, core samples were regularly mixed with blanks, duplicates, and standards. The program in the field was run in an efficient and proper manner following accepted engineering standards.

Blank samples represent material from the old mine, which is known to have a very low gold value. In total 56 blank samples were assayed. The sample statistics are shown in Table 11-1.

Table 11-1 Univariate Statistics, Blank Samples

Parameter	Result
Population	56
Minimum value	0.05
Maximum value	0.57
Mean value	0.07
Standard Deviation	0.08
CV	1.12

Three samples had gold values greater than three times the detection limit for gold (Sample C090930 with 0.57 g/t, sample C 090800 with 0.26 g/t, and sample C090770 with 0.17 g/t). Figure 11-1 shows the test results of the blank samples (excluding the highest value sample). The three samples, and others from the same batch, should be re-tested by the laboratory.

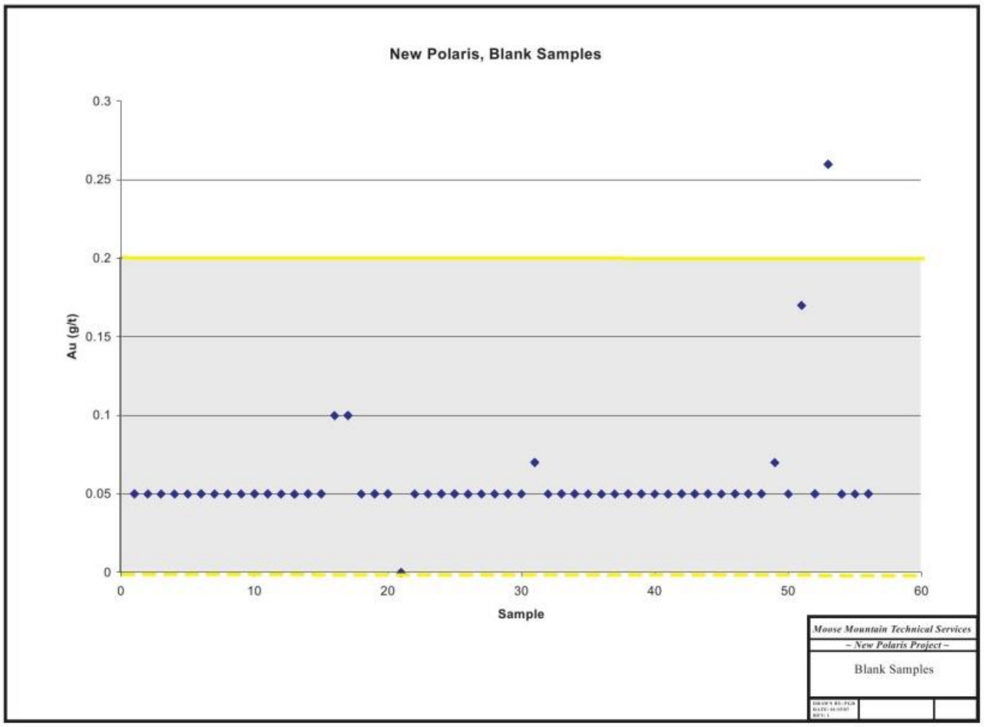

Figure 11-1 Blank Samples, 2006

Duplicate samples were made by cutting ½ of the drill core into ¼ core and submitting the quarters as two different samples. In total 45 duplicate samples were assayed. The sample statistics are shown in Table 11-2.

Table 11-2 Univariate Statistics, Duplicate Samples

Parameter	Result, First Sample	Result, Duplicate Sample	Result, Sample Difference
Population	45	45	45
Minimum value	0.025	0.025	-1.1
Maximum value	19.85	27.1	7.25
Mean value	1.52	1.81	0.30
Standard Deviation	3.72	4.76	0.18
CV	2.45	2.63	4.09

Three tests were completed to compare the duplicate sample results: a Student's t-test, which is a comparison of the mean values; an F-test, which is a comparison of the variance; and a Paired t-test, which is a test for bias. The Student's t-test shows that the means of the duplicate samples are likely to be the same (the calculated t-value is -0.32, well below the critical value of 1.96). The F-test shows that the variances of the two sample sets are likely to represent the same population (the calculated F-value is 0.61, within the range 0.065 and 1.68 of F-values characteristic of a normal distribution). The Paired t-test shows that the difference between the two data sets is minimal, and they are distributed around the value zero (the standard error is calculated to be 0.18, while the mean, 0.30, is between –0.065 and 0.657).

The three statistical tests indicate that the results of the duplicate sampling are acceptable.

Figure 11-2 shows the duplicate sample results. As shown, there is a strong correlation between the two sample sets, with a coefficient of correlation of 0.98.

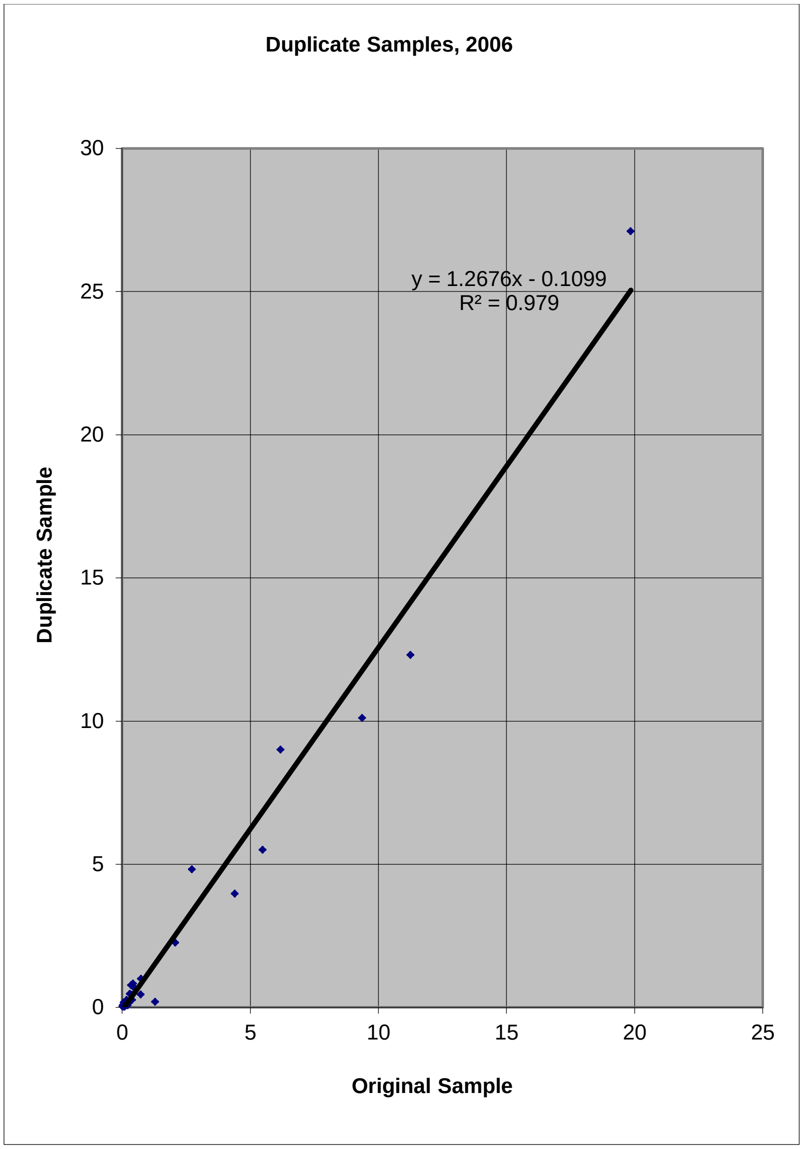

Figure 11-2 Duplicate Samples, 2006
(Values are Au g/t along both axes)

Three standard samples were submitted randomly for assay throughout the program to test the accuracy of the laboratory.

Table 11-3 Standard Samples

Standard	Mean (Au g/t)	Standard Deviation	Upper Range	Lower Range
PM 165	6.51	0.10	6.71	6.31
PM 415	2.37	0.12	2.49	2.13
PM 916	12.7	0.09	12.9	12.5

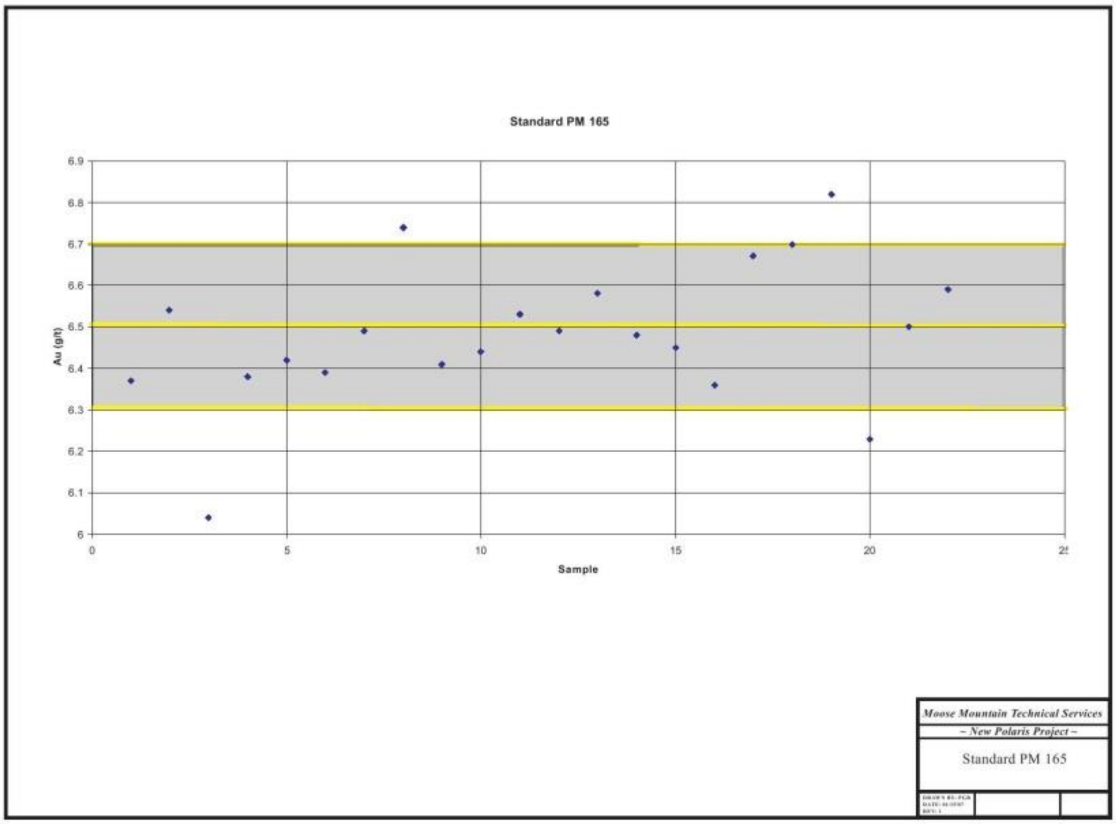

Figure 11-3 Standard PM 165, 2006

Figure 11-3 shows the results for standard PM 165. As shown, there are two samples with lower than acceptable values and two with higher than acceptable values. These samples, and others in the same batch, should be re-assayed.

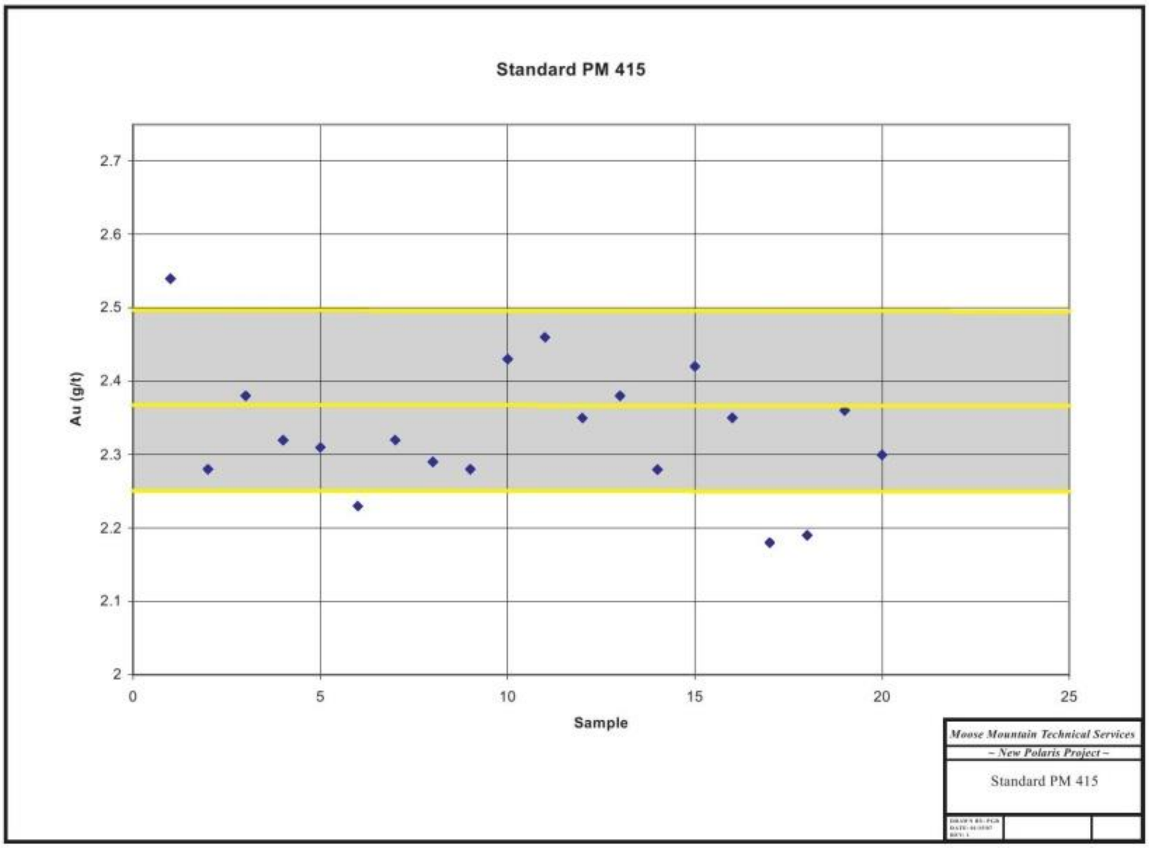

Figure 11-4 Standard PM 415, 2006

Figure 11-4 shows the results for standard PM 415. As shown, there is one sample with higher than acceptable values. This sample, and others in the same batch, should be re-assayed.

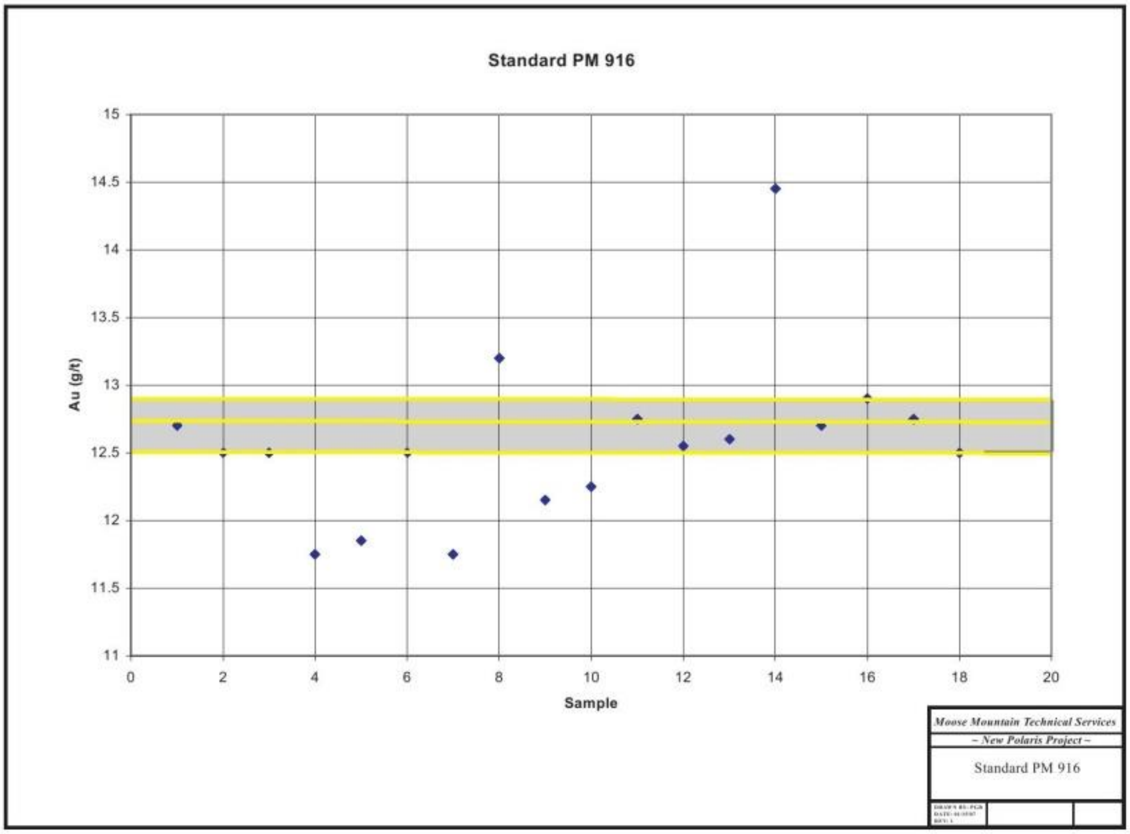

Figure 11-5 Standard PM 916, 2006

Figure 11-5 shows the results for standard PM 916. As shown, there are five samples with lower than acceptable values and two with higher than acceptable values. These samples, and others in the same batch, should be re-assayed.

A preliminary set of sample pulps was selected for re-assay by another laboratory. Acme Analytical Laboratory Ltd. (a highly accredited lab in Vancouver) was chosen as the second lab. The results are generally very consistent, except for one sample in the Chemex Sample 1 set, which assayed 22 g/t compared to 31.4 g/t by Acme and 31.4 g/t by Chemex the second time.

Table 11-4 Univariate Statistics, Round Robin Samples

Parameter	Chemex Sample 1	Difference Chemex 1 vs. Acme	Acme	Difference Acme vs. Chemex 2	Chemex Sample 2	Difference Chemex 1 vs. Chemex 2
Population	30	30	30	30	30	30
Minimum value	0.0	-9.43	0.01	-0.31	0.0	-9.4
Maximum value	40.7	0.36	42.11	2.04	41.4	0.8
Mean value	7.49	-0.50	7.99	0.19	7.80	-0.31
Standard Deviation	10.47	1.82	11.31	0.43	11.08	1.76
CV	1.40	3.65	1.42	2.25	1.42	5.70

Three tests were completed to compare the round robin sample results: a Student's t-test, which is a comparison of the mean values; an F-test, which is a comparison of the variance; and a Paired t-test, which is a test for bias. The Student's t-test shows that the means of the round robin samples are likely to be the same (the calculated t-value is -0.18 for the Chemex 1 vs. Acme samples; 0.11 for the Chemex 1 vs. Chemex 2 samples; and 0.07 for the Acme vs. Chemex 2 samples, well below the critical value of 1.96). The F-test shows that the variances of the three sample sets are likely to represent the same population (the calculated F-value is 1.04, for the Chemex 1 vs. Acme samples; 1.12 for the Chemex 1 vs. Chemex 2 samples; and 0.96 for the Acme vs. Chemex 2 samples, within the range 0.065 and 1.68 of F-values characteristic of a normal distribution). The Paired t-test shows that the difference between the Acme and Chemex 2 data set is minimal, and they are distributed around the value zero (the standard error is calculated to be 0.08, while the mean, 0.19, is between –0.065 and 0.657). In both cases, the Chemex 1 vs. Acme and Chemex 1 vs. Chemex 2 data sets, show a slight bias with the Chemex 1 samples being 0.32 to 0.33 g/t lower than the Chemex 2 and Acme respectively.

The three statistical tests indicate that the results of the round robin sampling are acceptable as per the Figures below.

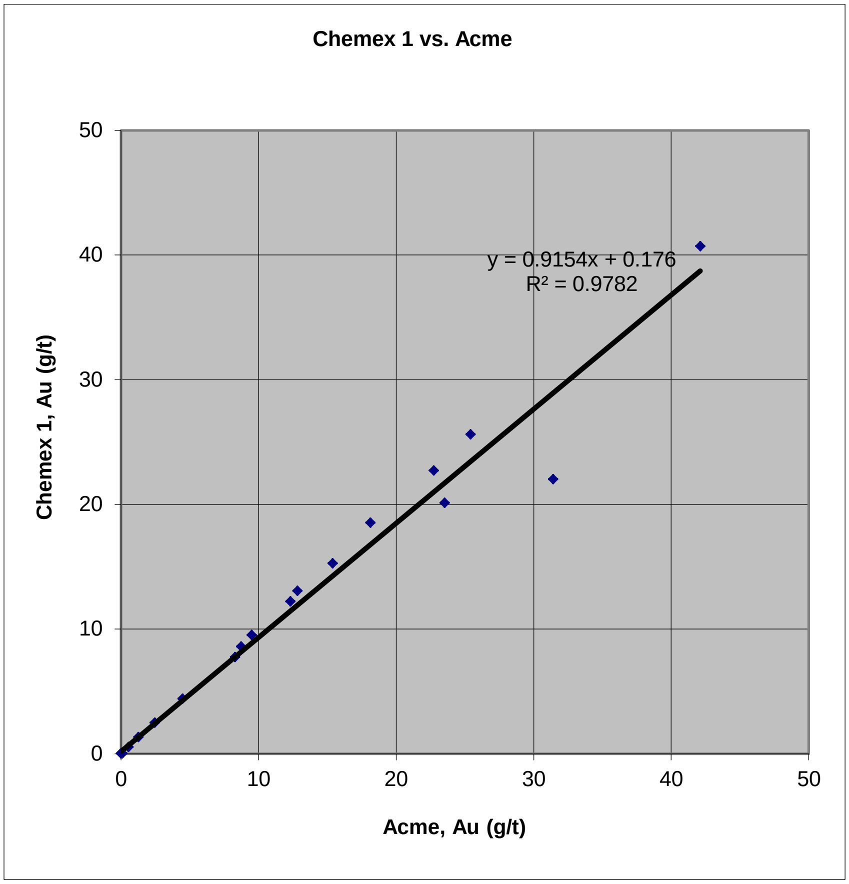

Figure 11-6 Round Robin, Chemex 1 vs. Acme, 2006

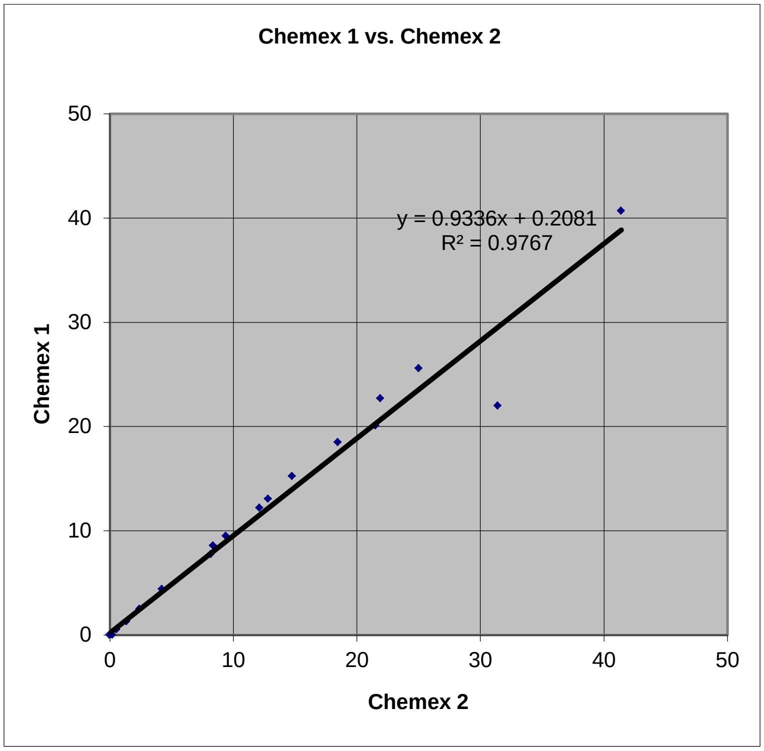

Figure 11-7 Round Robin, Chemex 1 vs. Chemex 2, 2006

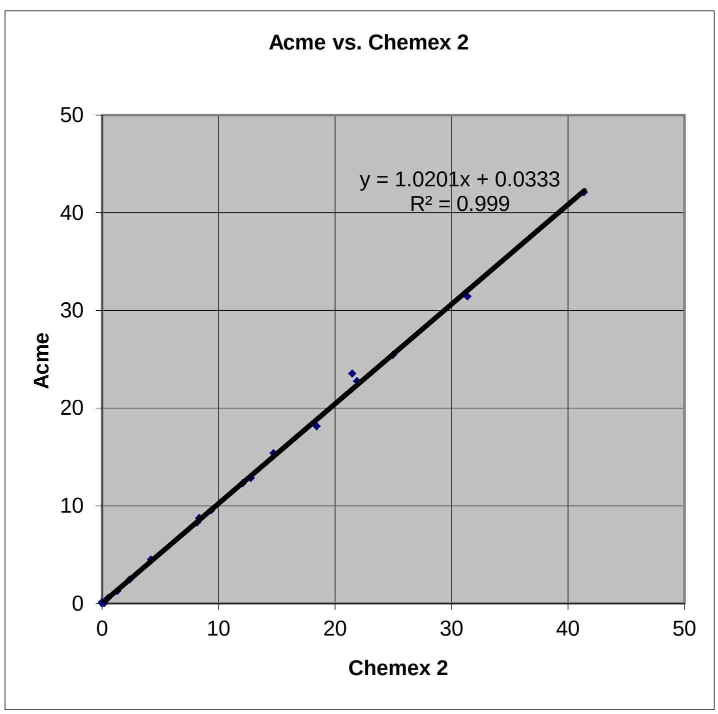

Figure 11-8 Round Robin, Acme vs. Chemex 2, 2006

12 Data Verification

R.J. Morris visited the property during 22-23 August 2006. The final drilling program on the property was more than 70% complete, with approximately 17,700 m of the 24,500 m of core drilled. At the time of the site visit core logging procedures from the 2006 program were examined, a general overview of the property was completed, including selected drill sites, historic core, an underground tour, and the condition of existing project infrastructure.

While on the property, the author examined underground workings to confirm the nature of mineralization, dimensions and extent of the vein system. The author also viewed a selection of core from key holes drilled from the early 1990's to the end of 2006 and compared his observations with those documented in the drill logs. In both the case of the underground workings and the core, the author found that his observations confirmed what was recorded in logs and sections. The author also confirmed that core had been properly cut and stored.

The core logging facility was clean and orderly. The system of check assaying is adequate. The only issue that the author has with the system is the use of quartered core for the duplicate samples. The sample size difference between the quartered and half core may account in part for the high relative difference between the original sample and the duplicate. In future, resubmission of pulps on a blind basis should be carried to help separate variance cause by analysis from that due to sample size or bias cutting of the core.

In addition to the site visit, a detailed review of the database was completed. Forty-one drillholes were selected from the C vein area, and the drill logs and assay sheets were compared with the database. Only minor differences were observed between the hard copy material and the database. As well, the input of the database into the geology and resource modeling software was also checked.

The quality control systems in place prior to the 2003 program are poorly documented and seem to follow the norms of that time period. Of concern is the manner in which the collar locations of drillholes were determined. Most of the holes were located using Brunton compass and chaining. Also, the down hole surveying was not consistently done. As a result, the exact location of the vein intersections is not as certain as those from the 2003 to 2006 program. Some re-drilling of older holes is recommended, especially where there are discrepancies with respect to the vein location between the recently drilled holes and those drilled in the 1990's.

The procedures used in the development of the database follow accepted engineering standards. There is confidence that selected drillholes for the Mineral Resource Estimate have error-free data and may be used to support Mineral Resource estimation. The data from those drillholes that do not have this confidence have not been used.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Gold in the New Polaris deposit is refractory and occurs dominantly in finely disseminated arsenopyrite grains. A 150-ton per day flotation mill was operated from 1937 to 1942 and again from 1946 to 1951 producing 231,604 oz of gold from a head grade of approximately 10 g/t.

Recent metallurgical test work has yielded positive results with a process flowsheet using flotation, bio-oxidation and CIL leaching.

13.2 Metallurgical Test Work History

Metallurgical test work performed on New Polaris material after 2003 is summarized in Table 13-1.

Table 13-1 New Polaris Metallurgical Test Work History

Laboratory	Sample Type	Tests	Comments
Resource Development Inc. (RDI) (2003)	Bulk sample	Grinding	Grind tests indicated a P_{80} of 75 µm.
		Diagnostic Leach	Indicated the following gold distribution: 66.4% Arsenopyrite/Pyrite, 20.1% Quartz, 9% Free gold, 4.5% Stibnite gold.
		Rougher Flotation	Overall good results observed (83 – 94% recovery).
		Rougher Concentrate Production – Test 4	17.6% mass pull, 90.15% AU recovery, approximately 15% carbonates.
	Rougher Concentrate from Test 4	Cleaner Flotation	Indicated only 1 stage of cleaning required. Cleaner flotation not sensitive to residence time. Test 7: Rougher test to generate concentrate for Tests 8, 9. 18.9% mass pull with 96.1% Au recovery. Confirmed no regrind required prior to cleaning.
	Tails and Conc. From Test 4	Gravity Concentration	Indicated ore is generally not amenable to gravity concentration.
	Rougher and cleaner Tails	Cyanidation – Bottle Roll	Poor gold recovery, high NaCN consumption.
	Bulk Sample	Rougher Concentrate Production	19.5 initial wt., 96.6% Au recovery, 12 kg conc. Concentrate shipped to Mintek and Oxidor, tested by RDI.

Laboratory	Sample Type	Tests	Comments
Mintek (2003)	Concentrate from RDI Rougher Test (2003)	Cyanidation – CIL and Nitric Acid pre-treatment.	17.6% extraction vs. 93.7% after nitric acid treatment. Confirms sulphide oxidation required for gold recovery.
		Diagnostic Leach	Confirmed majority gold content is refractory in sulphides.
		Bio Leach	2 Tests: normal bio-leach, bio-leach + acetone and ferric pre-wash. Poor dissolution, indicated inhibitory substance (As^{+3} or As^{+5}).
RDI (2004)		Batch Acid Pressure Oxidation (POX)	100% Sulphur oxidation after 1.5, 1 hour, respectively.
		Cyanidation – Bottle Roll Tests	9.4% baseline Au recovery, > 98% Au recovery after POX (1, 1.5 hours).
Oxidor Laboratory (2004)		Bio-Oxidation (BIOX™) Amenability Tests	Bio-oxidation test using OXL-1014-R-13 culture; adaptation, inoculation, build-up. 98% sulphide oxidation, 90% Au extraction after 9 days BIOX™. Confirmed inhibitory substance (As^{+3} or As^{+5}).
		Cyanidation – CIL	8.2% Au recovery without BIOX™, compared to 90% Au extraction after BIOX™.
Process Research Associates Ltd. (PRA) (2007)	Bulk Sample	Cyanidation – Bottle Roll Tests	Low Au recovery.
		Gravity Separation	Confirmed not amenable to gravity.
		Rougher and Cleaner Flotation	Tested various flotation conditions for flotation optimization. Pyrite/Arsenopyrite separation unsuccessful. Best results achieved after 1 rougher, 1 scavenger, and 1 cleaner stage: 15.2% mass pull, 94.9% Au recovery.
		Locked Cycle Flotation	Multiple recycled streams used, 5 cycles with 3 stages of cleaning.
	Final bulk flotation tails	Slurry Settling Test	Two settling tests: with/without Percol 156 flocculant. 21% solids increased to 71% solids, 0.9 m/h settling rate increased to 2.5 m/h with flocculant.
	Flotation Concentrate	Cyanidation – High Intensity	Poor results: 10.6% Au recovery, 31 kg/t NaCN consumption.

Laboratory	Sample Type	Tests	Comments
		Leach	
Outotec RSA (PTY) Ltd. (2018)	Flotation Conc.	BIOX™ Batch Amenability Tests (BAT)	Inoculum Adaptation, 7 BATs (12 – 22-day oxidation times). Achieved 89.6 – 99.1% Sulphide Oxidation.
	Flotation Conc. and BIOX™ Residue	Cyanidation – Bottle Roll Tests	8.1% baseline Au recovery. Approximately 95.7% Au recovery after 22-day BIOX™.
	BIOX™ Liquor from BAT 1	BIOX™ Liquor Neutralization Test (2018)	3 Tests: lime/limestone, lime only, and lime/limestone + $Fe_2(SO4)_3$. Slow neutralization, [Fe], [As] indicates As can be reduced to below EPA limit of 0.4 mg/L. Long term As stability requires addition of $Fe_2(SO4)_3$.
	BIOX™ Slurry from BAT 3	BIOX™ Residue Static Settling Tests (2018)	8 flocculants tested for clarity of liquor, dosage tests once flocculant chosen. 150 – 250 g/t Magna 405 flocculant produces good settling rate, 300 g/t, may improve results slightly.

13.3 Samples

Material used for metallurgical test work have been collected from anticipated mining zones in the New Polaris deposit. Samples were representative of grade and type of ore expected to be processed. For work performed by RDI and PRA, the samples used were bulk ore samples and flotation concentrates collected from this material. The flotation concentrate used by SGS South Africa for BIOX testing was produced by Inspectorate Labs in Vancouver using composited drill core collected from throughout the deposit. Outotec supervised the 2018 BIOX™ testing program conducted by SGS South Africa.

13.3.1 RDI 2003 Bulk Sample

Bulk sample material was tested and used to produce the flotation concentrate tested in 2004 by RDI, Mintek, and Oxidor Laboratories. The RDI Bulk Sample head grade is shown in Table 13-2 with mineralogy summarized in Table 13-3.

Table 13-2 RDI Bulk Head Grade

Method	Au	Ag	As	Sb	SiO_2	Fe_2O_3	S	MgO	Al_2O_3	CaO	Cu	Zn
	g/mt	g/mt	ppm	ppm	%	%	%	%	%	%	ppm	ppm
FA/AAS	19.48	< 1.71										
ICP			28,350	127								
XRF			24,500	300	45.1	9.8	1.92	9.79	13.7	14.1	169	63

Table 13-3 RDI Bulk Sample Mineralogical Data – XRD Results

Mineral	Approximate Weight (%)
Dolomite	32
Mica/Illite	32
Quartz	23
Arsenopyrite	5
Pyrite	< 5
K-Feldspar	< 3
Unidentified	< 5

13.3.2 PRA 2007 Samples

PRA used individual drill core to create a composite sample which was used mainly for rougher and cleaner flotation tests. Metallurgical data provided by PRA is summarized in Table 13-4 and Table 13-5.

Table 13-4 PRA Sample Grades

Method	Au	Ag	As	Sb	S_T	S^{-2}	S_{SO4}	C_{ORG}	C_T
	g/mt	g/mt	%	%	%	%	%	%	%
FA/AAS	9.66	1.0							
ICP			0.45	0.78	3.18	3.18	< 0.01	0.33	3.7

Table 13-5 PRA Sample Mineralogical Data

Method	Al_2O_3	BaO	CaO	Fe_2O_3	K_2O	MgO	MnO	Na_2O	P_2O_5	SiO_2	TiO_2	LOI
	%	%	%	%	%	%	%	%	%	%	%	%
Whole Rock Analysis	10.70	0.02	7.80	8.10	2.88	7.11	0.12	0.32	0.13	46.20	0.56	14.64

13.3.3 Outotec 2018 Samples

The Outotec BIOX™ campaign used flotation concentrates produced by Inspectorate. Samples used to create the flotation concentrate were composited from drill hole samples. The Inspectorate composite head sample data is shown in Table 13-6.

Table 13-6 Inspectorate Sample (Canarc 2015)

Composite #	# of Samples	Total Weight (kg)	Average Au Grade (g/t)	% S Total	% As Total
1	116	304	9.75	2.42	1.62
2	84	150	20.44	2.26	1.74
3	82	97.5	13.92	2.53	1.74
Combined	282	551.4	13.4	2.40	1.67

13.4 Mineralogy

Gold mineralization is associated primarily with the sulphides arsenopyrite, pyrite, and stibnite. Gold is disseminated throughout these minerals along with gangue minerals including quartz and carbonate material. Approximately 10% of the ore material consist of sulphides of which less than 0.1% is Stibnite. Diagnostic leach results described in 13.5.2 shows deportment of gold in the tested bulk sample. XRD analysis of an ore sample shows the following composition:

- 32% Dolomite
- 32% Mica/Illite
- 23% Quartz
- 5% Arsenopyrite
- < 5% Pyrite
- < 3% K-Feldspar
- < 5% “Unidentified” (possibly amorphous)

13.5 RDI 2003

13.5.1 Grind Test

A grind test was performed on the bulk sample. A rod mill was used, and grinding was conducted at 50% solids for 20 to 60 minutes. The slurry was wet-screened with a 400-mesh screen, then dried and dry-screened. Results of these tests show that grinding will take approximately 37 minutes to obtain a P_{80} of 75µm. Results of these tests are given in Figure 13-1.

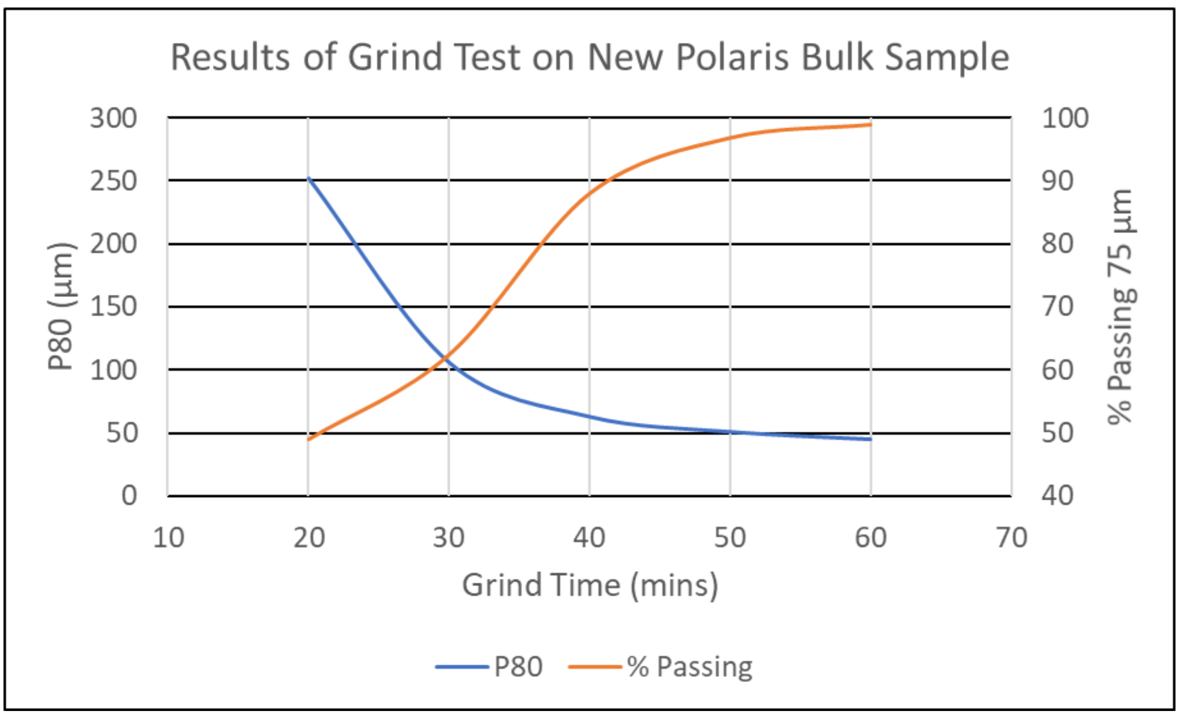

Figure 13-1 Results of Grind Tests on New Polaris Bulk Sample

The bond ball mill work index on the test composite was determined in a Bico Braun laboratory mill using the standard procedure of six cycles to stabilize the circulating loads. The work index was calculated based on a closing screen size of 105 microns.

The calculated work index was 19.6 kWh/tonne. The ore is therefore considered to be relatively hard.

13.5.2 Diagnostic Leach

For a diagnostic leach test, 1000 g of bulk sample were ground to a P_{80} of 75 µm and screened so that 100% of the material passed through 105µm. The following five sequential leach stages were carried out:

1. Cyanide leach (48 hours, 40% solids, approximately 2 g/L NaCN)
   - free-milling gold – **9% Au extraction**
2. 4-hour leach (pH 2, 20% solids) + 48-hour cyanide leach (1 g/L NaCN)
   - Stibnite associated gold – **4.5% Au extraction**
3. 4-hour Roast at 425°C + 4-hour cyanide leach (40% solids, 1 g/L NaCN)
   - Arsenopyrite and Pyrite associated gold – **27.2% of total extraction**
4. 4-hour Roast at 625°C + 48-hour cyanide leach (40% solids, 1 g/L NaCN)
   - Pyrite associated gold – **39.2% Au extraction**
5. Fire assay
   - Quartz associated gold – **20.1% Au extraction**

The high Au extractions after roasting confirms that a significant portion of gold is refractory in sulphides.

13.5.3 Rougher Flotation

A series of flotation tests were conducted to establish initial process conditions. All tests used three 5-minute floats. Individual conditions used for these tests are:

- Test 1: 200 g/t PAX, 50 g/t MIBC
- Test 2: 180 g/t PAX, 45 g/t MIBC, 500 g/t Na_2S in grind, 250 g/t $CuSO_4$ added after first two floats
- Test 3: 180 g/t PAX, 55 g/t MIBC, 2316 g/t H_2SO_4 (to obtain pH 5.8), 250 g/t $CuSO_4$ after first two floats
- Test 4: 180 g/t PAX, 45 g/t MIBC, 85 g/t Na_2S after first two floats

Good recoveries were achieved (83 - 94% Au recovery), with concentrates mass pulls of approximately 20% of the feed. Analysis of the concentrate produced by Test 4 shows that approximately 7.5% of the material releases CO_2, indicating that the concentrate has approximately 15% carbonate content. Results are given in Table 13-7.

Table 13-7 Summary of Rougher Flotation Tests – Adapted from RDI 2003

Test #	Comments	Recovery (% of initial feed)			Assay Value		
		Wt.	Au	As	Au (g/t)	As (%)	CO_2 (%)
1	PAX/MIBC	20.7	83.6	83.5	76.63	9.10	
2	Add Na_2S, $CuSO_4$	18.1	93.8	93.9	94.24	11.88	
3	Lower pH	21.0	87.0	86.1	78.08	10.05	
4	Rougher Conc. Gen.	21.0	90.1	89.1	81.96	10.85	7.47

13.5.4 Rougher Concentrate Production (Test 4)

The concentrate resulting from Test 4 was used for gravity Concentration, direct cyanidation, and cleaner flotation tests. The cleaning test work indicates that carbonate content can be reduced but is accompanied with significant gold losses.

13.5.5 Cleaner Flotation

A series of five tests were conducted to evaluate cleaner flotation conditions. Tests 5 and 6 used the rougher concentrate produced from Test 4 (described above), whereas Test 7 used a new bulk sample to produce concentrate that would be used for Tests 8 and 9. Conditions and results for each test is are shown below and in Table 13-8:

- Test 5: Two flotation stages (1 x 10 min. froth, 2 froths: 5 mins. and 4 mins.); 10 g/t PAX, 5 g/t MIBC
- Test 6: Timed flotation – 1-min. froth, 3-min. froth, 6-min. froth; 10 g/t PAX, 4 g/t MIBC
- Test 7: Conc. production – 3 floats: 8, 5, 8 mins.; 180 g/t PAX, 500 g/t Na_2S in grind, 250 g/t $CuSO_4$ after first two floats, 45 g/t MIBC
- Test 8: no regrind of Test 7 rougher concentrate; 10-min. float, 12 g/t PAX, 2 g/t MIBC
- Test 9: regrind of Test 7 rougher concentrate; 8-min. float, 24 g/t PAX, 4 g/t MIBC

Table 13-8 Summary of Cleaner Flotation Tests – Adapted from RDI 2003

Test #	Comments	Float time (min)	Recovery (% of feed)				Assay Value		
			Wt.	Au	As	CO_2	Au (g/t)	As (%)	CO_2 (%)
5	Cleaner 2 Con 1	5	43.4	63.2	62.1	3.1	118.00	15.00	0.54
	Cleaner 2 Con 2	4	11.8	22.7	23.5	7.3	155.14	20.80	
	Cleaner 2 Tail		5.9	7.1	7.4	5.2	98.68	13.10	6.58
	Calc. Cleaner 1 Conc.	10	61.1	93.0	93.0	15.6	123.3	15.94	
	Cleaner 1 Tail		38.9	7.0	7.0	84.4	14.54	1.89	16.20
6	Cleaner 1 Con 1	1	42.1	62.0	63.2	6.5	122.02	16.10	1.10
	Cleaner 1 Con 2+3	9	20.6	32.5	31.5	14.9	130.80	16.40	5.20
	Cleaner 1 Tail		37.3	5.5	5.3	78.6	12.07	1.52	15.10
7	Rougher Conc.	24	18.9	96.1	97.4		93.27	13.08	
	Rougher Tail		81.1	3.9	2.6		0.89	0.08	
8	Cleaner Conc.	10	80.8	97.2	97.9		106.78	15.7	
	Cleaner Tail		19.2	2.8	2.1		13.17	1.44	
9	Cleaner Conc.	8	46.7	56.5	55.2		116.26	15.60	
	Cleaner Tail		53.3	43.5	44.8		78.46	11.10	

- Test 5 shows that the 1st cleaner stage is effective in removing 84% of the carbonate but this is accompanied with a 7.1% gold loss.
- Results from Test 6 confirm that additional residence time and cleaning stages may improve grade but are associated with significant Au recovery losses.
- Test 7 conditions for rougher concentrate generation had the best rougher results with 96.1% gold recovery and Au 93 g/t concentrate grade with 18% mass pull. This indicates that cleaner flotation may not be required.
- Tests 8 and 9 show that regrind of rougher concentrate is not beneficial for upgrading concentrate, despite a slightly higher grade with regrind. While Test 9 produced an upgraded concentrate, a large portion of gold was rejected. Test 8 produced high grades and retained more than 97% of the gold.

13.5.6 Gravity Concentration

Gravity concentration tests used a Knelson concentrator with the objective of upgrading the material produced in rougher Flotation Test 4. Results are given in Table 13-9 and Table 13-10.

Table 13-9 Summary of Gravity Test (Rougher Tails) – Adapted from RDI 2003

Comments	Recovery (% of initial feed)			Assay Value	
	Wt.	Au	CO_2	Au (g/t)	CO_2 (%)
Knelson Concentrate	8.5	11.4	3.5	3.77	7.18
Knelson Tail	91.5	88.6	96.5	2.71	18.10
Calc. Rougher Tail	100	100	100	2.80	17.18

Table 13-10 Summary of Gravity Test (Rougher Conc.) – Adapted from RDI 2003

Comments	Recovery (% of initial feed)		Assay Value
	Wt.	Au	Au (g/t)
Knelson Concentrate	9.7	12.0	100.34
Knelson Tail	90.3	88.0	79.22
Calc. Rougher Conc.	100	100	81.88

Both tests show that the tails and concentrate from Test 4 could not be upgraded without significant losses of gold. The poor gravity recoveries indicate that the samples were not amenable to gravity separation.

13.5.7 Cyanidation – Bottle Roll Tests

Three tests were conducted to recover the gold from various materials collected from previous tests. All tests used 5 g/L NaCN at pH 11. Other conditions are as follows:

- Rougher Tails (from Test 4): 48-hour leach, 40% solids,
- Rougher Tails (from Test 7): 24-hour leach, 40% solids,
- Cleaner Tails (from Test 8): 24-hour leach, 25% solids.

Results of these tests are given in Table 13-11.

Table 13-11 Leach Test Results – Adapted from RDI 2003

Sample Leached	Calc. Head Grade (g/t)	Au Extraction (%)	NaCN Consumption (kg/t)
Test 4 Rougher Tails	2.13	23.3	3.88
Test 7 Rougher Tails	0.92	29.5	4.139
Test 8 Cleaner Tails	13.15	17.9	4.696

It is clear from these results that the flotation concentrate is refractory. It is not economical to leach the tails from either the rougher or cleaner stages without oxidation of the sulphides.

13.5.8 Rougher Concentrate Production

RDI also conducted a series of six rougher flotation tests to produce concentrate for future testing. Each test used 10 kg of material, and obtained three froths per test, using 8 minutes, 5 minutes, and 8 minutes for the float times. Each test used 180 g/t PAX, 500 g/t Na_2S during the grind, 250 g/t $CuSO_4$ after the first two floats, and 45 g/t MIBC. Overall, 12 kg of concentrate were produced. Assay results are given in
Table 13-12.

Table 13-12 Rougher Concentrate Production Analysis – Adapted from RDI 2003

Lab Assayed	Recovery (% of initial feed)					Assay Value			
	Wt.	Au	As	S	CO_2	Au (g/t)	As (%)	S (%)	CO_2 (%)
RDI	19.5	98.6	88.1	98.1		91.98	12.80	21.1	
Mintek						89.9	14.3	19.7*	
RDI						91.84	14.72	20.3	3.4
Oxidor						93.96			

** Sulphide value*

13.6 Mintek 2003

13.6.1 Cyanidation – CIL Tests

As part of the "diagnostic leach" tests, a direct cyanide leach with carbon was conducted to gain a baseline value for gold extraction from the rougher concentrate produced at RDI in 2003. The 24-hour leach used "excess reagents" including 5 kg/t NaCN, and 20 g/L activated carbon. Results are given in Table 13-13.

13.6.2 Diagnostic Leach

Three individual tests were performed on the rougher concentrate produced at RDI in 2003. The first test, a direct cyanide leach, has already been described above. The other two tests are described here:

- Dilute Nitric acid (10% w/w, 70°C, 4 hours) + CIL (20 g/L activated carbon, 5 kg/t NaCN, 24-hour leach),
- Concentrated Nitric acid (27.5% w/w, 70°C, 4 hours) + CIL (20 g/L activated carbon, 5 kg/t NaCN, 24-hours),
- A repeat test with excess cyanidation reagents: 20 g/L activated carbon, 50 kg/t NaCN, 24-hours.

Results of all three tests are given in Table 13-13.

Table 13-13 Results of Diagnostic Leach – Adapted from Mintek 2003

Test	Dissolution (%)		Au Extraction (%)	NaCN consumption (kg/t)
	Sulphide	Arsenic		
Direct CIL	0	0	17.6	2.7
Dilute HNO_3 + CIL	26	42.5	60.3	4.85
Conc. HNO_3 + CIL	96.3	97.6	71.9	4.9
			93.7	49.3

Baseline CIL resulted in only 17.6% gold recovery, whereas concentrated Nitric acid resulted in 71.9% recovery. With excess NaCN, the gold recovery value increases to 93.7%. The diagnostic leach confirmed the need for sulphide oxidation before leaching.

13.6.3 Bio-leach Tests

Three bio-leaching tests were undertaken by Mintek: one control test and 2 bio-leach tests. Each test used the following conditions: 40°C, 10% solids, pH 1.8 solution, air enriched with 0.3% CO_2, "0K" nutrient broth, and the two non-control tests used mesophilic bacteria taken from maintenance reactors. For each test, the levels of aqueous iron and arsenic were measured, along with pH, redox potential, and dissolved oxygen in solution.

The control test showed steady levels of iron and arsenic in solution (5% Fe, 4% As), and required 133 kg/t of acid to maintain pH. Dissolution of both elements throughout the test was slow, since there was no bacterial culture to oxidize the concentrate.

For Bioleach test 1, the pH lowered throughout the test, but required 150 kg/t of acid. Aqueous arsenic steadily increased throughout the test, reaching 93.2% dissolution. Aqueous iron slowly increased at first, but eventually reached 81.3% dissolution. This slow dissolution was thought to be due to an inhibitory substance that prevented bacteria from growing at first. For the second test, acetone and ferric pre-washes were used to try to resolve this issue.

Bioleach test 2 showed only 25% iron dissolution, and 40% arsenic dissolution. The use of pre-washes did not mitigate the problem of slow dissolution, and results were worse than in the previous test. It was hypothesized that the inhibitory substance in question was arsenic (III) or arsenic (V), but no conclusive evidence was presented.

13.7 RDI 2004

13.7.1 Batch Acid Pressure Oxidation (POX) Tests

Two series of tests were performed on the concentrate produced by RDI in 2003. One series was performed at 180°C, while the other used 200°C. Oxidation tests were conducted with 10% solids, and used treatment times of 30, 60, and 90 minutes. The concentrate was pre-treated for 2 hours at pH 2 (with sulphuric acid) to remove carbonates prior to oxidation. Results are given in Table 13-14.

Table 13-14 Results of Batch Acid POX Tests – Adapted from RDI 2004

	Baseline	180°C POX			200°C POX		
Oxidation Time (mins)	**0**	**30**	**60**	**90**	**30**	**60**	**90**
H_2SO_4 consumption (pre-treatment, kg/t)		145	136	138	138	121	125
% Sulphide oxidation		98.8	99.4	100	95.1	100	100
% gold recovery (48-hour)	9.4	94.7	98.4	98.2	83.7	98.7	98.0
NaCN consumption (48-hour, kg/t)	1.33	1.18	1.41	1.03	0.79	1.24	1.19

Oxidation was near-complete or complete after just 30 minutes of pressure treatment. No acid was required, and moderate lime consumption was observed (5.7 – 7.8 kg/t) to increase pH after oxidation. Gold recoveries exceeded 98%.

13.7.2 Cyanidation – Bottle Roll Tests

To analyze the efficacy of the pressure oxidation tests described above, bottle roll tests were used on the untreated concentrate, as well as the oxidized residues. Tests were conducted at pH 11, at 33% solids, for 48 hours with 1 g/L NaCN. As seen above, untreated concentrates showed refractory behaviour, due to preg-robbing organic carbon, as well as the refractory nature the concentrate. Over 98% extraction is seen from oxidized samples after high pressure tests. Low cyanide consumptions were observed across the board.

13.8 Oxidor 2004

13.8.1 Bio-oxidation Batch Amenability Tests (BATs)

Bio-oxidation tests were conducted in large 20 L stirred CSTR tanks, sparged with air. Bacterial adaptation was first performed on OXL-1014-R-13 culture. At first, no bacterial activity was observed; however, by day 10, unusually high activity was observed. This delay is thought to be the 2nd piece of evidence for an inhibitory substance at the onset of bacterial oxidation. During the adaptation, Redox potential fell from 825 mV to 590 mV SHE after introduction to the concentrate, and oxygen uptake was significantly lower than at the start of the tests. This may be because of oxidation of arsenic to its pentavalent form. Oxidation levels reached approximately 90% after 22 days oxidation.

13.8.2 Cyanidation - CIL Tests

At various points during the trials described above, a slurry sample would be removed and gold extraction using cyanide would be performed.

Baseline gold recovery results were extremely low: 8.2% without oxidative pre-treatment. After 9 days of oxidation, gold recovery increased to 90%, showing a good response to sulphide oxidation. This result may be anomalous though, and this level was not attained again until 22 – 23 day's oxidation.

13.9 PRA 2007

13.9.1 Gravity Separation Tests

Three gravity separation tests were conducted on the bulk composite sample from New Polaris. Conditions were as follows: 20% solids, 200G force gradient, 1.0 psi backwater pressure, using a Falcon SB40 concentrator. The resulting concentrate was panned to upgrade the gold contents. The tails and pan tails were combined to process via flotation. Results of these tests are shown in Table 13-15.

Table 13-15 Gravity Concentrate Test Results

Test #	P_{80} Size (µm)	Calc. Au, Head (g/t)	Pan Conc.	
			Au Grade (g/t)	Au Recovery (%)
GF1	69	10.3	183.1	3.6
GF2	68	10.7	245.6	5.3
GF3	30	10.8	264.9	5.5

Test GF3 had 100% of the material passing 75 µm. Average gold recovery was 4.8% and the average grade was 231.2 g/t. There was no discernable effect of particle size or head grade on recovery. The low recoveries confirmed the ore is not amenable to gravity concentration.

13.9.2 Rougher Flotation Tests

Rougher flotation was conducted at 35% solids by weight, unless specified. Tests were performed at natural pH. Where indicated, sodium sulphide (Na_2S) was used during the grinding stage. PAX and MIBC were used as collector and frother, respectively. Copper sulphate ($CuSO_4$) was sometimes used as an activator. Sulphur dioxide (SO_2) was used for pyrite/arsenopyrite separation.

13.9.3 Gravity and Pan Tail Flotation

Gravity concentrator and pan tailings from the three tests described above were then assessed using flotation trials. Results of these tests are shown in Table 13-16.

Table 13-16 Gravity Tails Bulk Flotation Test Results – Adapted from PRA 2007

Test #	Test Conditions	Recovery (%)				Grade (g/t, %)			
		Au	As	Sb	S_T	Au	As	Sb	S_T
GF1	pH 8.9 Rougher: 50 g/t PAX, 15 g/t MIBC Scavenger: 50 g/t PAX, 19 g/t MIBC, 250 g/t $CuSO_4$	96.2	82.5	99.5	97.1	33.0	5.73	2.78	9.66
GF2	Grind: 250 g/t Na_2S Rougher: 50 g/t PAX, 15 g/t MIBC Scavenger (2 floats): 50 g/t PAX, 19 g/t MIBC, 250 g/t $CuSO_4$	96.1	82.6	88.8	95.9	32.8	5.60	2.42	9.36
GF3	Rougher: 50 g/t PAX, 19 g/t MIBC Scavenger (2 floats): 50 g/t PAX, 14 g/t MIBC, 250 g/t $CuSO_4$	97.3	83.5	95.8	97.7	30.4	5.23	2.66	9.56

No discernable difference is seen between the test results, with the recovery results from test GF3 slightly higher, and the grade from test GF1 slightly higher. It seems that sulphidization (addition of Na_2S) and particle size did not significantly affect gold recovery or concentrate gold grade.

13.9.4 Arsenic and Antimony Depression Tests

The effect of copper sulphate addition was studied in order to suppress arsenic and antimony from the flotation concentrate. From the results shown in Table 13-17 less antimony and arsenic are seen in the F5 concentrate; however, the gold grade in the concentrate is also lowered. Gold recoveries are lower compared to the original flotation concentrate from Test F1. The conclusions given in the PRA report are that Stibnite and Arsenopyrite are two significant minerals in the deposit that bear gold, so depression of either of these minerals would result in significant gold losses.

13.9.5 Pyrite and Arsenopyrite Separation Tests

The same tests also included floats that attempted to separate Pyrite and Arsenopyrite. Conditions for these tests included a regrind of rougher/scavenger concentrate for test F5, and SO_2 conditioning for test F4. Results indicate incomplete or partial separation is achieved. It appears that some gold is associated with all three minerals arsenopyrite, pyrite, and stibnite. Another result from these tests was the presence of gangue slimes that need to be considered in future testwork.

Table 13-17 As/Sb Depression Flotation Test Results – Adapted from PRA 2007

Test #	Test Conditions	Recovery (%)				Grade (g/t, %)			
		Au	As	Sb	S_T	Au (g/t)	As (%)	Sb (%)	S_T (%)
GF1	pH 8.9 Rougher: 50 g/t PAX, 15 g/t MIBC Scavenger: 50 g/t PAX, 19 g/t MIBC, 250 g/t $CuSO_4$	96.2	82.5	99.5	97.1	33.0	5.73	2.78	9.66
F4	pH 9.3 Rougher: 50 g/t PAX, 17 g/t MIBC Scavenger (2 floats): 50 g/t PAX Cleaner: 8 g/t MIBC Pyrite/Arsenopyrite Sep: 50°C SO_2 at pH 3 for 20 mins, 17 g/t MIBC	87.9	82.3	60.8	91.6	54.9	7.72	2.63	16.9
F5	pH 9.1 Rougher: 50 g/t PAX, 18 g/t MIBC Scavenger (2 floats): 50 g/t PAX Cleaner: 25 g/t PAX 60 minute regrind (pH 8.1) Pyrite/Arsenopyrite Sep (4 floats): 50°C SO_2 at pH 4 for 20 mins, 100 g/t PAX	82.5	81.3	63.2	90.0	33.6	5.24	2.02	10.8

13.9.6 Cleaner Flotation Tests

Tests were conducted to investigate the effect of regrinding rougher concentrates on flotation grades. Three tests were done using cleaner flotation trials: tests F6 and F7 examined reground rougher concentrates using various reagents, while Test F8 tested rougher concentrate that was not reground but was pre-treated with Na_2S during the grinding stage. Results of these tests (found in Table 13-18 and Table 13-19) show that regrind of cleaner concentrates after two floats is unnecessary, since both gold grade and gold recovery is adversely affected. Test 8 shows good recovery and will likely be the basis of future tests.

Table 13-18 Rougher Regrind Cleaner Flotation Test Results – Adapted from PRA 2007

Test #	Test Conditions	Conc. or Tails	Recovery (%)				Grade (g/t, %)			
			Au	As	Sb	S_T	Au (g/t)	As (%)	Sb (%)	S_T (%)
F6	pH 8.9 Rougher: Na_2SiO_3, 10 g/t A3418, 50 g/t PAX, 19 g/t MIBC Scavenger (x3): 15 g/t A3418, 75 g/t PAX, 9 g/t MIBC 20 minute regrind Cleaner (x3): 650 g/t Na_2SiO_3, 10 g/t A3418, 51 g/t PAX, 9 g/t MIBC	Cleaner Conc.	45.3	37.7	51.2	66.7	63.2	8.27	5.99	29.7
		Cleaner Tails	24.7	23.9	6.6	11.4	89.0	13.5	1.99	13.1
		Ro/Scav Bulk	91.6	91.6	71.0	93.2	25.1	3.95	1.63	8.14
F7	pH 8.8 Rougher: Na_3PO_4, 10 g/t A3418, 50 g/t PAX, 19 g/t MIBC Scavenger (x3): 15 g/t A3418, 75 g/t PAX, 9 g/t MIBC 20 minute regrind Cleaner x3): 650 g/t Na_3PO_4, 10 g/t A3418, 51 g/t PAX, 9 g/t MIBC	Cleaner Conc.	47.1	37.6	55.1	65.1	65.2	7.08	6.52	28.9
		Cleaner Tails	16.9	18.8	4.5	7.2	100.2	15.2	2.29	13.8
		Ro/Scav Bulk	90.0	88.5	73.3	93.2	25.4	3.40	1.77	8.45
F8	Grind: 250 g/t Na_2S Rougher: 50 g/t PAX, 13 g/t MIBC Scavenger (x2): 50 g/t PAX, 13 g/t MIBC, 250 g/t $CuSO_4$ Cleaner (x2): 50 g/t Na_2SiO_3, 10 g/t PAX, 15 g/t MIBC	Cleaner Conc.	94.9	89.9	91.8	94.5	85.68	8.98	4.65	18.97
		Cleaner Tails	1.8	4.4	4.0	2.1	2.20	0.59	0.27	0.57
		Ro/Scav Bulk	96.7	94.3	95.8	96.6	49.96	5.39	2.78	11.10

Table 13-19 Test F8 Flotation Test Results – Adapted from PRA 2007

Test #	Test Conditions	Conc. or Tails	Mass Pull (%)	Recovery (%)				Grade (g/t, %)			
				Au	As	Sb	S_T	Au (g/t)	As (%)	Sb (%)	S_T (%)
F8	Grind: 250 g/t Na_2S Rougher: 50 g/t PAX, 13 g/t MIBC Scavenger (x2): 50 g/t PAX, 13 g/t MIBC, 250 g/t $CuSO_4$ Cleaner (x2): 50 g/t Na_2SiO_3, 10 g/t PAX, 15 g/t MIBC	2nd Cleaner Conc.	12.8	93.8	87.4	88.9	93.3	100.61	10.37	5.35	22.26
		2nd Cleaner Tails	2.4	1.1	2.5	2.8	1.2	6.32	1.56	0.90	1.53
		1st Cleaner Conc.	15.2	94.9	89.9	91.8	94.5	85.68	8.98	4.65	18.97
		1st Cleaner Tails	11.3	1.8	4.4	4.0	2.1	2.20	0.59	0.27	0.57
		Rougher Bulk	26.5	96.7	94.3	95.8	96.6	49.96	5.39	2.78	11.10
		Rougher Tails	73.5	3.3	5.7	4.2	3.4	0.61	0.12	0.04	0.14
		Total	100	100	100	100	100	9.70	2.02	0.78	3.18

A second cleaner stage is not necessary for the New Polaris sample, since the grade of the concentrate is not significantly improved, while the gold recovery decreases an additional 2%.

13.9.7 Locked Cycle Flotation Tests

Locked cycle tests were conducted with multiple recycle streams, as shown in the Figure 13-2, taken from the PRA 2007 report.

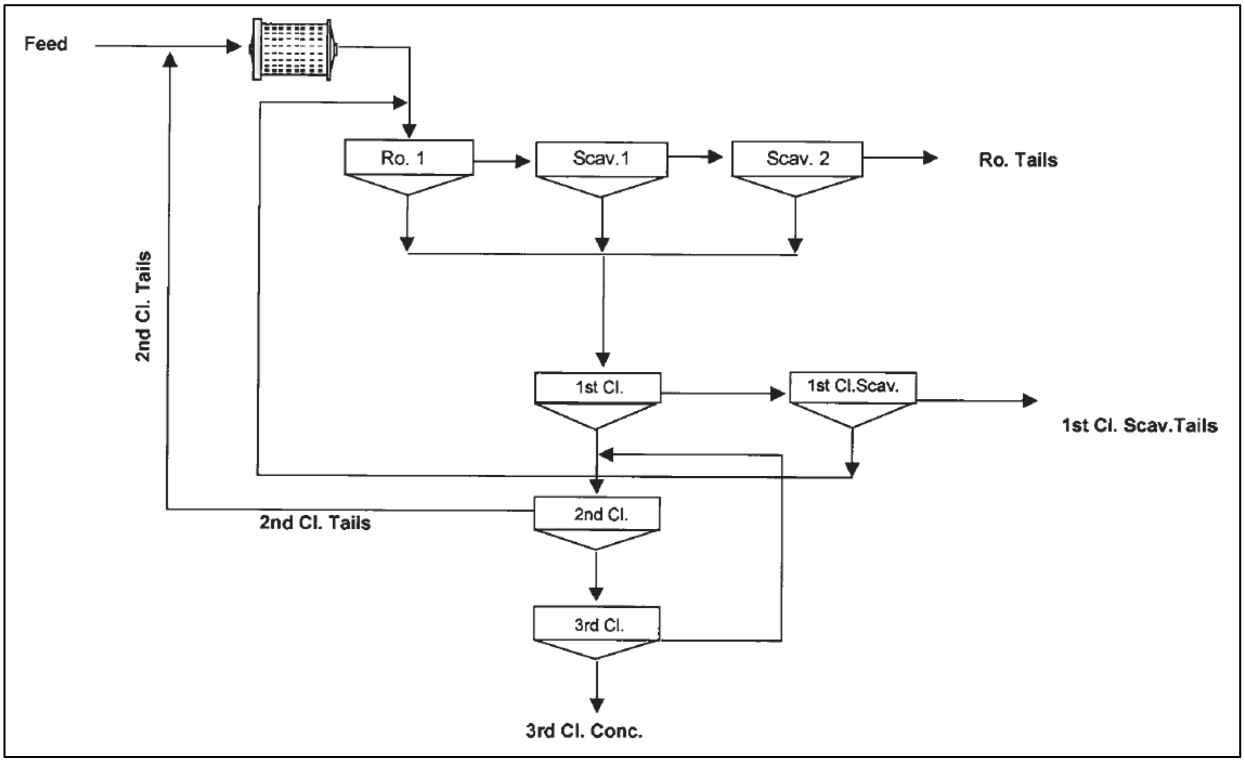

Figure 13-2 Locked Cycle Test Schematic Diagram (Process Research Associated Ltd. (PRA), 2007)

The bulk New Polaris sample was conditioned with 250 g/t sodium sulphide (Na_2S) in the rougher stage at pH 10 and used 500 g/t sodium silicate (Na_2SiO_3) in the 1st cleaner stage without a regrind. Each test was run for five cycles to monitor how gold recovery and grade reacted with multiple passes through the flotation stages. The gold recoveries and grades after each cycle is shown in Figure 13-3.

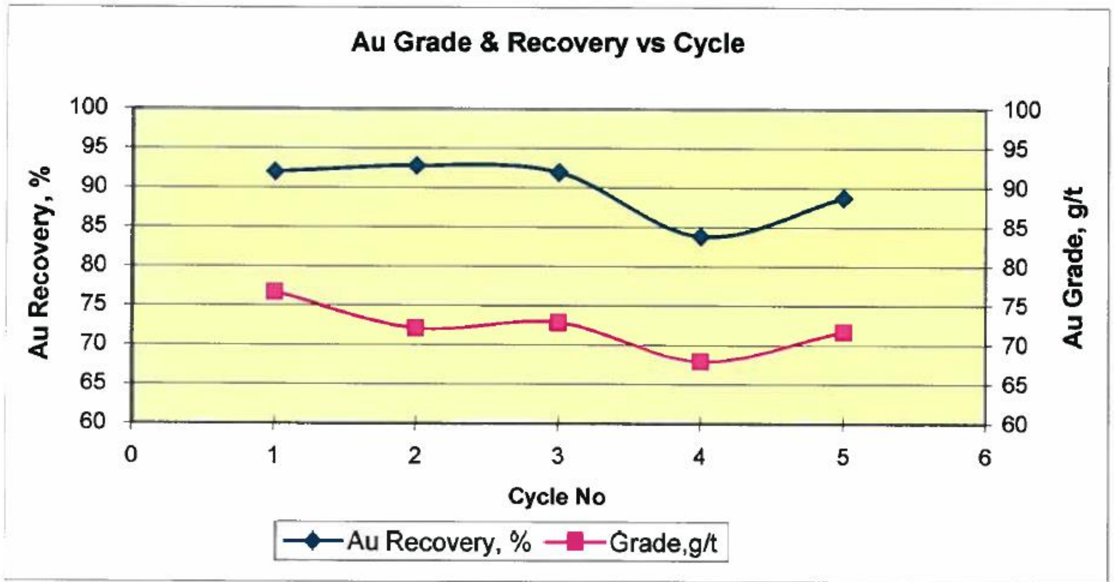

Figure 13-3 Locked Cycle Test Results (Process Research Associated Ltd. (PRA), 2007)

As shown above, the gold recovery of the system is maintained at around 92% through three cycles, after which a decrease occurs to about 87%. The grades of the final concentrate steadily decreases from approximately 76% to 71% after 5 cycles. Results are given in Table 13-20.

Table 13-20 Locked Cycle Flotation Test Results – Adapted from PRA 2007

Test #	Test Conditions	Conc./Tails	Mass Pull (%)	Recovery (%)				Grade (g/t, %)			
				Au	As	Sb	S_T	Au (g/t)	As (%)	Sb (%)	S_T (%)
F9LC	Rougher: 250 g/t Na_2S, 50 g/t PAX, 12 g/t MIBC Scavenger (x2): 50 g/t PAX, 12 g/t MIBC, 250 g/t $CuSO_4$ Cleaner (x3): 500 g/t Na_2SiO_3, 15 g/t PAX, 27 g/t MIBC	Final Conc.	12.8	91.8	90.8	86.9	94.0	72.3	10.4	5.34	22.7
		Cleaner Tails	12.0	2.6	2.9	7.8	2.1	2.14	0.36	0.51	0.53
		Bulk Tails	75.2	5.7	6.3	5.3	3.9	0.76	0.12	0.05	0.16
		Head	100	100	100	100	100				

The locked cycle tests used more cleaning stages than required as shown in the prior cleaning tests.

13.9.8 Slurry Settling Tests

Two slurry settling tests were conducted on the bulk tails from Test F6. Percol 156 was dosed in one test at pH 11 and compared to the test without flocculant. Results of the two tests show that tails without flocculant added had a settling rate of 2.7 m/day. By contrast, tails which had flocculant dosed had a settling rate of 21.8 m/day. Both tests had pulp densities that started at 21% and ended at 71%, but the rate at which the tails settled were improved greatly by the presence of the flocculant in the second test. Settling test results are shown in Figure 13-4.

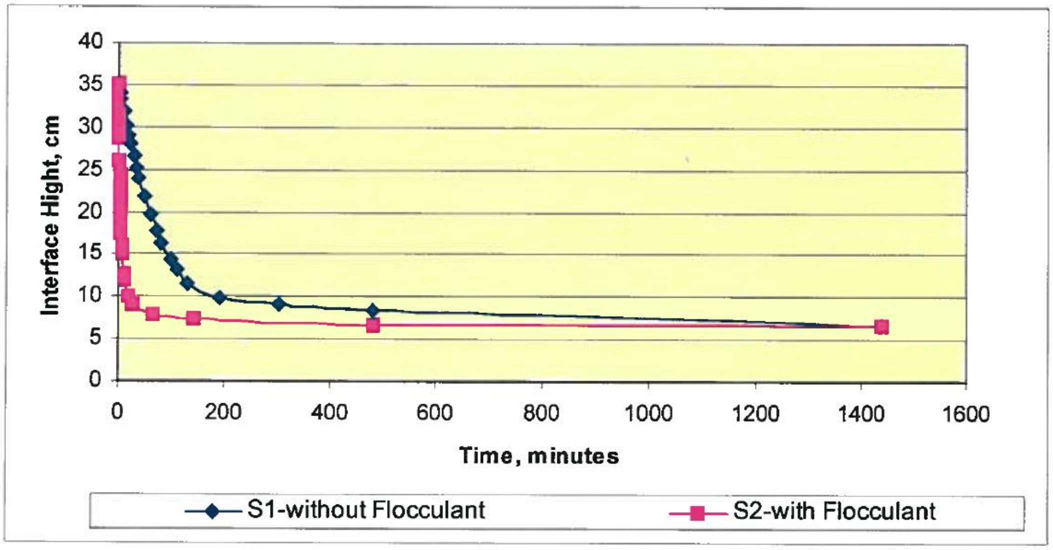

Figure 13-4 Slurry Settling Test Results (Process Research Associated Ltd. (PRA), 2007)

13.9.9 Cyanidation – Bottle Roll Tests

Bottle roll tests without pre-oxidation were conducted at pH 10.5 – 11, with 1.0 g/L NaCN and 40% solids. Total leach time was 72 hours, with intermediate samples at 6, 24, and 48 hours. Results show that only 5.4% gold recovery was achieved after 72 hours. Another cyanidation test was conducted on the concentrate produced with Test F4 and F5 conditions (37 µm) with slightly better results.

13.9.10 Cyanidation – High Intensity Tests

Cyanide gold leaching was also conducted on the concentrate produced by Tests F4 and F5, with a P_{80} of 37µm. Excess leaching reagents and more amenable test conditions were used (10 g/L NaCN, 20% solids) to leach the concentrate for 72 hours as before. Results show that approximately 10.6% of the gold was leached from this concentrate, showing the need for oxidative pre-treatment.

13.10 Outotec 2018

13.10.1 BIOX™ Batch Amenability Tests (BATs)

Outotec conducted a testing campaign to determine the amenability of bio-leaching the New Polaris flotation concentrate using the Outotec BIOX™ technology. Tests were conducted using 250 g of material (80% passing 75 µm) in a "9K" nutrient broth at 7.7% solids at pH 1.5 (with sulphuric acid). The samples were then inoculated with a bacterial culture (applied at 10% solids) and kept between 38 – 42°C at a pH of 1.2 – 1.4. Bacterial activity was then monitored, and more ore was added once activity increased to desirable levels until 20% solids was achieved in the conditioning reactor. Once the ferrous iron levels reduced to 0.1 g/L in solution, 300 mL of the slurry was extracted and transferred to a 3 L beaker with "0K" nutrient broth, and 20% solids was then achieved for the build-up phase.

Tests were stopped at different times, and slurry samples were extracted for analysis and cyanide gold leaching. Bio-leaching tests were conducted in 5 L stirred and aerated tanks, at pH 1.2 – 1.4, and 20%

solids at 40 ± 2°C. Dissolved oxygen levels were maintained above 2 mg/L, and the slurry was stirred at 460 rpm. Throughout the tests, ferrous and ferric iron, dissolved oxygen, pH and Redox potential were monitored. Results were collated and compared to the gold recoveries obtained from cyanidation in Table 13-21.

13.10.2 Cyanidation – Bottle Roll Tests

Bottle roll tests were conducted on the BIOX™ residues that were extracted from the tank at different times throughout the bio-leaching phase. Cyanidation was conducted for 24 hours at 20% solids, at pH 11 (using a 100 g/L lime slurry), with 20 g/L activated carbon and 20 kg/t NaCN. Results are given in Table 13-21.

Table 13-21 Summary of BIOX™ BAT and Bottle Roll Tests – Adapted from Outotec 2018

BAT #	BIOX™ time (days)	Sulphide Oxidation (%)	Arsenic Sol'n (%)	Iron Sol'n (%)	Au Dissolution (%)		Consumption (kg/t)	
					Residue	Calculated Head	NaCN	Lime
0	0	0	0	0	8.1	12.7	6.5	1.1
6	12	89.6	65.7	70.6	86.2	86.7	14.0	21.8
5	13	94.7	70.5	79.2	90.8	91.3	11.8	25.8
2	14	93.9	72.9	73	90.2	89.3	13.8	30.7
7	17	97.9	73.9	77.6	90.0	89.2	11.7	21.4
4	20	97.3	75.2	74.9	93.7	93.3	14.2	28.1
1	22	99.0	69.0	74.3	96.3	96.0	11.3	22.1
3	22	99.1	66.4	73.3	95.1	94.9	13.2	24.0
Final	22	99.05	67.4	73.8	95.7	95.45	12.25	23.05

Results show that sulphide oxidation was initially very slow. However, after 22 days, bio-oxidation of the sulphides in the New Polaris concentrate was over 99% complete, and subsequent gold extraction is over 95%. Cyanide consumption was quite high, at 12.25 kg/t concentrate.

13.10.3 BIOX™ Liquor Neutralization Tests

Outotec conducted three neutralization tests on the liquor from BAT 1, to determine if the leach liquor from the BIOX™ leaching process could be processed to remove harmful elements such as arsenic or ferric iron. TCLP (Toxicity Characteristic Leaching Procedure) tests were undertaken on the solid products (residues) that were produced to determine their long-term stability in natural environments.
The first test used both lime (100 g/L) and limestone (200 g/L) slurries to neutralize the liquor. The second test used a lime slurry only. The final test replicated the conditions of Test 1, however it also

included addition of ferric sulphate to ensure the iron to arsenic ratio in the liquor was maintained above 3.2. This proportion of ferric iron to arsenic is necessary to produce neutralization products that are more stable in the long term.

Counter-current decantation (CCD) tests were performed, with flocculant being dosed four times in 15-minute intervals at first, and then four 60-minute intervals. Settling times were maintained at 30 seconds, and the slurry was diluted to 7% solids. The flocculants were made to 0.05% strength solutions (from 100 g/L stock solutions). Results of these tests are shown in Table 13-22.

Table 13-22 Summary of Neutralization Tests – Adapted from Outotec 2018

		Neutralization Products					
Test #	Test Conditions	Solids (%)		Solution (mg/L)		TCLP (mg/L)	
		Fe	As	Fe	As	Fe	As
1	Lime + Limestone slurry	15.7	5.8	0.1	0.3	< 0.1	10
2	Lime slurry	14.8	6.1	0.4	0.4	< 0.1	3.9
3	Lime + Limestone + Ferric Sulphate	16.1	5.0	< 0.1	0.4	< 0.1	2.3

Test results show that after neutralization the liquor from BAT 1 passed EPA limits on aqueous iron and aqueous arsenic in solution, with all tests showing less than 5 mg/L iron and arsenic in solution. TCLP test results also showed that neutralization products exhibited long-term stability, except for Test 1, which allowed 10 mg/L of arsenic to leach from the solid residue. This was rectified in Test 3, with ferric sulphate added to the liquor before neutralization.

13.10.4 BIOX™ Slurry Settling Tests

Outotec conducted slurry settling tests on the residue from BAT 3. The purpose of these tests was to select an appropriate flocculant to settle solids after BIOX™ leaching, as well as determine an adequate dosage for the chosen flocculant. For these tests, 33 mL of the BIOX™ slurry was dosed into a 100 mL cylinder, with 67 mL water and flocculant. Two inversions were then performed quickly, and a third inversion gently. The residue "mud layer" was marked over time. To choose the appropriate flocculant, the clarity of the liquor was monitored, and the settling rate in the cylinder was recorded. Results of the tests are as shown in Table 13-23.

Table 13-23 Summary of Flocculant Screening Tests – Outotec 2018

	Flocculant Tested (100 g/t)							
	Magna 10	Magna 333	Magna 345	Magna 455	Magna 336	Magna 156	Magna 1011	Magna 405
Settling time (sec)	30	30	30	30	30	30	30	30
Displacement (mm)	5	3	3	3	3	3	5	13
Settling Rate (m/h)	0.60	0.36	0.36	0.36	0.36	0.36	0.60	1.56
Overflow Clarity	Cloudy	Cloudy	Cloudy	Cloudy	Cloudy	Cloudy	Cloudy	Clear

From these results Magna 405 was chosen for the clear overflow stream it produced, as well as the high settling rates that were observed. To conduct the dosage optimization tests, 500 mL cylinders were used, and flocculant solutions were diluted to 25 g/L from 250 g/L stock solutions. Inversions were made as described above, and settling rates were then recorded for each dosage. Results of these tests are taken from Outotec and presented in Figure 13-5.

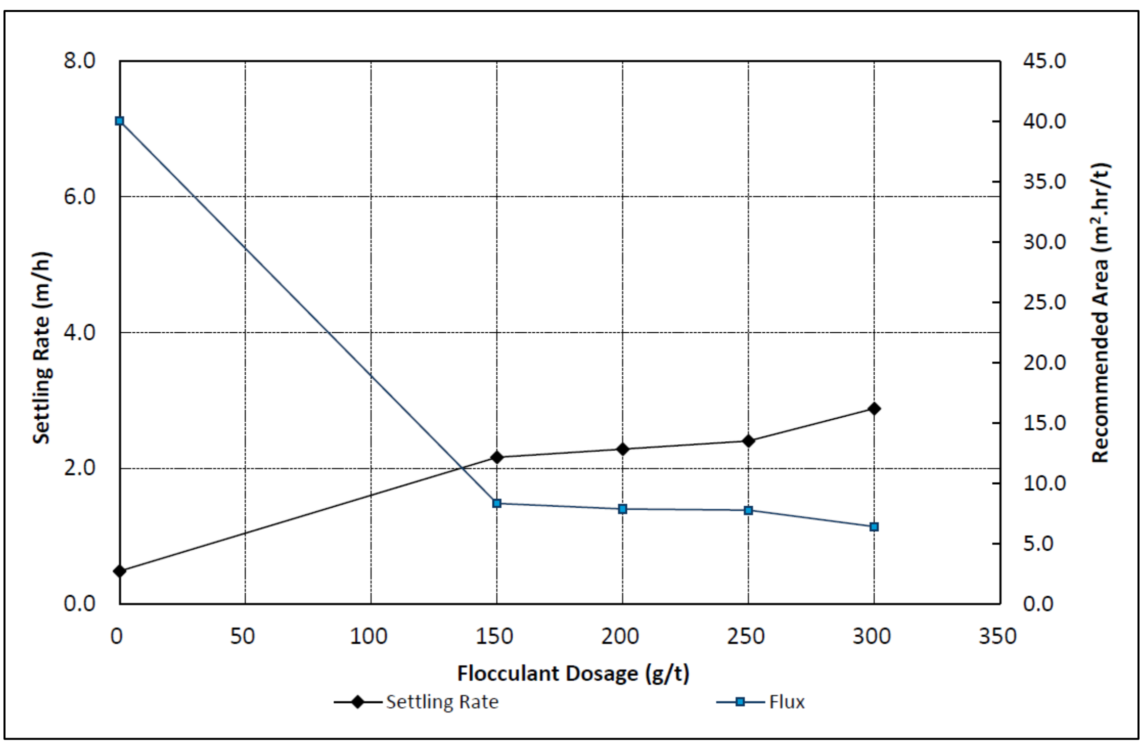

Figure 13-5 Slurry Settling Tests on BIOX™ Products (Outotec RS (PTY) LTD, 2018)

Results indicate that 150 – 250 g/t flocculant dosage were enough to produce a settling rate of 2.3 m/h, and a thickener area flux of 8 m^2h/t.

A second series of tests was undertaken to analyze flocculants for the settling of neutralized slurry.

Results of these are as shown in Table 13-24.

Table 13-24 Summary of Flocculant Screening Tests – Outotec 2018

	Flocculant Tested (100 g/t on 7.6% solids)					
	Magna 10	Magna 333	Magna 345	Magna 455	Magna 336	Magna 405
Settling time (sec)	45	45	45	45	45	45
Displacement (mm)	7	6	14	10	7	6
Settling Rate (m/h)	0.56	0.48	1.12	0.80	0.56	0.48
Overflow Clarity	Clear	Clear	Clear	Clear	Clear	Clear

In comparison to the first settling tests, which used the BIOX™ slurry as it was directly from the leaching reactor, the above results show the settling results after the slurry had been neutralized to a pH of 7. The best flocculant was Magna 345. Dosage optimization was carried out; results shown in Figure 13-6.

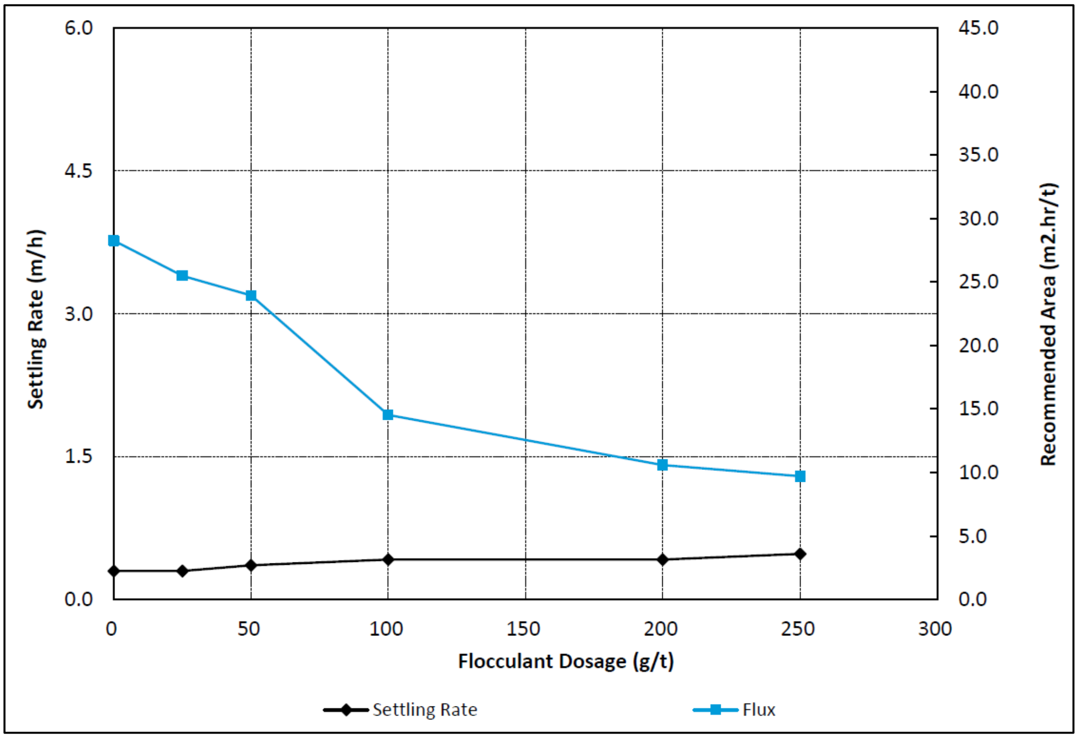

Figure 13-6 Slurry Settling Test Results on BIOX™ Residues (Outotec RS (PTY) LTD, 2018)

Results show that a dosage of 250 g/t will result in a settling rate of approximately 0.5 m/h, which translates to a thickener area flux of 10 m^2h/t. Thus, the two flocculants for liquor neutralization and their dosages are: 250 g/t Magna 405 for un-neutralized slurry, and 250 g/t Magna 345 for a neutralized slurry.

13.11 Recommended Flowsheet

The preliminary flowsheet for the New Polaris project is given below in Figure 13-7.

Test work has demonstrated that both BIOX and POX are potential pre-oxidation process options for New Polaris. BIOX has been selected by Canarc as the base case treatment route due to the lower capital cost and ease of operation compared to a POX circuit.

13.12 Metallurgical Performance Projections

Various process stage recoveries are listed in Table 13-25.

Table 13-25 New Polaris Projected Metallurgical Recoveries

Area	Recovery (%)
Sulphide Flotation	94.9
BIOX and CIL Leach	95.6
Carbon Loss	0.1
EW	99.9

An overall gold recovery for the process flowsheet in Figure 13-7 is estimated at 90.5%.

13.12.1 Deleterious Elements and Other Factors

To the extent known there are no additional processing factors or deleterious elements that could have a significant effect on the potential economic extraction of gold at New Polaris, other than those already mentioned in this Section.

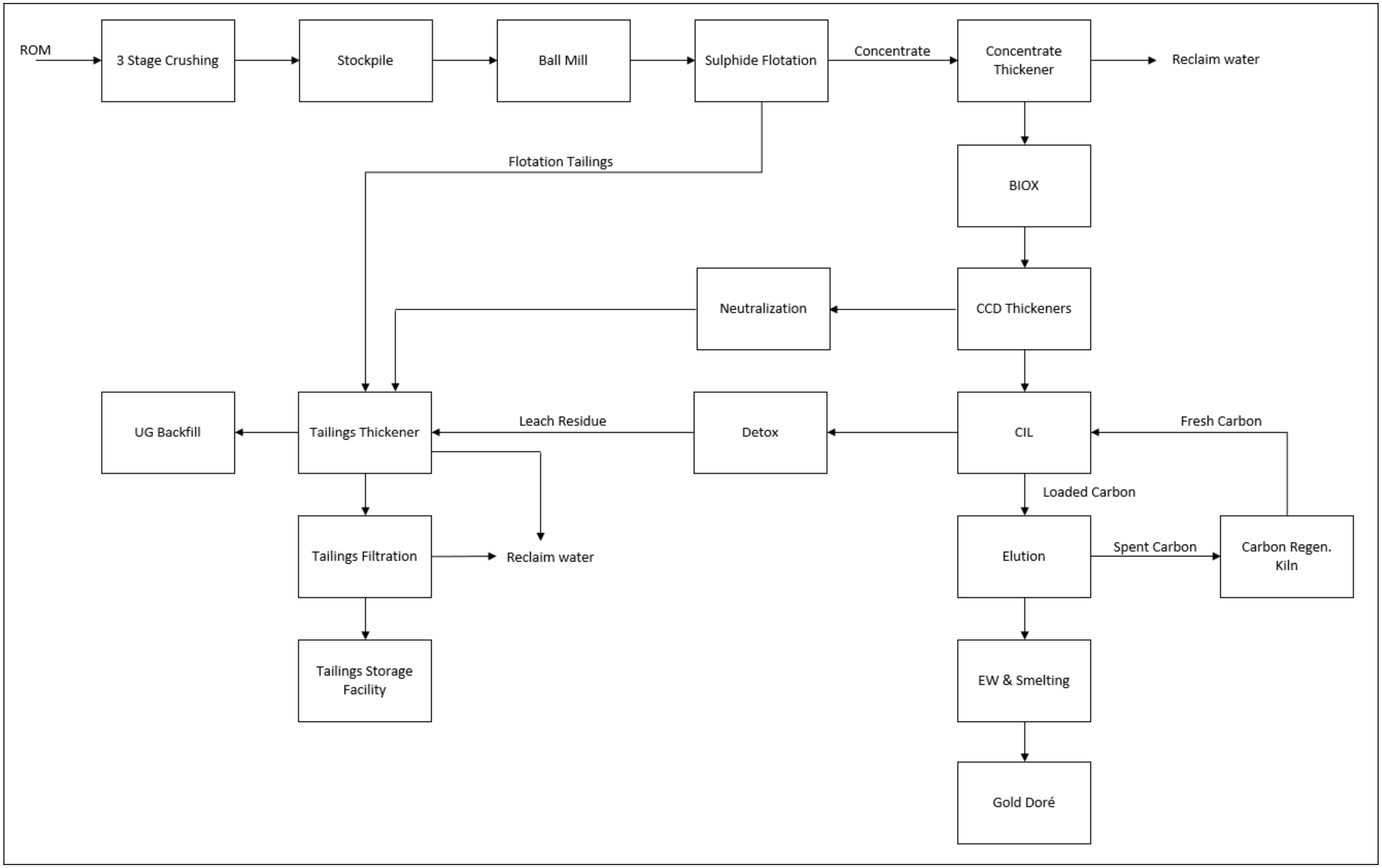

Figure 13-7 New Polaris Process Flowsheet

14 Mineral Resource Estimates

14.1 Introduction

The Mineral Resources for the New Polaris Project have been updated with revised estimates by Sue Bird, P. Eng of MMTS in accordance with updated Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (CIM 2014). Updated CIM standards have resulted in changes to the Classification based on QA/QC. There are changes to the resource tonnage and grade based on updated prices, recoveries and on additional factors to control the "reasonable prospects of eventual economic extraction" including a minimum mining width and an applied underground shape confirming the Resource Estimate.

14.2 Resource

The Resource Estimate for the New Polaris deposit is summarized in Table 14-1. The resource has been summarized at various cutoff grades with the base case Au grade cutoff of 4.0 g/t highlighted. At each cutoff the total material within a potential confining mining shape is reported. Therefore, a separate mining shape has been created for each cutoff in the table.

The base case cutoff grade of 4.0 g/t Au is based on the following economic considerations: gold price of US$1,300/oz, exchange rate of 0.77 US$:C$; Payable gold % of 99.9%, Offsite refining costs of US$7/oz, mining costs of C$65.20/t, process costs of C$62.70/t, G&A (General and Administration) costs of C$37.00, sustaining capital costs of C$19.83/t, and a 90.5% process recovery.

The "reasonable prospects for eventual economic extraction" confining shape also considers a minimum mining width of 2.0 m, and removes shapes considered too small and separated from the primary mining volumes. Previous underground mining has been accounted for by using stope and development solids to code a percent of the block outside of the mined out shapes.

MMTS is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that could materially affect the Mineral Resource Estimate. Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.

The effective date of this Resource estimate is February 28, 2019.

Table 14-1 Summary of Indicated and Inferred Total Resource

Confining Shape Target Grade - (g/t Au)	In Situ Tonnage (Ktonnes)	In Situ Grades AU (g/t)	Au (koz.)
Indicated			
2.0	1,880	10.0	605
3.0	1,798	10.4	599
4.0	**1,687**	**10.8**	**586**
5.0	1,556	11.3	567
6.0	1,403	12.0	540
7.0	1,260	12.6	509
8.0	1,105	13.3	472
9.0	947	14.1	428
Inferred			
Confining Shape Target Grade - (g/t Au)	**In Situ Tonnage (Ktonnes)**	**In Situ Grades AU (g/t)**	**Au (koz.)**
2.0	1,639	9.5	502
3.0	1,582	9.8	497
4.0	**1,483**	**10.2**	**485**
5.0	1,351	10.7	464
6.0	1,223	11.2	441
7.0	942	12.5	380
8.0	753	13.8	334
9.0	653	14.6	306

Notes for Mineral Resource Estimate:

- *The Mineral Resource Estimate was prepared by Sue Bird, P.Eng. in accordance with CIM Definition Standards and NI 43-101, with an effective date of February 28, 2019.*
- *Mineral Resources that are not mineral reserves do not have demonstrated economic viability.*

14.3 Database and Assumptions of the Resource Estimate

The database for the New Polaris resource estimate consists of 1,056 diamond drill holes with a total of 11,590 sample intervals assayed for Au. Table 14-2 summarizes the break-down of drillholes and assays from the total, to those used in the resource estimation. Six veins have been modelled, with Vein 5 split into two Domains for modelling due to changing orientation.

Only data since 1988 has been used to estimate the Au grade because previous data is historic, and no information is available on the drilling year, owner, or sampling security and analysis.

Table 14-2 Summary of Assay Data in the Project Area

Source	Drillholes	Total Length (m)	Au Assay Intervals above Zero	Au Assay Length (m)	Percent Assayed and above Zero
Project Total	1056	106,458.6	11,590	11,850.7	11%
Block Model Total	709	85,546.0	9,645	8,481	10%
Within block model and Since 1988	286	73,898.1	7,895	8,393.2	11%
Domain Total	173	50,704.0	1,357	1,101.7	2%
Domain Total since 1988	143	47,042.4	1,287	1,033.2	2%

A summary of assay intervals and length for the 6 modelled veins is shown in Table 14-3. Two additional veins (3 and 4) have been interpreted geologically but have not been considered in the resource estimate due to lack of drilling for these domains.

Table 14-3 Summary of Assay Data within the Domains used for Resource Estimation

Vein	Domain	Au Assay Intervals above Zero	Au Assay Length (m)
1 (CHIE)	11	183	141.1
2 (CLOE)	21	402	315.3
5 (CWM)	51 and 52	429	343.5
6 (GEO6)	61	168	158.0
7 (Y19)	71	77	54.6
8 (Y20)	81	29	20.7

There is QA/QC available only for drilling done in 2006. Therefore, to validate the data from 1988 through 2005, the assay statistics have been compared to the 2006 data. Figure 14-1 shows and cumulative probability plots (CPPs) of the Au grade distribution by Year, indicating no obvious bias in the drilling data since 1988. Data prior to 1988 shows a higher grade distribution. This may be due to areas drilled, as the higher grades now mined out were drilled in these earlier years. However, the pre-1988 data has not been used since it could not be validated.

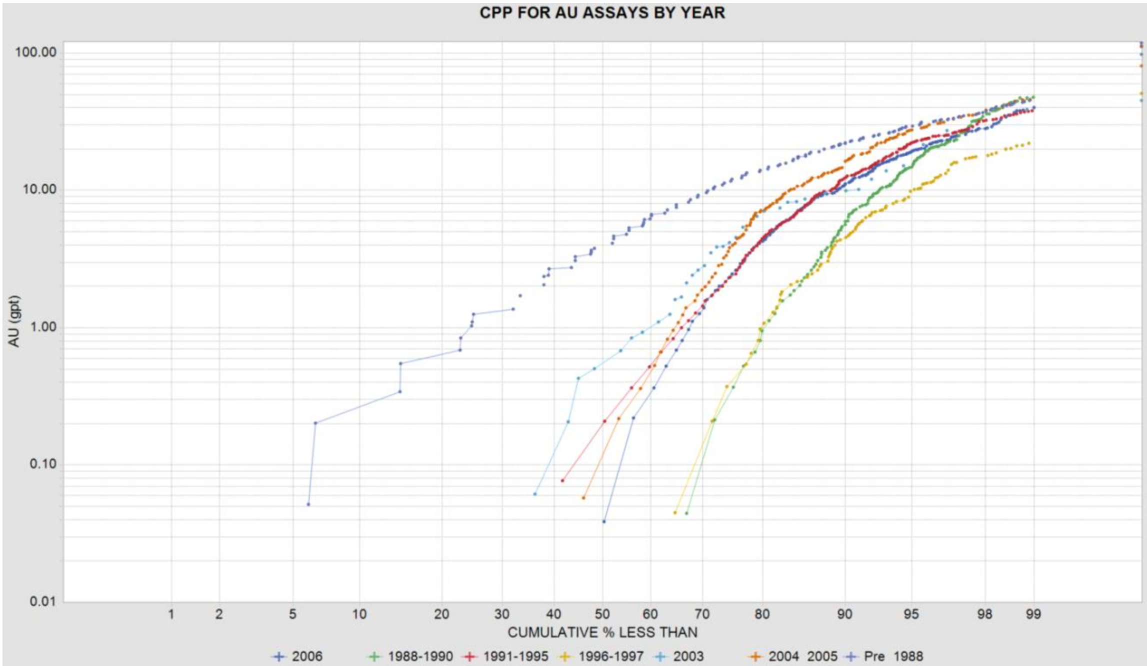

Figure 14-1 CPP of Assay Data by Year

Figure 14-2 illustrates all the drillholes within the block model bounds that intersect the interpreted veins. The coding of the assay data and the solids used to code the data is illustrated.

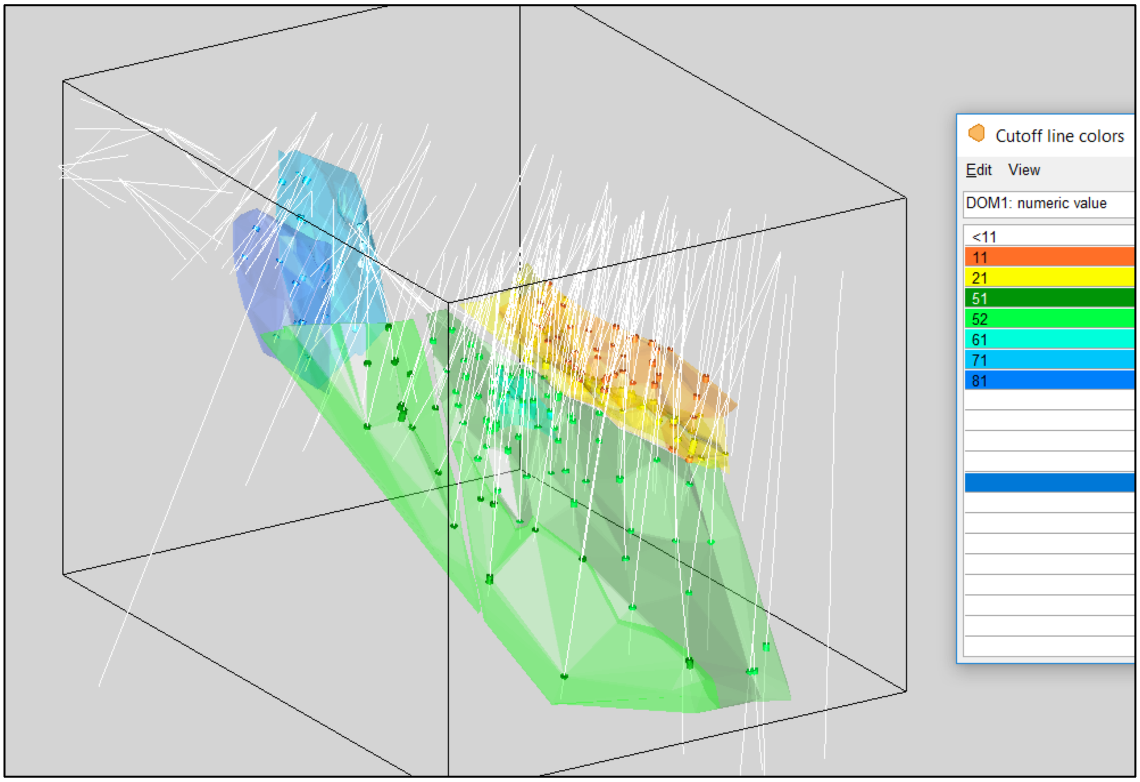

Figure 14-2 Drillholes used and Domains, 3D orthogonal view looking Northeast

14.4 Assay Statistics, Capping and Compositing

MMTS has examined the available assay statistics using boxplots, histograms, and cumulative probability plots (CPPs). The gold grade distribution is shown below in Figure 14-3 as CPPs by Domain. The grade distribution for Au is mainly lognormal except at very high grades where outliers are evident and therefore capping of assays is done. Capping values for Au assays are summarized in Table 14-4.

Table 14-4 Summary of Capping Value by Vein for Gold

Vein	Vein #	Au Capping Value (g/t)
CHIE	1	---
CLOE	2	70
CWM5	5	60
GEO6	6	50
Y19	7	60
Y20	8	---

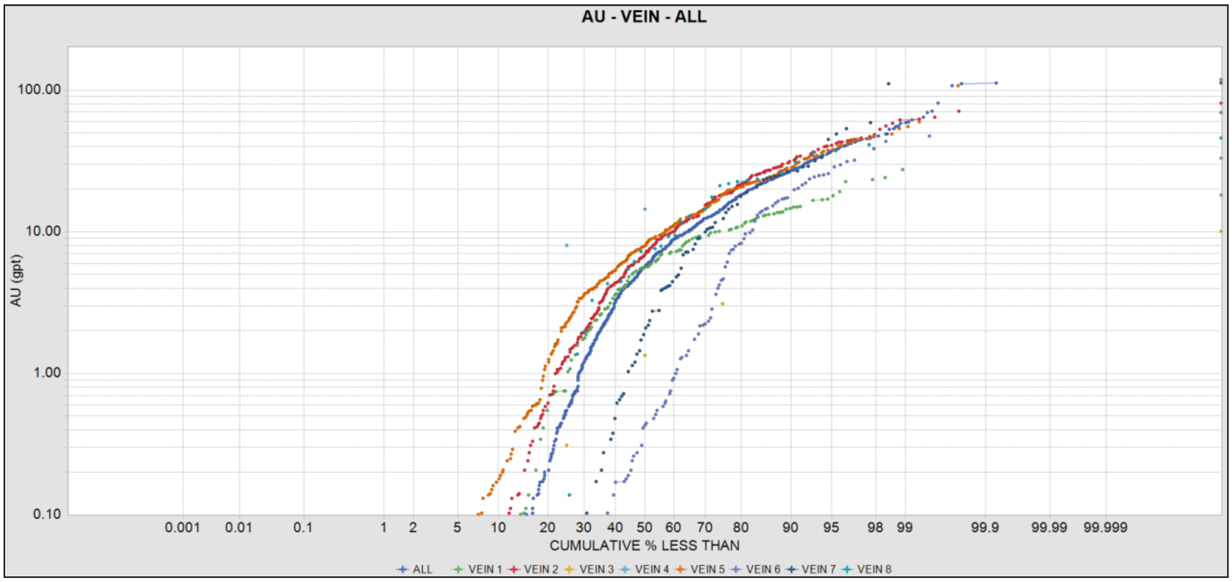

Figure 14-3 New Polaris Veins – CPP Plot -Au

Assay sample lengths have varied with the drill programs. A histogram of the assay intervals for the New Polaris deposit is shown in Figure 14-4. The majority of sampling has been done using a 1.5 m interval length. Therefore, this is the value that has been used for the base length when compositing. The compositing also honored the domain boundaries. Assay intervals less than 0.75 m have been added to the previous composite. To ensure correct compositing the length weighted mean grades of the composite are compared to the original assay data. Assay and composite statistics are summarized in Table 14-5 showing negligible differences in grade values between composites and assays coded with a domain. Theses statistic also indicated that the Coefficient of Variation (C.V.), at values primarily around 1.0 and not over 1.85 are appropriate for using a linear interpolation method.

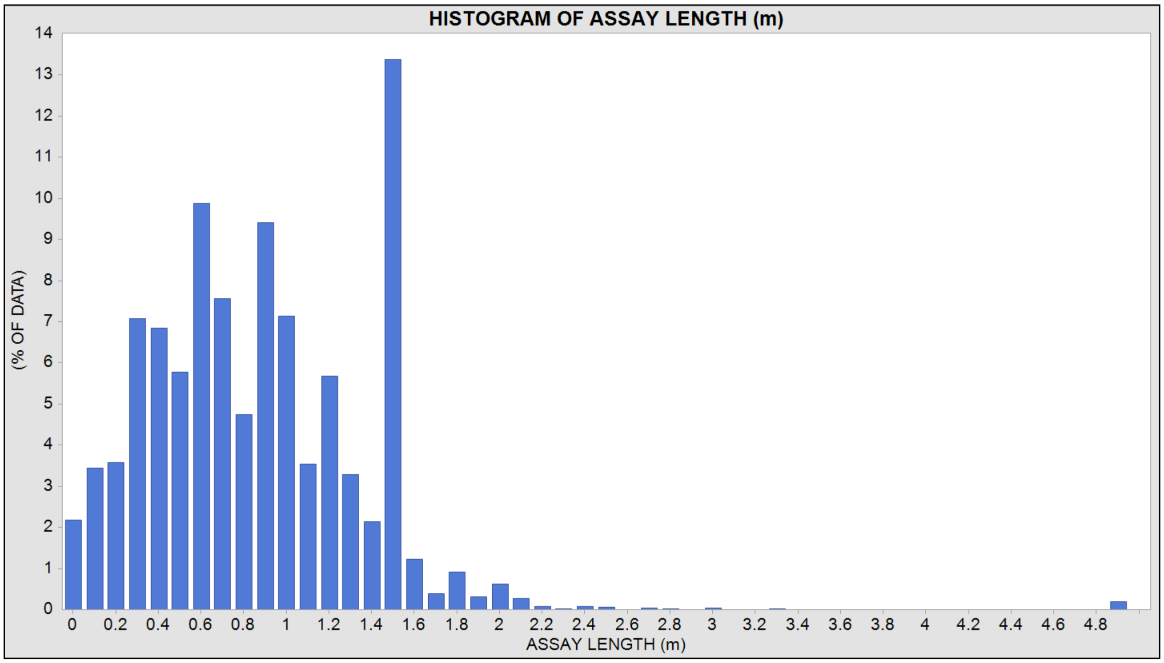

Figure 14-4 Histogram of Assay Lengths within Mineralized Zone – New Polaris

Table 14-5 Summary of Assay and Composite Statistics

Source	Parameter	Vein						
		1	2	5	6	7	8	ALL
Assays	Num Samples	186	440	450	191	133	43	1443
	Num Missing Samples	7	7	1	5	1	0	21
	Min (g/t)	0	0	0	0	0	0	0
	Max (g/t)	33.02	80.61	118.50	69.60	111.94	45.94	118.50
	Weighted mean (g/t)	6.71	10.73	11.28	4.96	6.72	12.03	9.18
	Weighted SD	6.35	13.04	14.07	10.18	14.61	12.68	12.79
	Weighted CV	0.95	1.22	1.25	2.05	2.18	1.05	1.39
Comps	Num Samples	110	282	292	139	100	32	955
	Num Missing Samples	21	10	3	6	0	0	40
	Min	0	0	0	0	0	0	0
	Max	27.60	56.52	118.50	57.07	57.70	42.23	118.50
	Weighted mean	6.72	10.66	11.28	4.91	6.68	12.03	9.14
	Weighted SD	5.43	10.89	11.78	9.11	10.95	11.11	10.76
	Weighted CV	0.81	1.02	1.04	1.85	1.64	0.92	1.18
Mean Au Grade Difference (%)		0.2%	-0.6%	0.0%	-1.0%	-0.5%	0.0%	-0.4%

14.5 Variography

Correlograms have been used to determine appropriate search distances and anisotropy. In all cases correlogram have been used and spherical models were fit to the experimental data. Domain 5 was split into an eastern segment and a steeper dipping western segment. The parameters for all models are summarized in Table 14-6.

Table 14-6 Variogram Parameters

Domain	Azimuth	Dip	Axis	Total Range (M)	NUGGET	SILL 1	SILL 2	Short Range (m)	Long Range (m)
11	ROT	70	Major	100	0.3	0.4	0.3	20	100
	DIPN	0	Minor	120				40	120
	DIPE	-45	Vertical	90				10	90
21	ROT	70	Major	70	0.2	0.7	0.1	40	70
	DIPN	0	Minor	90				50	90
	DIPE	-45	Vertical	60				40	60
51 - EAST	ROT	110	Major	60	0.4	0.3	0.3	20	60
	DIPN	-10	Minor	70				15	70
	DIPE	-70	Vertical	40				10	40
52 - WEST	ROT	110	Major	60	0.4	0.3	0.3	20	60
	DIPN	-10	Minor	70				15	70
	DIPE	-80	Vertical	40				10	40
61	OMNI DIRECTIONAL			60	0.6	0.3	0.2	10	60
71	ROT	0	Major	80	0.4	0.4	0.2	60	80
	DIPN	0	Minor	70				30	70
	DIPE	-85	Vertical	30				10	30
81	ROT	340	Major	80	0.4	0.4	0.2	60	80
	DIPN	0	Minor	70				30	70
	DIPE	-90	Vertical	30				10	30
Waste	OMNI DIRECTIONAL			80	0.7	0.2	0.1	20	80

14.6 Block Model

Table 14-7 summarizes the block model extents for the New Polaris resource estimate. The block model is considered a multiple percent block model allowing up to three domains and their corresponding percentage within each block. Figure 14-5 illustrates the three dimensional blocks coded by the majority domain code.

Table 14-7 Block Model Parameters

New Polaris Model Parameters				
DIRECTION	**MINIMUM**	**MAXIMUM**	**BLOCK SIZE**	**# BLOCKS**
EASTING	1350	2100	5	150
NORTHING	500	1500	5	200
ELEVATION	-600	130	5	146

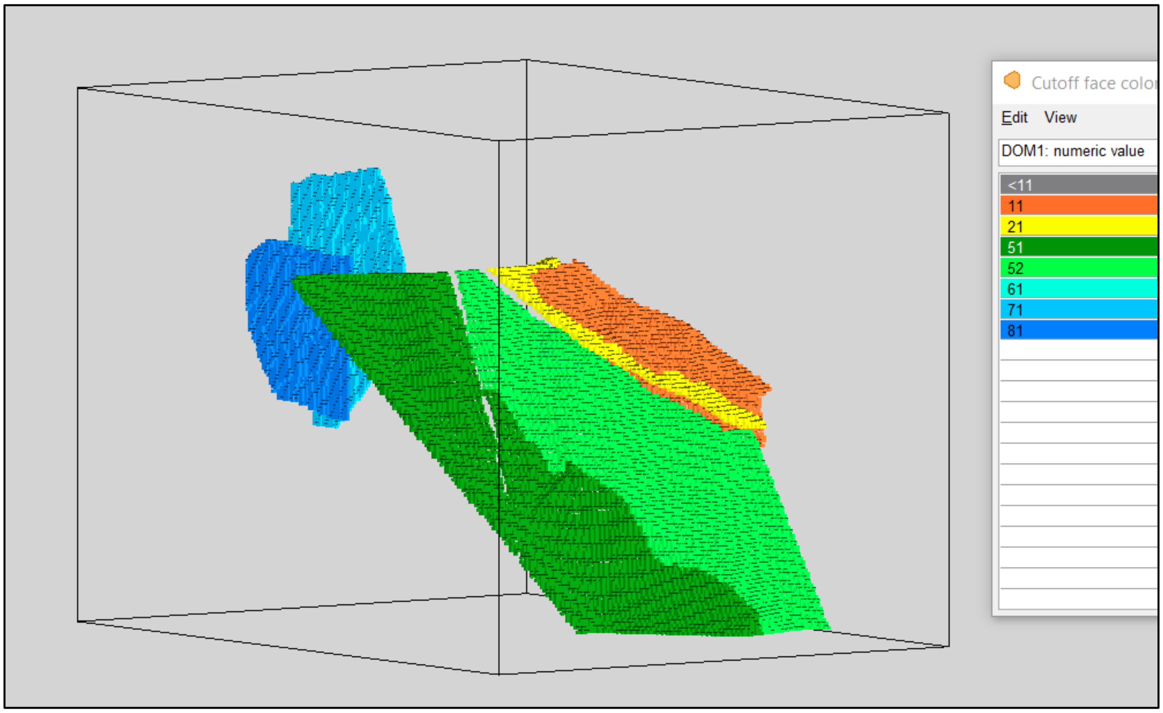

Figure 14-5 Model Extents (black) and the Blocks coded by Majority Domain, 3D Orthogonal View looking Northeast

Interpolation Parameters

Each domain has been interpolated by ordinary kriging (OK) separately, with sharing between domains only allowed for Domain 51 and 52, which belong to the same vein and are defined separately due to a change in orientation of the vein. These two parts of Domain 5 have been estimated using a soft boundary with both sides of the boundary seeing the same composites.

Interpolation has been done in 4 passes with the search distances increased for each pass. The search ellipse directions and distances in each of the principle axes for each pass are shown below in Table 14-8. The sample selection criteria varied by pass as well, with the parameters used summarized in Table 14-9.

Table 14-8 Summary of Search Orientation and Distance for each Interpolation Pass

Domain	Rot	Dist1	Dist2	Dist3	Dist4
1	70	20	75	100	175
	0	30	90	120	210
	-45	10	67.5	90	157.5
2	70	17.5	52.5	70	122.5
	0	22.5	67.5	90	157.5
	-45	15	45	60	105
5 - EAST	110	15	45	60	105
	-10	15	52.5	70	122.5
	-70	10	30	40	70
5 - WEST	110	15	45	60	105
	-10	15	52.5	70	122.5
	-80	10	30	40	70
6	0	10	45	60	105
7	0	20	60	80	140
	0	17.5	52.5	70	122.5
	-85	7.5	22.5	30	52.5
8	340	20	60	80	140
	0	17.5	52.5	70	122.5
	-90	7.5	22.5	30	52.5
Waste	0	20	60	80	140

Table 14-9 Sample Selection Criteria during Interpolation

Search Parameters	Search Distance			
	Pass 1	Pass 2	Pass 3	Pass 4
Min. # Comps	6	6	6	2
Max. # Comps	12	12	12	6
Max. # Comps/DH	3	3	3	2
Max. # / Split Quadrant	6	6	6	6

14.6.1 Classification

Classification of the resource is according to the definitions in National Instrument 43-101 and CIM (2014). Classification is based on the variography, drillhole spacing, true thickness and continuity of blocks within each possible classification. The criteria are summarized in Table 14-10. There are no blocks considered measured due to lack of QA/QC for all data except the 2006 drilling, as well as due to the spottiness of possible blocks which could be considered Measured based on drillhole spacing.

Table 14-10 Classification Criteria

Criteria	Explanation	Classification
True thickness > 2m	Minimum mining width	Inferred
Avg. Distance to 2 drillholes <=30m	Range at approximately 80% of the sill of the variogram	Indicated
Distance to nearest drillhole <=10m and # drillholes >2	Close proximity to drillhole with extrapolation minimal	Indicated
Domains 7 and 8	Indicated downgraded due to spottiness of possible indicated blocks and lack of QA/QC	Inferred
Below -375m	Indicated downgraded due to spottiness of possible indicated blocks when extrapolation is considered	Inferred

14.6.2 Bulk Density

A total of 87 specific gravity determinations were available for examination. All measurements were made from core drilled in 1996 and 1997. Unfortunately, the majority of these samples were from parts of the deposit not included in this resource estimate (higher up on the C Vein). A total of 17 samples were measured from C Vein Domain 6 and 1 from Y Vein Domain 7. These 18 samples had an average specific gravity of 2.81 and this value was used for all estimated blocks.

14.6.3 Mined Out Areas

All existing stopes and underground workings have been removed from the resource estimation by coding a Mined portion to each block. The resource is then restricted to the portion of the block remaining. It is noted that the majority of the previous mining is well above or to the northwest of the current resource, as illustrated in Figure 14-6. Only the upper portion of Vein 2 (CLOE) contains previous stoping.

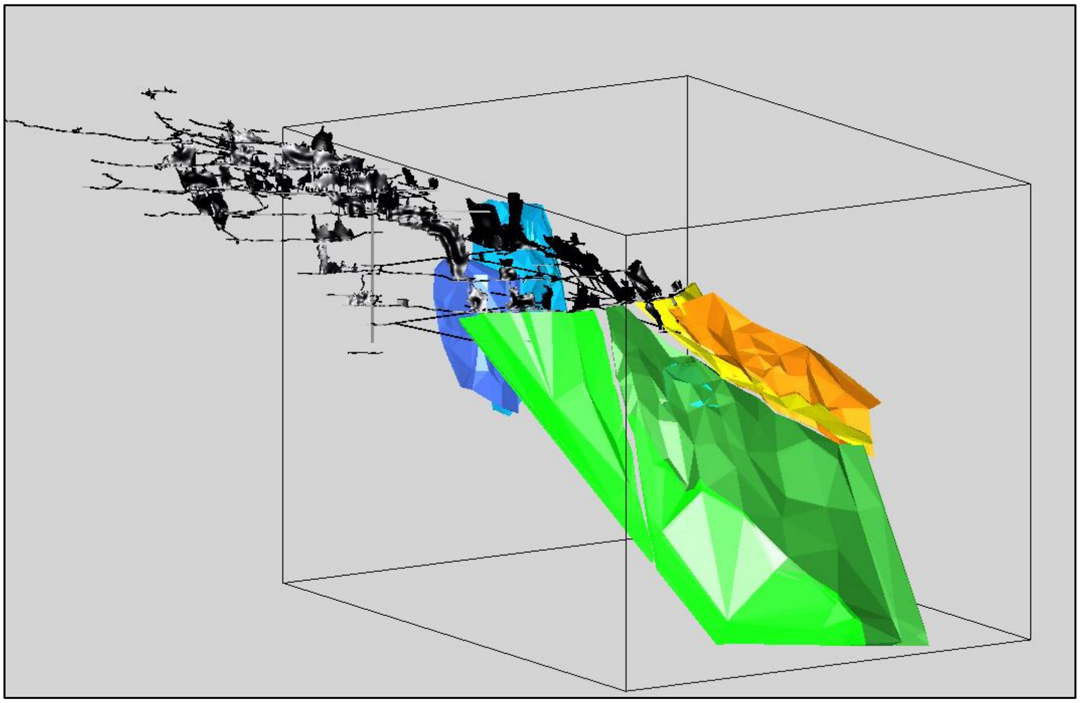

Figure 14-6 Modelled Veins and Underground Workings/Stopes (black), 3D Orthogonal View looking Northeast

14.7 Block Model Validation

Block model validation has been completed by a review and comparison of the mean grades in each domain with those of the de-clustered composite data (Nearest Neighbour interpolation). Further validation includes comparison of swath plots and visual comparisons of the modelled grades with the original assay data in section and in plan.

Table 14-11 summarizes the comparison of grades by Domain for the Indicated material.

Table 14-11 Summary of Model and De-Clustered Composite (NN) Mean Grades

Source	Au Grade Parameter	Estimation Domain 11	21	51	52	61	71	81	ALL
Model Values – Ordinary Kriging	Num Samples	3243	5649	1545	4105	946	844	291	16623
	Num Missing Samples	3	0	0	3	0	0	0	6
	Min	0.049	0.019	0.163	0.951	0.000	0.005	0.062	0.000
	Max	20.94	38.94	43.73	35.97	26.72	30.69	11.93	43.73
	Weighted mean	8.68	14.38	16.97	13.01	8.99	13.89	6.92	13.31
	Weighted CV	0.42	0.46	0.65	0.38	0.54	0.63	0.43	0.52
De-Clustered Composites	Num Samples	3243	5649	1545	4105	946	844	291	16623
	Num Missing Samples	3	0	0	3	0	0	0	6
	Min	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
	Max	27.60	55.93	60.00	53.20	45.05	31.99	16.73	60.00
	Weighted mean	9.15	14.64	20.47	12.68	9.97	13.99	8.17	13.79
	Weighted CV	0.68	0.79	0.93	0.81	1.13	0.86	0.85	0.87
Difference		**-5.5%**	**-1.8%**	**-20.7%**	**2.6%**	**-10.9%**	**-0.7%**	**-18.0%**	**-3.6%**

14.7.1 Swath Plots

Swath plots of the mean Au grade in the modelled veins have also been created to help validate the model. Figure 14-7 illustrated the swath plots for Indicated blocks in both the northing and easting directions. The swath plots compare the Au grade for the OK interpolations with those of the de-clustered NN interpolations. The NN grades are less smooth, but the plots show that there is no systematic bias in the OK interpolations.

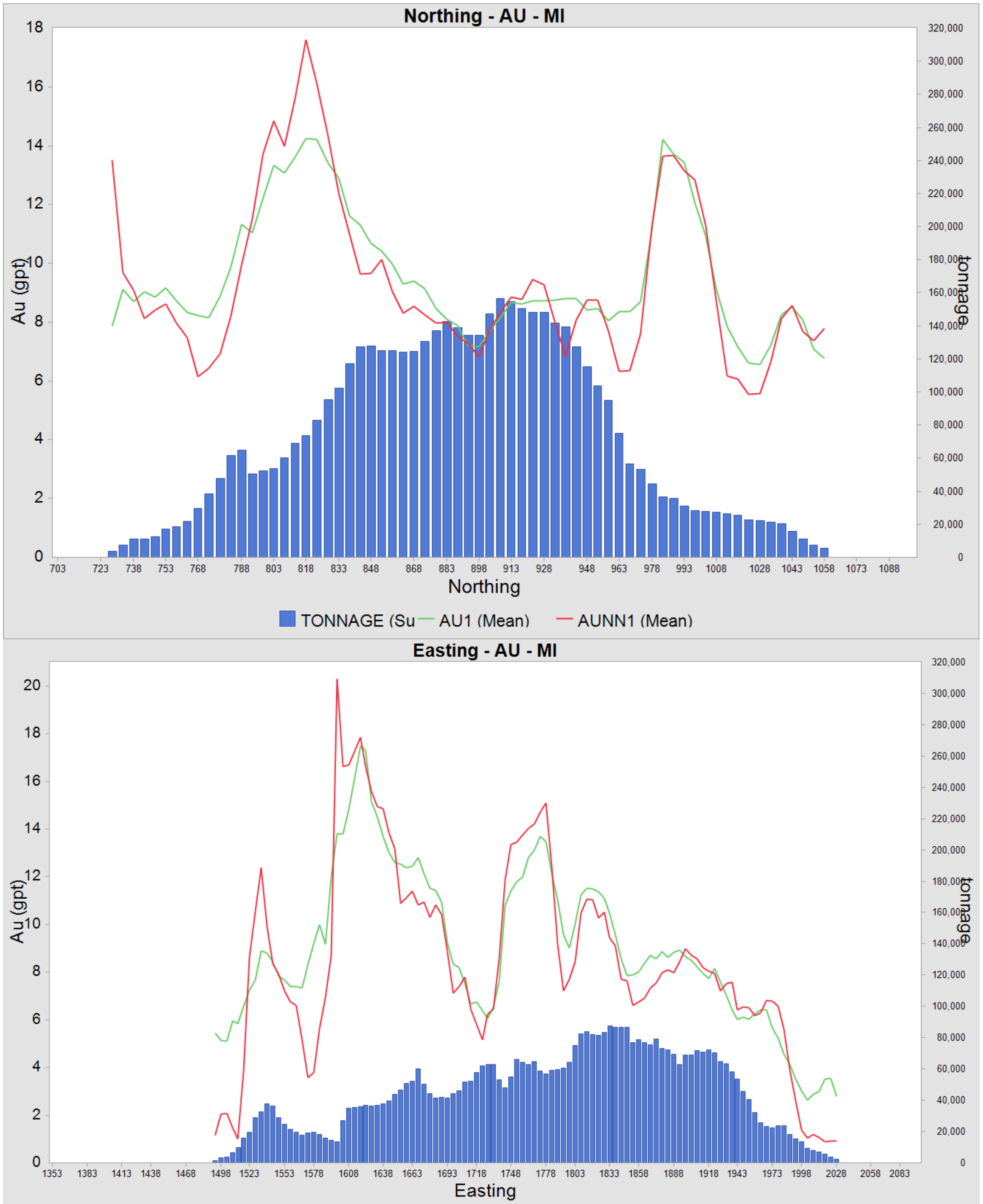

Figure 14-7 Swath Plots for Au

14.7.2 Visual Comparisons

The OK model Au grades have been compared to the assay grades in section and plan to ensure the model matches the data. Examples of the sections are given in Figure 14-8 through Figure 14-10. The assay data has been projected +/- 10 m from the section. In the section plots, the block size is scaled according to the percent of the block coded to be within the domain.

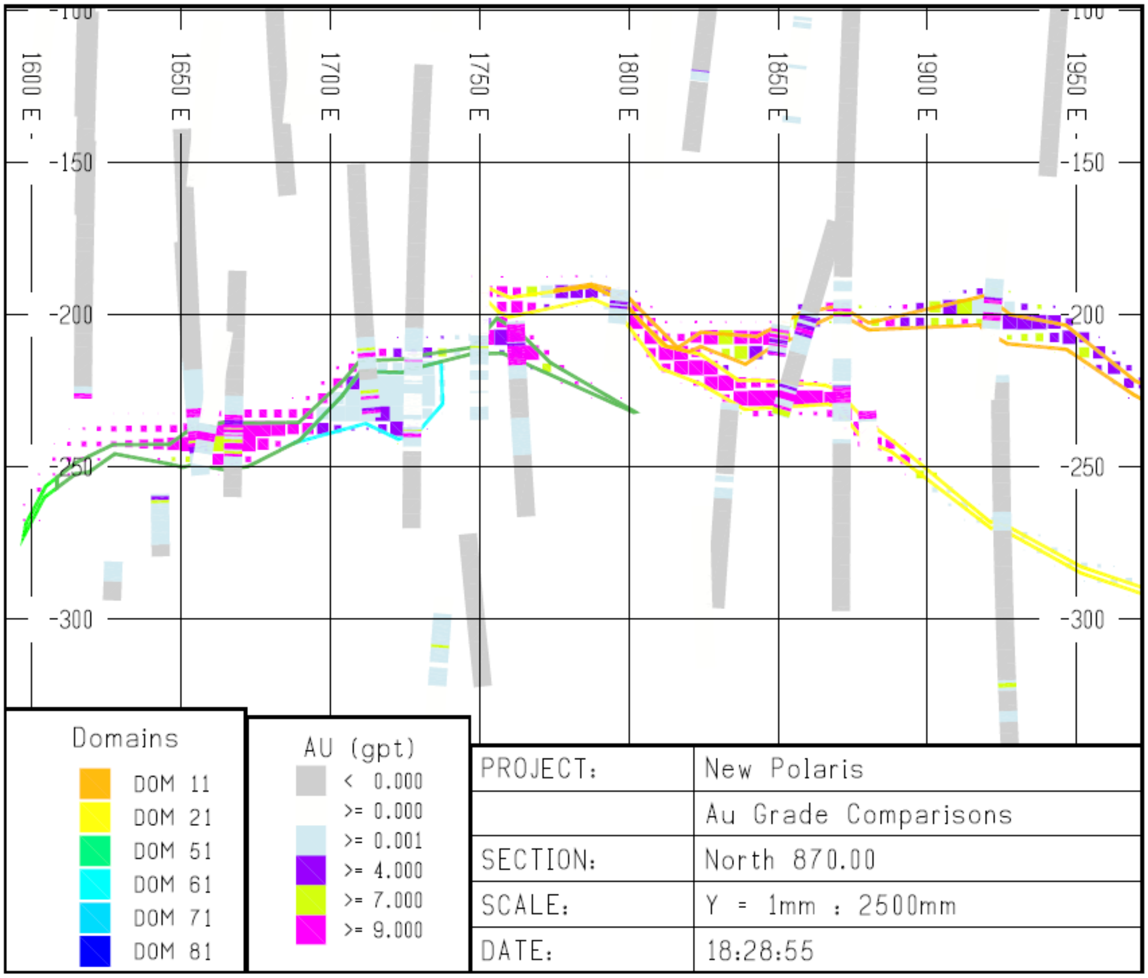

Figure 14-8 Comparison of Assay Au Grades and OK block grades, West East Cross-section, North 870 in mine specific space, looking north

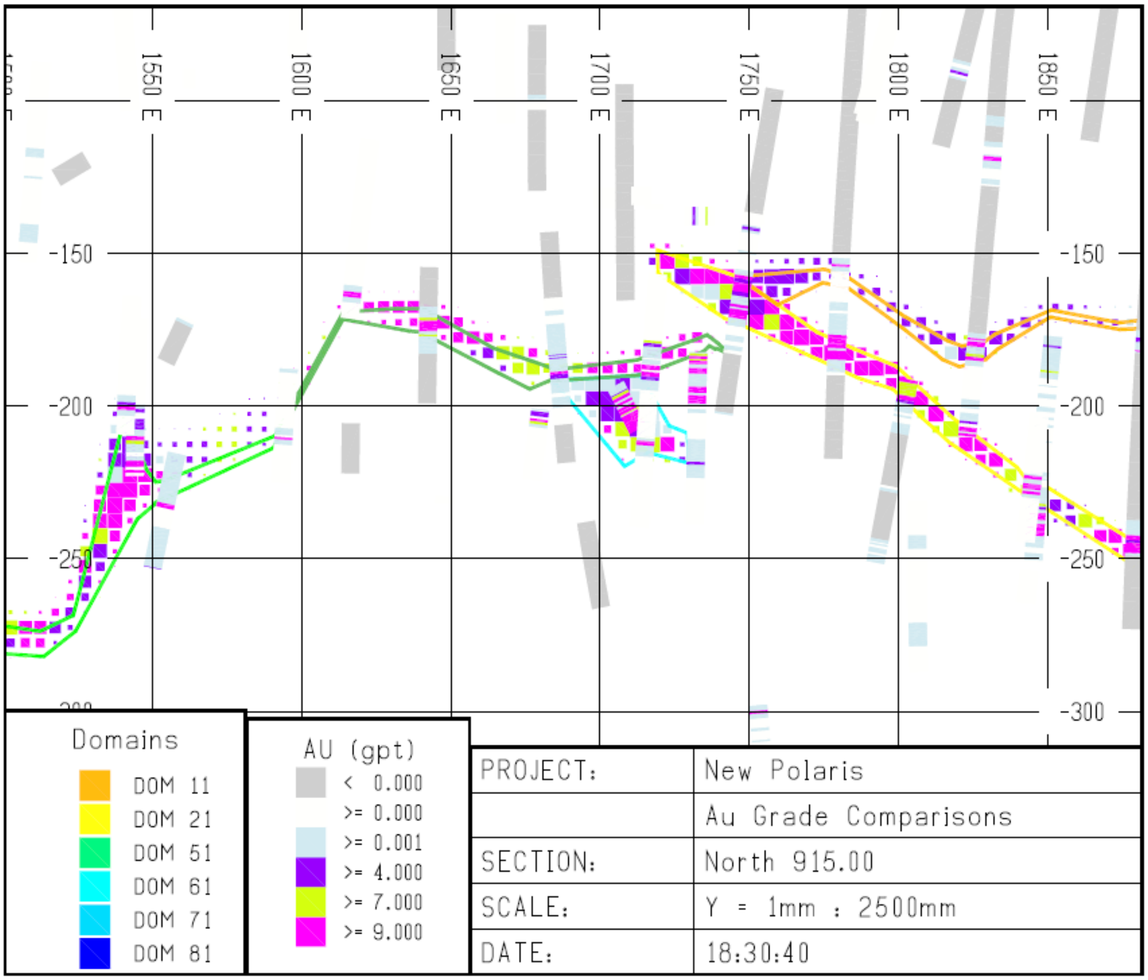

Figure 14-9 Comparison of Assay Au Grades and OK block grades, West East Cross-section, North 915 in mine specific space, looking north

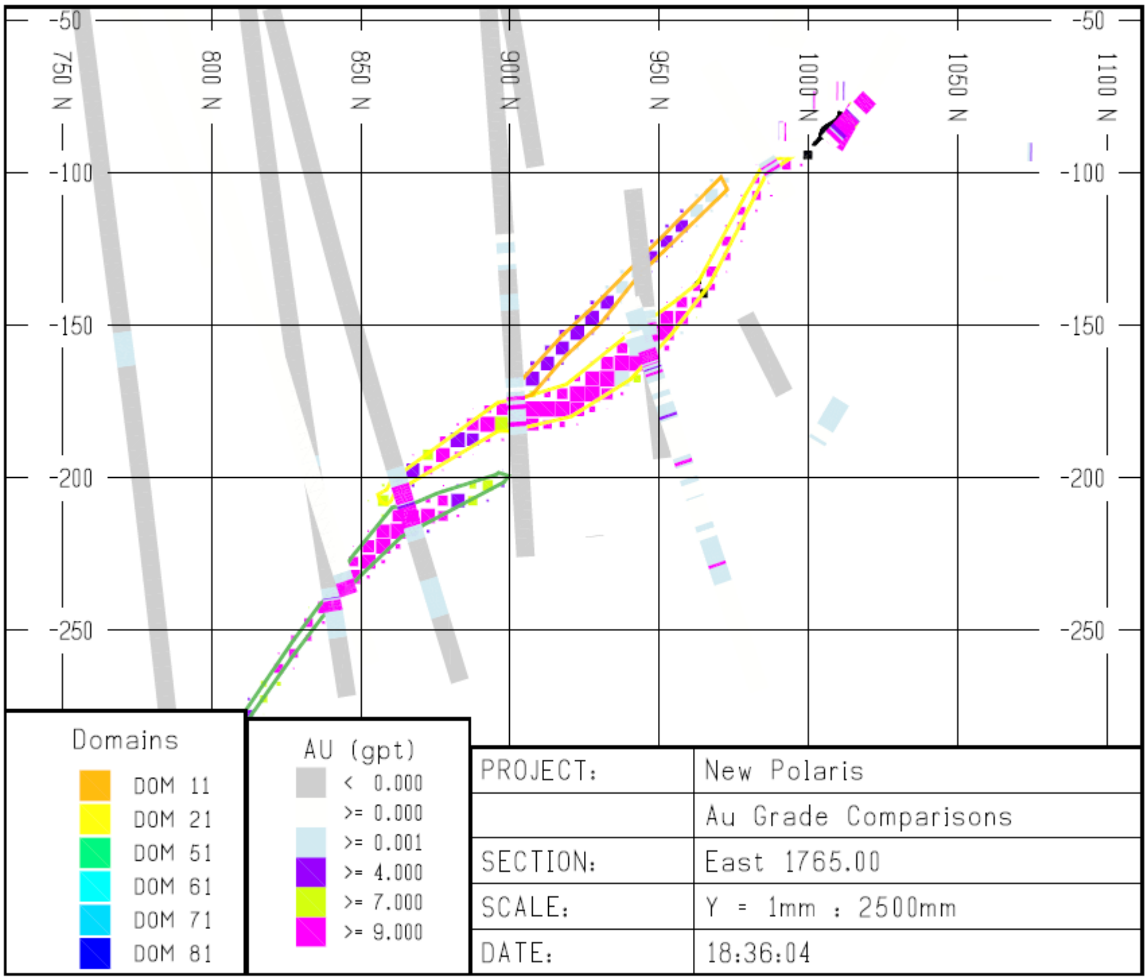

Figure 14-10 Comparison of Assay Au Grades and OK block grades, North South Cross-section, East 1765 in mine specific space, looking west

15 Mineral Reserve Estimates

This section is not relevant to the Technical Report

16 Mining Method

16.1 Introduction

The potentially mineable resources at the New Polaris deposit will be extracted using a combination of longhole stoping (LH) and conventional cut and fill (CCAF) underground mining methods, with paste backfill being used. The proposed mine plan will reach a target production rate of 750 tpd over a total mine life of 8.7 years. LH stoping will account for about 58% of total production, CCAF will account for about 18% and ore development will account for the remaining 24%.

The New Polaris deposit will be accessed from surface from a single decline, and all mineralized material and waste rock will be trucked out of the mine via this decline. This decline will also act as an exhaust airway along with the Polaris shaft; the new raise-bored ventilation raise will act as the fresh air intake.

A subset of both Indicated and Inferred Mineral Resources are included in the mine production plan. Inferred resources are considered too speculative geologically to have economic considerations applied to them to be categorized as mineral reserves, and there is no certainty that the Inferred resources will be upgraded to a higher resource category.

The confining stope shapes utilized in the mine plan targets Mineral Resources above a 6 g/t gold grade. Mineral Resources within the Y veins have not been included in the mine plan. Mineral resources above Level 1 (-140 m elevation) in the C veins have not been included in the mine plan.

16.2 Mine Planning Criteria

The mine planning criteria used are listed below:

- The pre-production period is approximately 14 months, with some development material processed in month 15 and during commissioning in the following months;
- The “ramp-up to full stope production takes approximately 24 months but development muck keeps the needs of the mill full as this ramp-up is achieved;
- Mining and maintenance are carried out by the owner;
- Major mobile mining equipment is assumed leased to own over a five year period;
- Mechanized mining equipment will be utilized for development and LH stoping;
- Conventional equipment (slushers, jacklegs, stopers) will be used for CCAF mining;
- Mined voids from both mining methods will be filled with paste backfill;

Other key mine planning criteria are summarized in Table 16-1.

Table 16-1 Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	365
Shifts per Day	Shifts	2
Hours per Shift	Hour	10
Work Rotation	4 weeks in/4 weeks out	
Nominal Ore Mining Rate	tpd	750
Annual Ore Mining Rate	tpa	273,750
Ore Density	t/m^3	2.80
Swell Factor		1.35
Target Cutoff Grade	g/t	6.0

16.3 Geotechnical Criteria

No geotechnical criteria were used in this report. Reasonable assumptions on ground conditions for the mining methods recommended were made.

16.4 Mine Production Plan

The PEA developed mine production plan and mill feed tonnes and grade are summarized in Table 16-2 below.

Table 16-2 New Polaris Mine Production and Development Schedule

	LOM	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Lateral Development (m)	**25,737**	3,996	5,400	5,400	3,939	3,600	3,402				
Raise Development (m)	**1,598**	284	269	172	339	330	204				
Development Ore (kt)	**554**	9	217	114	77	60	76	0	0	0	0
Stope Production, CaF & LH (kt)	**1,753**	0	62	149	190	206	199	270	275	273	129
Total Ore (kt)	**2,306**	8.9	279	263	268	266	275	270	275	273	129
Grade (g/t Au)	**10.32**	10.07	10.47	10.42	10.23	10.83	10.69	10.62	9.56	9.67	10.47
Mill Production (kt)	**2,306**	0	208	274	274	274	274	274	274	274	182
Grade (g/t Au)	**10.32**		10.55	10.42	10.24	10.84	10.68	10.63	9.56	9.68	10.34
Au Produced (oz)	**669**		62	80	79	83	82	82	74	74	53

16.5 Mining Methods

Two mining methods are proposed for the New Polaris deposit, sublevel longhole (LH) stoping and conventional cut and fill (CCAF). LH will be used in the steeper areas of the zone while CCAF will be utilized in the shallow dipping (less than 55°) and thinner areas. The CCAF stopes are generally located at the "on strike" extremities of the LH stoping areas.

Approximately 58% of total production will be from LH mining; CCAF will account for about 18% and ore development will account for the remaining 24%.

The mineralized zone has been divided into nine vertical levels 50 metres apart. On each level an undercut drift has been driven through the mineralized material and a footwall extraction drift accessing the undercut via drawpoints, has been driven in the footwall waste. Each level represents a single longitudinal LH stope, which is further divided vertically into three vertical stoping blocks by driving through the mineralized zone to its extremities. The CCAF stopes are only accessed by the undercut drifts but service and muck raises are driven from one extraction level up to the next.

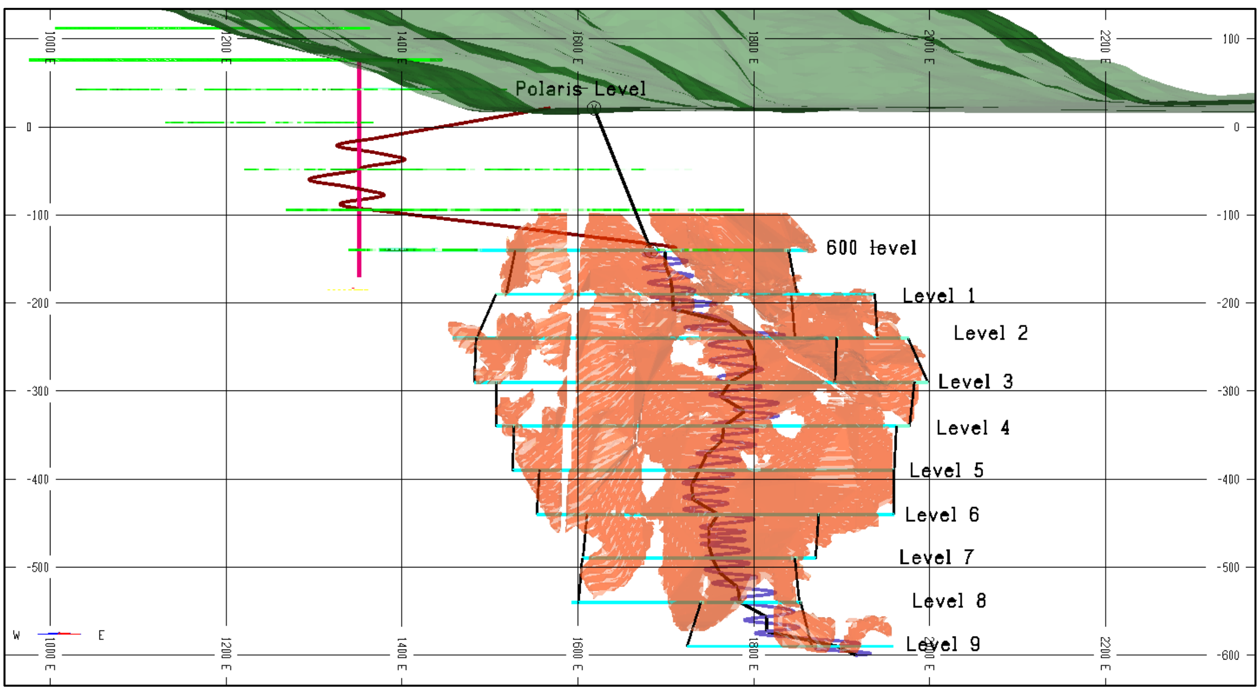

Figure 16-1 New Polaris Mine Plan Development and Orebody, Sectional View in mine specific space, looking North

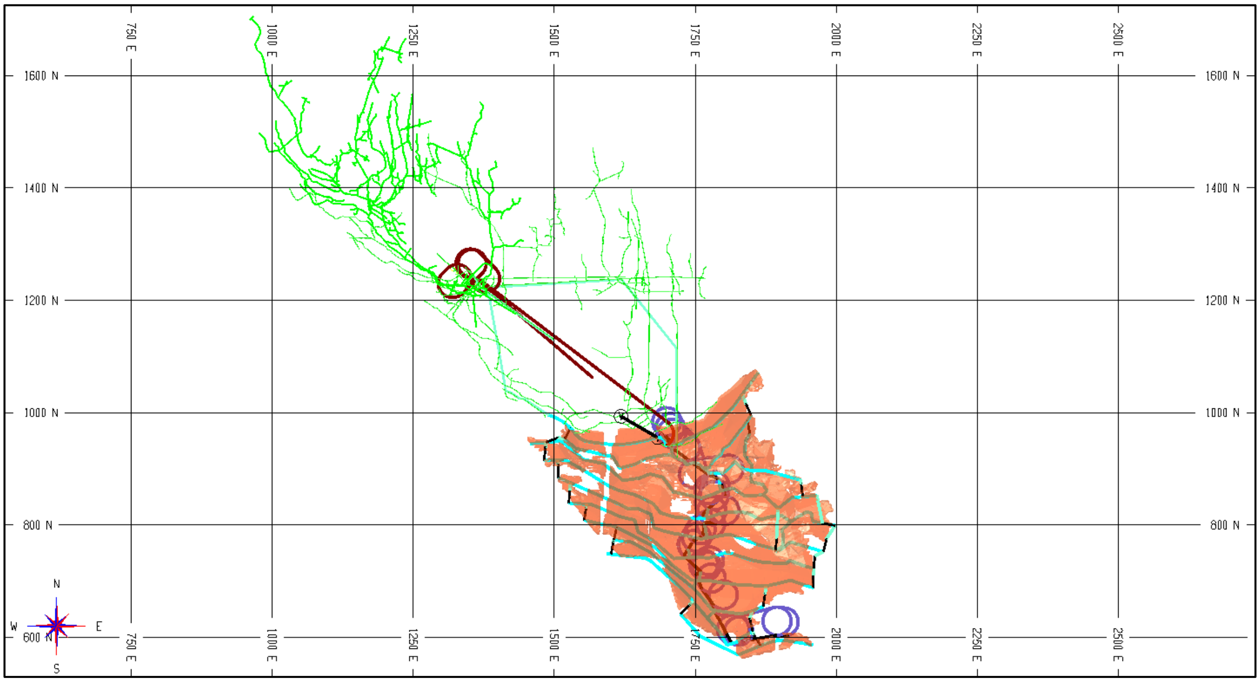

Figure 16-2 **New Polaris Mine Plan Development and Orebody, plan view in mine specific space**

The Figures above show the outlines of the 6 g/t target stopes, and the planned development to access these areas.

- The transparent orange solid represents the mining limits on the orebody.
- The transparent green surface is existing topography.
- The green lines represent the existing underground drifts down to the 600 level (-140 m elevation).
- The pink vertical line represents the existing shaft from the Polaris level down to the 600 level.
- The brown line represents the development path of a new ramp from the surface down to the 600 level.
- The blue spiral line represents the development of a new ramp from the 600 level to the bottom of the orebody, accessing sublevels and extraction drifts along the way.
- The cyan lines represent development of extraction drifts, starting on the 600 level, and progressing down from levels 1 through 9.
- The black semi-vertical lines represent the development of a new ventilation raise that ties into the ramp all the way from surface to the bottom of the orebody. This raise can also be used as a escape way during operations.
- Ventilation raises are also represented by black lines on the ends of each extraction drift, tying the levels together.

16.5.1 Sublevel Longhole Stoping

The LH stopes have been designed with a longitudinal orientation, thus, to maintain production two working areas on each level are required to be active at one time. One working area needs to be in the drill/blast/muck cycle while the other is in the backfilling cycle. Paste backfill will be used to fill mined stope panels and will be introduced through a network of boreholes and horizontal piping ultimately filling the stope panels from the extraction drift of the level above.

Longhole stoping provides high productivity at low mining costs from a small number of working faces. No geotechnical parameters have been used in the stope design thus to remain conservative, stopes panels have been designed to be 25 m along strike, 16.7 m high with the width being equivalent to the width of the mineralized zone and up to 8 m in some places. LH stopes will be 50 metres in height and comprise three sublevels at 16.7m vertical intervals. The main access ramp runs in the footwall of the zone and access each sublevel approximately in the longitudinal centre. Mining of each stope panel will be initiated by developing a slot raise at the far end by drilling and blasting with longhole techniques.

Stope extraction sequencing is planned to be from the longitudinal extremities of the zone retreating to the centre, where the access ramp is located. At the same time, mining will advance vertically up the mineralized zone. Three mining phases have been designed:

- from level 2 vertically to the 600 level,
- from level 5 vertically to level 2,
- and from level 9 vertically to level 5.

This will be carried out by establishing two horizontal sill pillars, which will ultimately be extracted. The first pillar will be located just below level 2 with the second just below level 5. This will allow initial mining to commence from level 2 vertically upward to the 600 level. While this is taking place, development of the zone at level 5 will be carried out so that mining of the stopes from level 5 up to level 2 to can take place. Lastly, while this mining is taking place, development of the stopes from level 9 up to level 5 will be taking place.

As the stopes are mined from the orebody extremities toward the access ramp, they are backfilled (all three sublevels) from the extraction drift above. Approximately 28 days is allowed for curing before the next stope, longitudinally adjacent to the previous one, can be started. The adjacent stope can be drilled while the previous stope is curing, although blasting is not recommended. When blasting does start, it is done by three slot (drop) raises located at the end of the stope almost on the backfill contact. Longhole drilling is carried out using up-holes from the extraction level and a combination of up and down holes from the upper sublevels. The up and down combination for the longhole drilling is implemented to reduce dilution from hole deviation.

Longhole rings will be blasted into the open stope and mineralized material will be mucked from the bottom of the stope by load-haul-dump (LHD) with remote control. This material will be trammed to the access ramp where low profile haul trucks will be loaded for transporting material up the ramp to a stockpile or direct dumping into the jaw crusher. Once a stope has been completely mined from the bottom sublevel through to the top one, backfilling can commence.

No rib pillars between stopes longitudinally have been planned as each successive adjacent stope will be mined up to the backfill contact.

An example of longitudinal Longhole Stoping is shown in the Figure below.

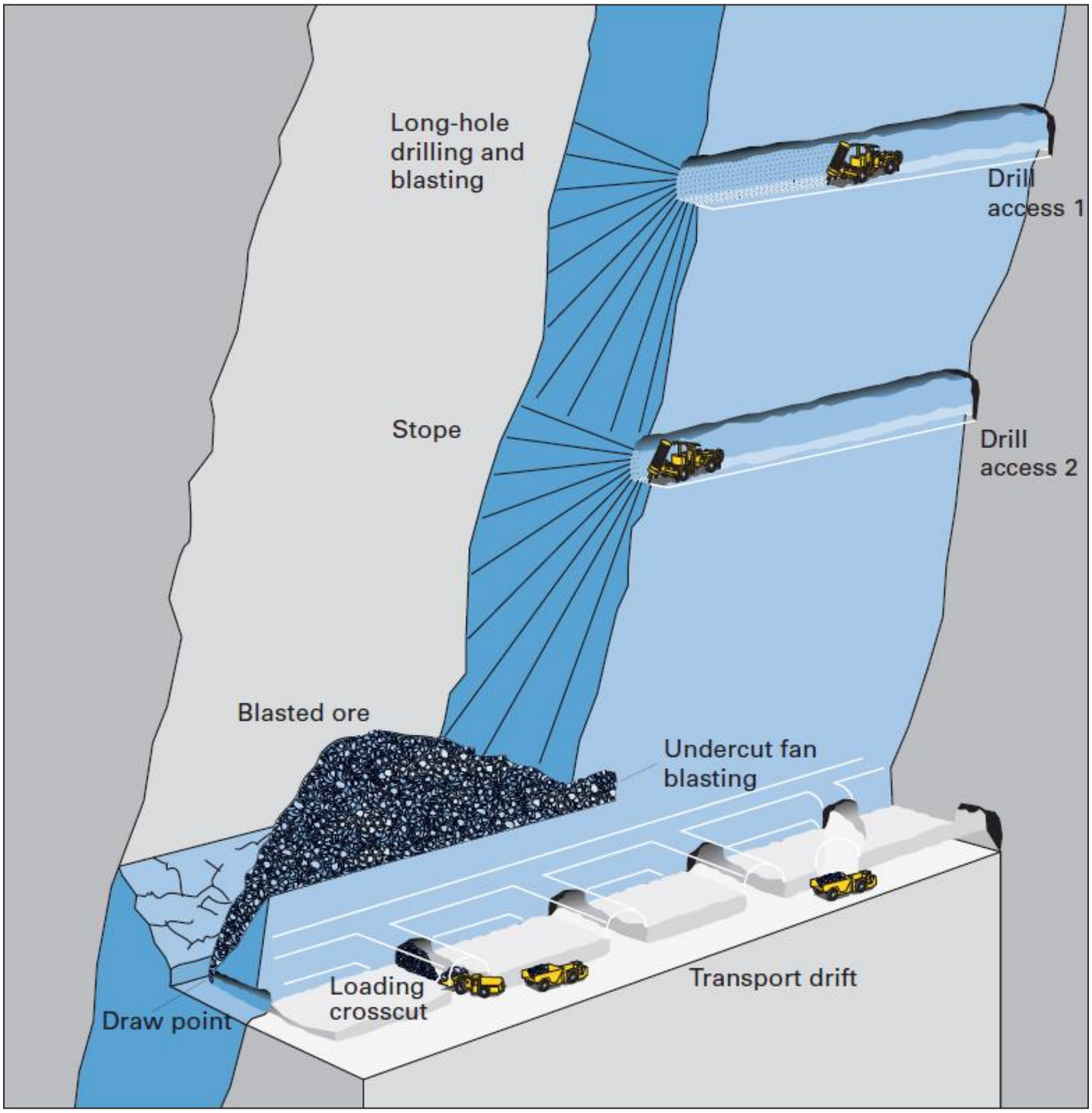

Figure 16-3 Example of longitudinal Longhole Stoping (from Atlas Copco, 2003)

16.5.2 Conventional Cut & Fill

For the thinner areas, generally at the longitudinal extremities of the zone, CCAF mining will be utilized. An example of cut and fill stoping is shown in Figure 16-4 below. Although more expensive than LH mining, it is more selective and produces less dilution. CCAF is chosen over mechanized cut-and-fill mining (MCF) even though it is less productive. This is because MCF would have required much more waste development to access each vertical cut of the mineralized zone. CCAF stopes are developed in 50 metre intervals from the extraction level of one stoping panel up to the next one. Stopes are up to 125 metres long and are accessed by a manway/muck raise carried from the extraction level up to the stope through the backfill. A ventilation/service raise is established in each stope up to the upper extraction level.

Mining is carried out in approximately 2.4 m high overhand cuts carried out using hand-held drill equipment comprising jacklegs, stoppers and slushers. Broken material is scraped to the muck raise using a slusher. Material is picked up in the stope undercut and trammed to the truck loading bay located off the access ramp. After each level has been cut, backfill is introduced into the stope via pipe localized in the ventilation raise. Crews are relocated to other stopes while the fill cycle is being carried out.

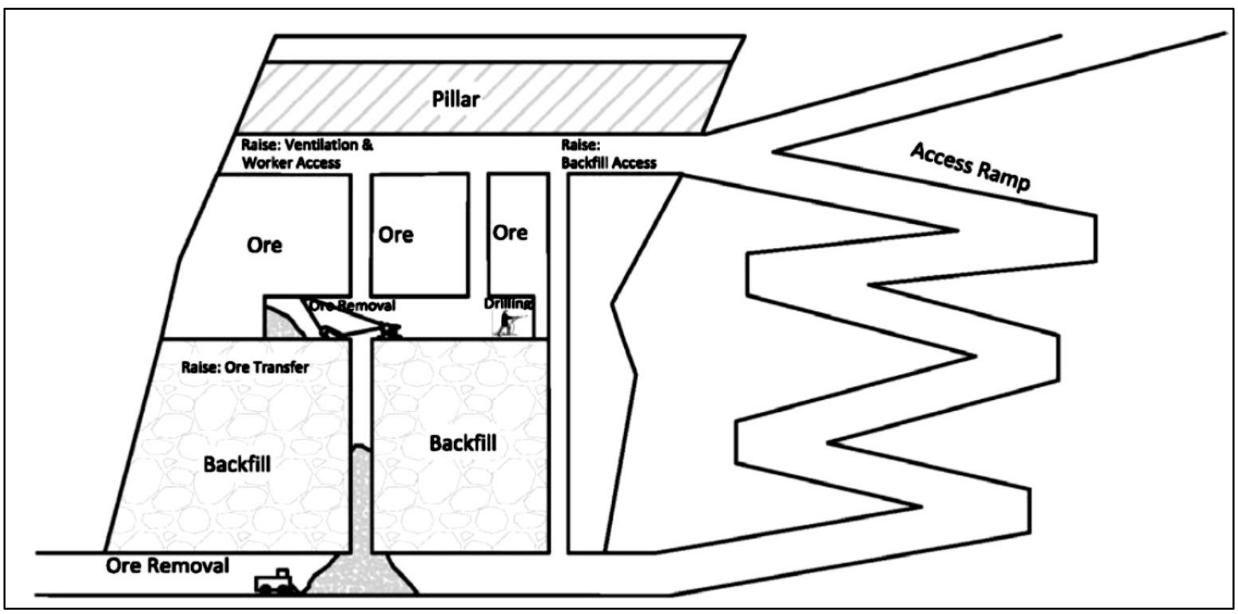

Figure 16-4 Example of Mechanized Cut & Fill (Atlas Copco, 2003)

16.6 Mine Design

The New Polaris deposit will be accessed via the New Polaris ramp collared at surface at the 24 m elevation. This ramp will be driven down to the existing 600 level at a grade of approximately -13% tying into the Polaris shaft at the existing 150 level and 300 levels for ventilation. Upon reaching the 600 level, this temporary ventilation system will be converted into a permanent system described in Section 16.7 Mine Ventilation.

Infrastructure on the 600 level will consist of a fresh air raise to surface, the primary sumps, explosives magazines, a fuel bay and a refuge station. A hanging wall drift for diamond drilling will also be excavated. Lastly, the existing footwall waste drift will be slashed/extended to cover the length of the zone and equipped with the paste backfill distribution system.

From the 600 level, a ramp will be driven in the footwall waste at approximately 14% to 9 level. Along with the ramp, a raise will be driven in segments, level to level, to provide fresh air. The ramp will serve as an exhaust air route.

Just below levels 2, 5 and 9, secondary sumps will be established which will pump mine water to the 600 level sumps. A second refuge station will be established near the sumps below level 6.

The ramp driven down the footwall of the mineralized zone will access each sublevel of the LH stopes as well as each extraction level. The accesses to these sublevels will be at 16.7 m vertical intervals. Each access will have a truck loading bay for either waste or mineralized material as well as a small sump to collect water from the level. This water will be drained to the closest secondary sump on the ramp.

16.7 Mine Ventilation

The permanent ventilation system will utilize the following:

- a new fresh air raise (downcast) driven from 600 level to surface
- the existing workings on 600 level
- the new ramp from surface as an exhaust way
- and the Polaris shaft as an exhaust way

Upon reaching the 600 level with the ramp, a new fresh air raise to surface will be driven with a raiseborer. It will have a fan/heater located at the collar and will provide heated fresh air to 600 level. On 600 level, the existing workings will be slashed to complete the ventilation circuit to both the new ramp and the Polaris shaft, both of which will now act as return airways.

From the 600 level to 9 level, fresh air will be provided through the raise driven adjacent to the ramp that accesses these levels. Fresh air will then flow through each access from the ramp to the extraction drift and then in both directions, east and west through the extraction drifts to the return air raises located at the extremities. These return air raises bring the exhaust air to the 600 level and then through the slashed historic workings to the Polaris shaft and the new ramp.

The design basis of the ventilation system at the New Polaris project is to meet the requirements of the BC Health, Safety and Safety Regulation Code for Mine in British Columbia.

The Code calls for 0.06 m^3/s of ventilating air for every kilowatt power of diesel equipment operating. Further to these needs, an additional factor of 15% has been added to account for leakage. The horsepower rating of each piece of underground equipment has been collected, and then utilization factors representing the diesel equipment in use at any time were applied to estimate the amount of air required.

Table 16-3 lists the air requirements for full production with the total of 78 m^3/s (165,500 CFM – cubic feet per minute) air volume required.

Table 16-3 Diesel Equipment Ventilation Requirements

Equipment Type	Units	HP per Unit	HP (Total)	Availability (%)	Utilization (%)	HP (Utilized)	Air Volume (m^3/sec)	Air Volume (ft^3/min)
Jumbo 2 Boom	2	147	294	85	25	62	2.8	5,926
LH Drill	1	99	99	85	25	21	0.9	1,996
Mechanized Bolter	1	99	99	85	25	21	0.9	1,996
20 t Truck	3	300	900	85	90	689	30.8	65,308
3.0 m^3 LHD	3	201	603	85	70	359	16.1	34,033
2.0 m^3 LHD	1	70	70	85	50	30	1.3	2,822
Scissor Lift	1	147	147	90	30	40	1.8	3,765
ANFO Loader	1	147	147	90	30	40	1.8	3,765
Fuel/Lube Truck	1	147	147	90	50	66	3.0	6,275
Utility Truck	1	147	147	90	40	53	2.4	5,020
SV/Mechanic Vehicles	4	127	508	90	30	137	6.1	13,010
Subtotal:							**67.9**	**143,913**
Losses at 15%:							10.2	21,587
Total:							**78.1**	**165,500**

16.7.1 Emergency Warning System

A stench gas system utilizing ethyl mercaptan will be installed at the collar of the new fresh air raise as well as on the compressed air system in the new ramp and may be triggered as appropriate to alert underground personnel in the event of an emergency. Airflow velocities will permit all personnel to be alerted of the emergency within an acceptable period. Once employees underground smell the warning gas, they will immediately take refuge in appropriately outfitted lunchroom/refuge stations located on 600 level and just below 6 level. If required and after confirmation that all personnel are secured in refuge stations, ventilation flow can be adjusted via VFD or remote access.

16.8 Underground Mine Services

16.8.1 Mine Power

The main consumers of underground power will be the following:

- Mine ventilation (primary and secondary fans);
- Underground pump;
- Electric-hydraulic jumbos and electric-hydraulic bolting jumbo;
- Air (compressed air located near the New Polaris ramp portal);
- Mine lighting; and
- Refuge stations.

The average underground power load is expected to be – 1.4 MW; and average annual consumption is expected to be 6.2 MW.

16.8.2 Communications

A leaky feeder communication system will connect the mine with surface operations. Telephones will be located at key infrastructure locations such as the ramp portal, the top of the fresh air raise, each of the refuge stations and at the ramp at the access to each extraction level.

16.8.3 Compressed Air

Compressed air will be used for stoper, jackleg drilling, secondary pumping, ANFO loading, and blasthole cleaning. The mine will have a dedicated compressed air system located near the truck shop on surface. Compressed air will be delivered underground in a 150 mm diameter pipe via the new ramp and 100 mm pipes in the extraction levels and sublevels.

The underground mobile drilling equipment such as jumbos, bolting jumbos and the production drill will be equipped with their own compressors.

16.8.4 Mine Water Supply

Mine supply water from the process freshwater tank will be distributed to the underground levels via 100 mm diameter pipelines in the new ramp. Further distribution to work headings will be via 50 mm diameter water lines. Pressure reducers will be located along the length of the line.

16.8.5 Mine Dewatering

No calculations have been carried out on mine dewatering but it is assumed that the mine will be wet and that pumping will be required.

A primary sump will be established on the 600-level close to the top of the ramp extending down the footwall of the mineralized zone. Three secondary (smaller) sumps will be established just off the ramp below level 2, level 5 and level 9. They will pump water from one sump up to another or directly to the 600 level sumps depending upon water flow.

16.8.6 Explosives and Detonator Storage

Explosives and detonators will be stored on surface in approved magazines. From there, they will be transported underground to one of two sets of magazines. One set of magazines will be located on the 600 level and the second will be located on level 5. Day boxes near active advancing faces will be used as temporary storage for daily explosive consumption.

Emulsion will be used as the major explosive for mine development and production mining. All personnel underground will be required to be in a designated Safe Work Area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes at the end of the shift.

16.8.7 Fuel Storage & Distribution

A mobile equipment station for fueling and lubrication will be located on the 600 level to provide fuel for the underground mobile equipment fleet. Additionally, there will be a fuel truck and a lube service truck to service the less mobile mining equipment. During development, fuel will be transported into the mine by a fuel truck; before production commences, a fuel station will be constructed on 600 level and will be supplied with fuel by a fuel line through a borehole from surface. From this fuel bay, mine equipment may load fuel directly or the fuel truck will load and transport fuel to them.

16.8.8 Central Blasting

Central blasting will be used at the New Polaris mine, which will allow the operation to initiate blasts remotely from a safe control point on the surface. Digital central blast systems have been sourced from the major suppliers of explosives. These systems are extremely safe and contain redundancy coding that prevents accidental initiations. These systems will work through the leaky feeder mine communications system.

16.8.9 Mobile Equipment Maintenance

Mobile underground equipment will be maintained in the surface maintenance shop located near the portal of the New Polaris ramp. A mechanic's truck will be used to perform emergency repairs underground. Maintenance directions will be provided by the mine supervisor. A maintenance planner will ensure the availability of spare parts and supplies and provide management and supervision to maintenance crews.

16.9 Mine Safety

Self-contained portable refuge stations will be provided in the main underground work areas. The refuge chambers are equipped with compressed air, potable water, and first aid equipment; they will also be supplied with a fixed telephone line and emergency lighting. The refuge chambers will be capable of being sealed to prevent the entry of gases. The refuge stations will be moved during development. Once the 600 level has been completed, one refuge station will remain there near the top of the ramp; the other will eventually be located just below level 6.

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the underground electrical installations, pump stations, fuelling stations, and other strategic areas.

Every vehicle will carry at least one fire extinguisher of adequate size. It is recommended that underground heavy equipment be equipped with automatic fire suppression systems.

In the event of a fire, if appropriate, personnel will be directed to the refuge stations or depending upon location of the fire, personnel will be directed to surface via the designated escape route. The fresh air raise will be equipped with a ladder and will act as an escape way although depending upon location of the fire, airflow may be reversed, and personnel may walk up the ramp in fresh air.

16.10 Paste Backfill

It is important for the New Polaris project to limit the environmental impact by using as much of the plant tailings as paste fill underground. Some tailing from the BIOX and CIL circuits will also be used for paste backfill. A total of 42% of all tailings will be used for paste backfill.

Paste backfill will be sourced either directly from the paste fill plant located near the mill or tailings will be reclaimed from the tailing impoundment facility and run through the plant and then sent underground. Paste fill will be pumped through one of two boreholes located near the new fresh air raise and will be collected underground on the 600 level from where it will be distributed with a pipe network and additional boreholes from the footwall extraction drifts on each level.

16.11 Mine Equipment

The selection of underground mining equipment is based on mine plan requirements, mining methods, operating ramp, drift and stope dimensions. With just under a nine-year mine life, it is assumed that the equipment will go through one life cycle and will not require any rebuilds throughout.

Two 2-boom electric-hydraulic jumbos will be used for ramp, lateral development in waste and ore development in stopes.

Production drilling will be carried out with a single electric-hydraulic longhole drill capable of drilling both up and down holes up to a 4” diameter.

Production mucking will be carried out with 3.5 m^3 load-haul-dump (LHD) units equipped with remote-control operating capabilities. These units will be used for mucking both waste and ore during development and from stopes (remote-control) during operations. All material will be transported to surface in 28 tonne low-profile haul trucks.

All major pieces of equipment are described in Section 21.1.3.

16.12 Mine Personnel

Mine department personnel is divided into three categories:

- Technical Services
- Mine and Maintenance Supervision
- Mine operations (operators and maintenance support)

The mining department personnel at full-scale production of 107 is shown in the Table 16-4 below.

Table 16-4 Mine Personnel

Position	Number
Technical Services	
Senior Mine Engineer	2
Mine Engineer	2
Surveyors/Technicians	4
Chief Geologist	1
Senior Geologist	1
Beat Geologists	2
Samplers	2
Subtotal:	**14**
Mine and Maintenance S/V	
Manager of Mining	2
Mine Supervisors	4
Safety/Training Foreman	2
Mine/Maintenance Clerks	2
Subtotal:	**10**
Mine Operations and Maintenance	
Development Miners	14
LH Drillers	2
LH Blasters	2
LH Muckers	2
Truck Drivers	14
Paste Backfill Crew	4
Diamond Drillers	4
Service Crew	4
Equipment Operators	4
Paste Backfill Plant Crew	4
Fuel/Lube/Truck Operators	2
Mine Labourers	4
Remuck Scoop Operators	2
Mechanics/Welders	15
Electricians	2
Shop/Electrical Helper	4
Subtotal:	**83**
Total:	**107**

Technical Services

The technical services department comprises mining engineers, geologists, surveyors, technicians and samplers. This staff is deemed sufficient to handle all technical aspects required for operation of the mine as well as carrying out exploration.

Mine and Maintenance Supervision

Mine and maintenance supervision will be required to supervise the mining and maintenance work forces to ensure that tasks are being carried out efficiently and safety. Mine trainers are included in this group as are maintenance planners.

Mine Operations and Maintenance

Mine operations and maintenance include general mine services, direct mining activities and maintenance support as follows:

- General mine services includes the underground pastefill crew and plant operating crew, diamond drilling, delivery of fuel and lubricants, delivery of mine consumables (explosives, ground support material, piping, etc.), truck and LHD operators for remucking waste and mineralized material, and road maintenance.
- Direct mine activities include driving ramps, drifts, sublevels and raises as well as all activities associated with longhole mining (drilling, blasting, mucking).
- Maintenance support includes mechanics, welders, electricians required to keep mobile equipment and stationary equipment operating.

Technical staff and management will work 10-hour day shifts over a four-week period then take a four-week break; they will be replaced by an alternating crew. Underground (direct operating labour, general mine services labour and maintenance labour) will work two 10-hour shifts per day.

With a four-week on/ four-week off work schedule, four mining, four maintenance, and two technical service crews will be required.

17 Recovery Methods

17.1 Process Flowsheet

The New Polaris process includes crushing, grinding, sulphide flotation, Bio-Oxidation (BIOX™), Carbon-in-Leach (CIL), carbon desorption, Electrowinning and smelting to produce a gold doré.

The preliminary process flowsheet for New Polaris is shown in Figure 17-1.

17.2 Process Design Criteria

The New Polaris plant is designed to process 750 tonnes of mill feed per day (tpd). The crushing stage availability is assumed to be 75% and the grinding and flotation plant availability is assumed to be 80%.

Preliminary BIOX™ plant design by Outotec is designed to treat 105 tpd of flotation concentrate with an availability of 95%.

Process design criteria for New Polaris are summarized in Table 17-1.

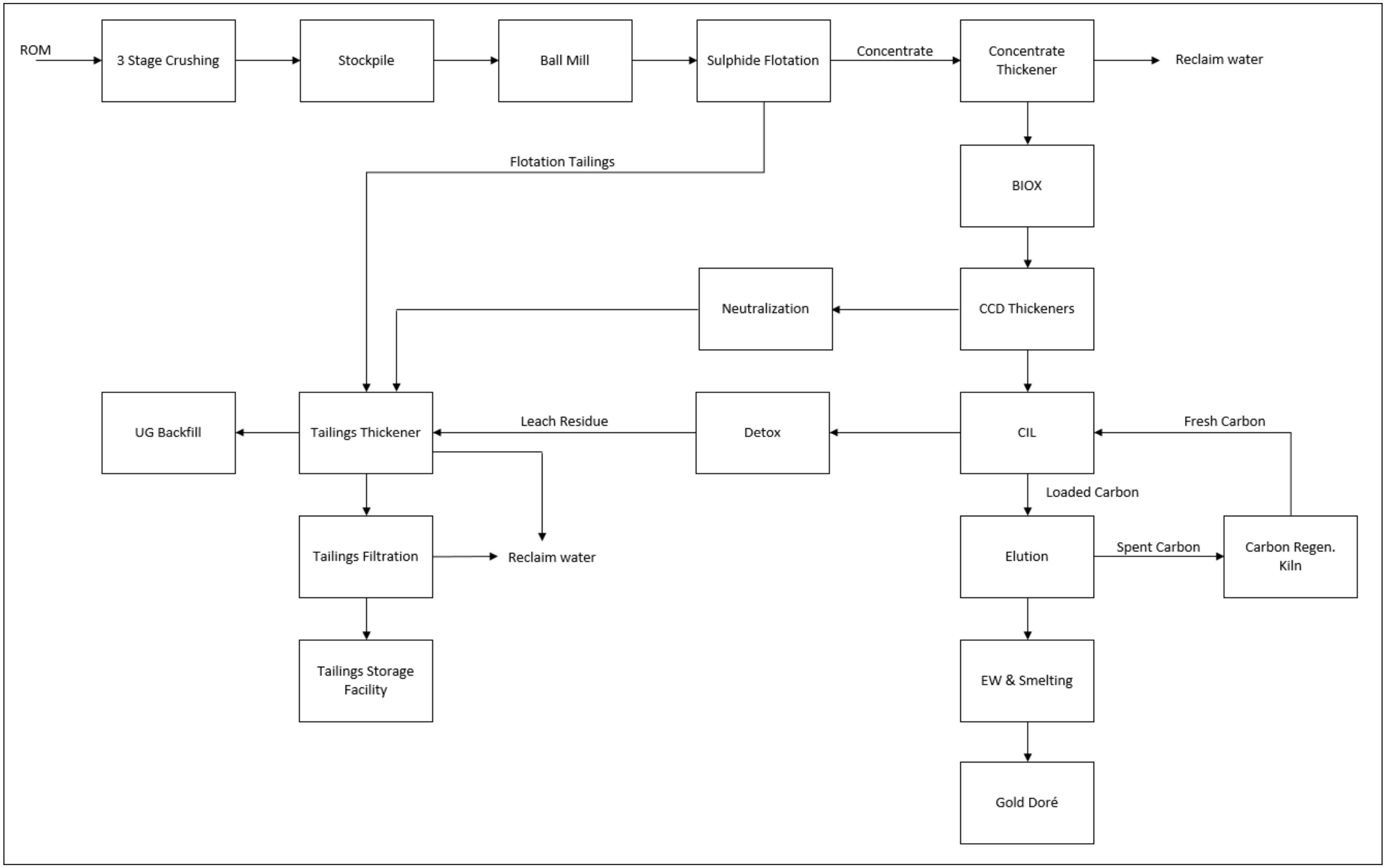

Figure 17-1 New Polaris Process Flowsheet

Table 17-1 Summary of New Polaris Preliminary Process Design Criteria

Description	Unit	Value
Annual Mill Feed Throughput	tpa	273,750
Operations		
Crusher Availability	%	75
Grinding and Flotation Availability	%	80
Plant Daily Throughput	tpd	750
Plant Hourly Capacity	tph	39
Average ROM Feed Au Grade	g/t	10.3
Crushing		
Primary Crusher	type	Jaw
Secondary Crusher	type	Cone
Tertiary Crusher	type	Cone
Fine Ore Stockpile Live Capacity	t	750
Grinding		
Bond Work Index	kWh/t	19.6
Grinding Feed Particle Size F_{80}	mm	12
Grinding Product Particle Size P_{80}	µm	75
Mill Type		Ball Mill
Number of Mills		1
Rougher/Scavenger Flotation		
Residence Time	mins	37.5
Number of Cells		6
Cell Type		Tank
Cell Volume	m^3	12
Rougher/Scavenger Mass Pull	%	26.5
Cleaner Flotation		
Residence Time	mins	27.5
Number of Cells		2
Cell Type		Tank
Cell Volume	m^3	7.6
Final Concentrate Mass Pull	% of mill feed	15.2
BIOX™ Leaching		
Plant Capacity	tpd	105
Residence Time	days	6
Number of Tanks		6
CIL and Carbon Desorption		
CIL Residence Time	hours	24
Number of CIL tanks		6

Description	Unit	Value
Carbon Concentration	g/L	20
Estimated Carbon Loading	g/t	950
Elution Strip Rate	strips per week	7
Cyanide Destruction		
Method		ASTER™ and SO_2/Air
Reagents Used		Molasses, $Na_2S_2O_5$, $NH_4H_2PO_4$
Tailings Thickener		
Target U/F Density	%	60
Number of Thickeners		1
Tailings Filtration		
Filter Type		Ceramic Vacuum Disc
Design Rate	t/h/m^2	0.5
Target Moisture Content	%	16
Availability	%	85
Number of Filters		2

17.3 Process Description

17.3.1 Crushing

Crushing is designed in a 3-stage circuit including a jaw primary crusher, followed by secondary and tertiary cone crushing. Crushed ore with a P_{80} of 12 mm is then conveyed to into a 750 t live capacity stockpile. Ore from the stockpile is reclaimed from a tunnel beneath the stockpile using vibratory feeders.

17.3.2 Grinding

The grinding circuit uses a 1000 kW ball mill in closed circuit with a cyclone to reduce particle size from an F_{80} of 12 mm to a P_{80} of 75 µm. The grinding circuit processes 750 tpd with an availability of 80% and a throughput of 39 tph.

17.3.3 Flotation

Flotation is carried out in 6 Rougher and Scavenger cells, and 2 cleaner cells. Air is sparged into each cell. Flotation reagents and dosages are listed below.

Rougher stage

- Conditioning (17.5 mins): 250 g/t Na_2S, 50 g/t Potassium Amyl Xanthate (PAX)
- Float (20 mins): 13 g/t MIBC frother

Scavenger stage

- Conditioning (5 mins): 25 g/t PAX
- Float #1 (15 mins): 4 g/t MIBC
- Conditioning (12.5 mins): 250 g/t $CuSO_4$ (conditioning), 25 g/t PAX
- Float #2 (12.5 mins): 9 g/t MIBC

Cleaner stage

- Conditioning (2.5 mins): 5 g/t PAX
- Float (25 mins): 6 g/t MIBC, 50 g/t Na_2SiO_3

Flotation concentrate is pumped to a concentrate thickener. Flotation tails are pumped to the tailings thickener.

17.3.4 BIOX Leaching

Flotation concentrate is pumped from the concentrate thickener to three storage tanks at 50% solids. Concentrate is then pumped into three primary oxidation reactors in parallel. Bio-oxidation occurs after conditioning and inoculation with bacterial culture. Residence time in the primary reactors is 3 days. Partially oxidized concentrate will then be pumped to three secondary reactors in series. Residence time for each reactor is 1 day.

Concentrate is oxidized for a total of 6 days. Oxidized concentrate is then pumped to Counter Current Decantation (CCD) thickening circuit to prepare for cyanide leaching.

CCD thickening washes oxidized concentrate and removes acid generated during sulphide oxidation, as well as dilutes aqueous elements such as ferric iron or arsenic that are released during oxidation. The CCD process will use thickeners with a wash ratio of 4.2. Final CCD underflow is pumped at 35% solids to the CIL stage.

17.3.5 Neutralization of CCD overflow

Neutralization of the CCD overflow liquor takes place with addition of lime slurry or limestone slurry, depending on the pH of the liquor. Lime reacts with the sulphuric acid to produce gypsum ($CaSO_4*2H_2O$) which is a stable solid by-product. This removes sulphuric acid from the liquor and increases the overall pH of the resulting slurry. Neutralized slurry is pumped to the tailings thickener.

17.3.6 CIL

CCD underflow is pumped to a pH conditioning tank, where lime is added to increase the pH of the slurry to 11. Sodium cyanide is added after pH conditioning. The slurry is then pumped into a series of 6 CIL tanks and flows countercurrent to 20 g/L activated carbon. Total residence time for the slurry in the CIL stage is 24-hours. Barren activated carbon enters the circuit in Tank #6 and is advanced countercurrent to the slurry flow. Screens are used to filter the carbon and carbon pumps are used to move carbon between tanks. Loaded carbon removed from Tank #1 is moved to the carbon desorption. Leach tails are pumped from Tank #6 to the detoxification stage.

17.3.7 Carbon Desorption and Regeneration

Carbon from the CIL loaded carbon screen is pumped to acid wash. Acid wash is carried out with diluted hydrochloric acid in an acid wash column inside an acid-proofed concrete sump to ensure that all spillage is captured and kept separate from other process streams.

After acid wash, the carbon is pumped to an elution circuit that includes an elution column, strip solution tank, strip solution pump, and a strip solution heat exchanger. The elution circuit operates in closed circuit with electro-winning cells.

Strip solution heat exchangers maintains the strip solution temperature at 145 °C during the stripping cycle and ensures that the temperature of loaded solution entering the electro-winning cells is below 100 °C.

Eluate flows directly from the top of the elution column to a loaded solution tank after cooling through heat exchangers. The eluate is pumped from the loaded solution tank to electro-winning cells to recover gold onto stainless steel cathodes and as sludge. Barren solution from electro-winning gravitates back to the strip solution tank. The gold is removed from the cathodes and the sludge is collected from the Electrowinning cells and vacuum filtered before refining.

The resulting gold sludge will be vacuum filtered, washed, and dried in a furnace before smelting. Barren solution will be recycled back into the carbon desorption stage.

17.3.8 Refining

Filtered gold product from the Electrowinning stage is directly smelted with fluxes in an induction furnace. Gold doré will be poured into molds, weighed, stamped and stored in a safe. Exhaust from the furnace is passed through a wet scrubber to remove entrained particles and dust, and then vented out of the plant via a chimney stack.

17.3.9 CIL Tailings Detox

Detoxification of the CIL tails includes a combination of the Outotec ASTER™ process and SO_2/air process. The ASTER™ process uses biological degradation of thiocyanate and cyanide and is particularly designed for subsequent upstream re-usage of treated solution in the Outotec BIOX process. The SO_2/air process using sodium metabisulphite ($Na_2S_2O_5$) is included as a contingency to ensure targeted cyanide levels in leach residue are achieved.

17.3.10 Tailings Thickening and Filtration

Detoxified CIL residue, flotation tails and neutralized CCD overflow are pumped to the tailings thickener. Thickener underflow density is 60% solids. Thickener overflow is pumped to the process water tank. Approximately 42% of the thickened tails are sent to an underground backfill operation. The remaining thickener underflow is filtered using ceramic vacuum disc filters with a target moisture content of 16%.

Filtered tails are conveyed onto a stockpile, loaded onto haul trucks, and transported 2 km to the co-disposal facility (CDF).

17.4 Reagents and Power Consumption

Estimated reagent and consumables are summarized in Table 17-2.

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Table 17-2 Summary of Reagents and Consumables

Process	Reagent/Consumable	Consumption (kg/t Mill Feed)
Crushing	Jaw Crusher Liner	0.020
	Cone Crusher Liner	0.016
Grinding	Ball Mill Liners	0.109
	Grinding Media (Balls)	1.000
	Re-grind Media (Balls)	0.015
Flotation	Sodium Sulphide (Na_2S)	0.250
	Potassium Amyl Xanthate (PAX)	0.101
	MIBC frother	0.028
	Copper Sulphate ($CuSO_4$)	0.250
	Sodium Silicate (Na_2SiO_3)	0.013
BIOX™	Limestone ($CaCO_3$)	46.232
	Lime ($Ca(OH)_2$)	6.863
	Nutrients	1.782
	Antiscalant	0.004
	Corrosion Inhibitor	0.004
	Biocide	0.004
	Defoamer	0.001
	Flocculant	0.004
CIL/Elution	Sodium Cyanide (NaCN)	1.5
	Activated Carbon	0.030
	Defoamer	0.038
	Lime (Anhydrous) (CaO)	4.0
	Hydrochloric Acid (HCl)	0.076
	Sodium Hydroxide (NaOH)	0.198
Detox/ASTER™	Ammonium Hydrogen Phosphate ($NH_4H_2PO_4$)	0.023
	Molasses	0.015
	Sodium Metabisulphite ($Na_2S_2O_5$)	1.0

17.5 Process Water and Power

A water balance for the New Polaris processing plant has not been completed.

Water supply is described below in Item 18 (Project Infrastructure), along with potable and fire water. Raw water will be stored at the plant site in a fresh water tank. Recycled water from the plant will come from the detoxification stage described above and will be stored in a process water tank.

Electrical power requirement for the New Polaris project is estimated at an average 2,900 kW of operating power and approximately 5,000 kW of connected power.

18 Project Infrastructure

The following Section discusses the project infrastructure planned for the Project, including the Overall Site Development, Site Services, Surface Mobile Equipment, Buildings and Facilities, and the Co-Disposal Facility (CDF). Plans for the PEA are conceptual in nature so the quantities and dimensions listed below are provided only to give an understanding of the scope of what has been included in the cost estimates listed in Section 21. The reader is cautioned that as further engineering is completed on the Project the plans listed below could be significantly altered.

The site general arrangement is shown in Figure 1-2.

18.1 Overall Site Development

18.1.1 General Site Work

The New Polaris site is a re-development of the former mine and town site of the past producing Polaris Taku mine, which operated on and off between 1937 and 1951. General site preparations include clearing of previously developed ground and overgrowth to accommodate the expansion of the existing exploration site to include all required infrastructure for the operation of the New Polaris mine.

18.1.2 Airstrip Improvements

The existing New Polaris airstrip is too small to accommodate aircraft appropriately sized for the mine during full operations. The airstrip will be realigned to extend it by 280 m up to a total length of 860 m. The airstrip will also be widened from 50 m to 60 m. This will provide sufficient length and width to accommodate a DHC-5 Buffalo, or similar aircraft, capable of carrying up to 40 passengers, or a mixture of passengers and cargo. This is the maximum length attainable without re-routing existing creeks/drainages or relocating the entire airstrip.

The airstrip realignment will require clearing and grubbing to prep the airstrip alignment. An assumed 50 cm compacted gravel base course will provide the running surface of the airstrip. The airstrip will require year-round maintenance to ensure that the runway surface meets the specifications of charter airline services operating on the airstrip. This entails grading and periodic placement of additional gravel as required, as well as snow clearing during winter months.

18.1.3 Roads and Access

The existing site road network has not been actively maintained since the last mine closure in 1951 and is therefore assumed to be in varying states of disrepair, subject to overgrowth, road wash outs, and other seasonal/environmental effects.

While some minor upgrading and maintenance has been completed in order to facilitate recent exploration at the site, it is assumed that in order to bring the site up to full operational levels, it will require a combination of new road works as well as upgrading of existing road networks. Where possible the new road network will utilize existing alignments to reduce costs and environmental footprints.

18.1.4 Road to Barge Landing

The most significant road building and upgrading activity is to re-establish a haul road between the mine site and the proposed barge landing in order to facilitate transport of material goods and consumables into the site during the barging season.

The haul road is approximately 10 km in length, with 2.5 km requiring balanced cut/fill road building running south from the site along the hillside above the Tulsequah River. The remaining 7.5 km follows a similar alignment to the original haul road, running southeast between the west bank of the Tulsequah river and the Flannigan Slough. The haul road will be 4.5 m in width, allowing for single lane traffic with an allowance for shoulders. Pull outs will be placed at intervals to allow passing of two-way traffic appropriate to the level of service of the road. Radio control of the road will also be required.

Road building and upgrading activities include clearing and grubbing, as well as trimming back overgrowth where required. A minimum 25 cm of road coarse gravel (locally sourced) will be used for the running width of the haul road. This minimum increases to 50 cm for sections running along the edge of the Flannigan Slough. Five culverts and one steel bridge will be required to cross drainages along the 10 km haul road.

18.1.5 Road to CDF

A haul road between the mine site and the CDF to facilitate transport of waste rock and tailings will be constructed.

The haul road is approximately 2 km in length, requiring balanced cut/fill road building. The haul road will be 4.5 m in width, allowing for single lane traffic with an allowance for shoulders. Pull outs will be placed at intervals to allow passing of two-way traffic appropriate to the level of service of the road. Radio control of the road will also be required.

Road building and upgrading activities include clearing and grubbing, as well as trimming back overgrowth where required. A minimum 25 cm of road coarse gravel (locally sourced) will be used for the running width of the haul road. One 25 m culvert will be required to cross drainages along the haul road.

18.1.6 Barge Landing and Dock

The barge landing will be located about 0.5 km downstream from the confluence of the Tulsequah River and the Taku River, on the north side of the Taku. The barge landing will be constructed from wood and steel piles, backed with rock fill gabion baskets and earth fill to create a level platform along the river edge capable of docking up to two barges simultaneously. The barge landing facility will also include a small office trailer, genset, diesel fuel day tank, temporary storage area, and container handler and/or mobile crane.

18.1.7 Aggregate Supply

All aggregate supply will be locally sourced to meet infrastructure needs as required. Aggregate is assumed to require screening but no crushing.

18.1.8 Topsoil Storage

Topsoil excavated during site development will be appropriately stored for later reclamation use.

18.1.9 Power Supply and Distribution

Total installed power requirements for the site are estimated to be 6.9 MW, with a net load of 3.9 MW per hour. Power consumption is estimated from a preliminary estimation of connected power, typical power load factor, and utilization time.

Table 18-1 Total Site Power Requirements

Area	Installed MW	Net Load MW per hour	Power Consumption MW per Year
Mine	1.1	0.7	6.2
Process Plant	5.0	2.9	25.3
Site and Camp	0.8	0.3	2.6

New Polaris will obtain power from onsite diesel generators. Power generation capacity is estimated using a conversion of 3.7 kWh/L of diesel.

Seven 1 MW diesel generators will form the main power supply for the site, with an additional two 1 MW diesel generators for backup and maintenance. The generators will be placed adjacent to one another with day tanks servicing each unit, all housed within a power generating facility.

Power will be transformed down to 600V for surface distribution of power to the process facility and other site infrastructure. The layout of the surface power distribution is subject to final layout of all site infrastructures.

18.1.10 Communications

A satellite-based internet and surface telephone/radio communication system is planned.

The underground mine will use a leaky feeder system to maintain communications between the surface and underground.

18.2 Site Services

18.2.1 Compressed Air Plant and Compressors

A compressed air plant and air compressors to supply the underground operations, mill and surface operations with a compressed air supply are included.

18.2.2 Fresh, Fire and Potable Water

Fresh water will be sourced on site and piped on surface to a treatment plant as well as a holding tank for firefighting. Surface piping will then transport potable water to all required infrastructure. The treatment plant will include a sand filter, chlorination equipment, pumps and day tank; all pre-assembled within a shipping container.

18.2.3 Sewage/Septic

Sewage will be piped along similar alignments as the potable water, transporting it to a sewage treatment facility. All required tankage, pumps and control systems are pre-assembled within a shipping container. Treated effluent will be discharged to a septic field while solids are dewatered and bagged for waste disposal.

18.2.4 Waste Disposal

Disposal of domestic, sanitary, and other waste generated by the camp and other site facilities during operations will be incinerated on site using a skid mounted diesel fueled incinerator located in an enclosed shed facility.

18.2.5 Fuel Storage and Distribution

Bulk fuel storage will consist of a tank farm with capacity for approximately 6 million liters of diesel fuel, stored in 11 equal sized steel tanks. The tank farm will be surrounded by an earth berm and impoundment, constructed from locally sourced compacted sand and gravel. The berm and impoundment will be lined with an impermeable membrane to contain fuel in the event of a breach or leak from one or more tanks.

An offload and dispensing system will be located at the bulk fuel storage site to facilitate stockpiling of fuel during the barging season, and distribution around the site using a single tanker truck.

Smaller “day tanks” will be located at various locations throughout the site to supply fuel for equipment and power generation as required for the camp, mine, plant, truck maintenance facility, barge landing, and other infrastructure sites. These tanks will be filled by the tanker truck as required.

Estimated annual fuel consumption is shown in the following Table:

Table 18-2 Estimated Annual Project Fuel Requirements

Area	Fuel Consumption L per Year (x1,000,000)
Mine Power	1.7
Mine Mobile Fleet	1.4
Process Plant Power	6.8
Camp Power	0.7
Surface Mobile Fleet	0.4
Total	**11.0**

18.2.6 Explosives Storage

Explosives will be stored on surface in a secure gated facility, located outside the minimum distance to inhabited areas as outlined by Natural Resources Canada guidelines.

Explosives will be barged in during the barging season and stockpiled in the explosive's storage facility for the remainder of the non-barging season. Transport of explosives from the storage facility for use either in the mine or on surface will be carried out by the surface fleet of flatbed trucks.

The facility will be comprised of eight 40-foot shipping containers, each surrounded by an earth berm to prevent propagation in the event of an unplanned detonation. Each container will be capable of storing up to 23,000 kg of explosives, with the total capacity of the facility capable of storing up to one year's worth of explosives. Future planning may reduce this to account for the 100-day shipping season.

18.2.7 Portal Rehabilitation

New portal development will be required in advance of mine operations. It is possible that this portal development will be done in advance of additional exploration activities.

18.3 Mobile Fleet

18.3.1 Plant Mobile Fleet

The plant mobile equipment includes a wheel loader to load the crusher, pickup trucks for personnel transport and a fuel truck to transport fuel for power generation.

Table 18-3 Plant Mobile Equipment Fleet

Item	Units
Front End Loader	1
Pickup Trucks	4
Fuel Truck	1

18.3.2 Site and Road Maintenance Fleet

The surface mobile equipment is used for site preparation and maintenance, aggregate supply, tailings haulage and compaction, snow clearing, equipment and material transports, and barge loading. The following equipment make up the fleet. The miscellaneous item covers off a transport bus, manlifts, maintenance personnel vehicles, container handlers and a crane for barge loading.

Table 18-4 Surface Mobile Equipment Fleet

Item	Units
Motor Grader	1
Track Dozer	1
Gravel Truck	1
Tailings Trucks	2
Snow plow	1
Compactor	1
Flatbed Picker Truck	1
Forklift	2
Tool Carrier	1
Miscellaneous	1

18.4 Buildings and Facilities

18.4.1 Camp, Administration Building and Dry

The camp, mine dry and administration buildings are assumed to be made up of prefabricated trailer units. The camp portion is sized for 90 persons. Minimal installation requirements include clearing and grubbing, some small pile foundation work, and hook up of site services. Opportunities to use existing site building for accommodations or offices have not been examined. The costs of refurbishment may be less than supply and installation of trailer units.

18.4.2 Assay Lab

Lab and assay facilities will utilize two modified shipping containers and will be placed adjacent to the mill building.

18.4.3 Maintenance Facility

The maintenance facility will be a steel framed structure built on a concrete slab with insulated fabric cladding. It will include three maintenance bays and one wash bay, with either a single overhead crane or mobile gantry cranes to service all bays. Warehousing of parts and consumables for the maintenance facility will utilize shipping containers located adjacent to the maintenance facility.

18.4.4 Warehouses

Other buildings required on site will utilize fabric structures placed on concrete slabs, including a 20 m by 25 m heated warehouse and a 24 m by 46 m cold warehouse.

18.4.5 Co-Disposal Facility

New Polaris process tails and waste rock is planned to be co-disposed in a facility constructed 2.5 km north of the deposit, portal and process plant. The location of this Co-Disposal Facility (CDF) is shown as the 'Tailings Site' in Figure 18-1 below. The plan described below, the location chosen, and all costing associated with construction of the CDF is conceptual in nature. Detailed site studies, geotechnical analysis and construction engineering is required in future studies to confirm the adequacy of site selection, design assumptions and cost estimates used in this Report.

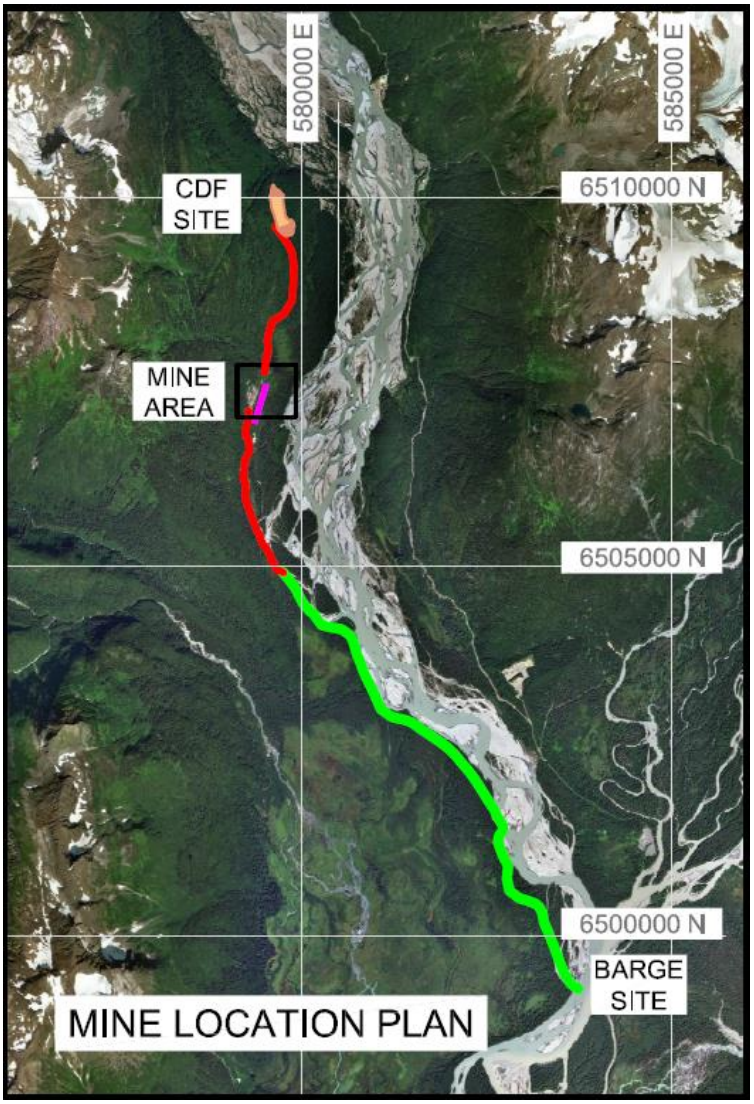


Figure 18-1 CDF in relation to Project

The co-disposal methodology employed will be co-placement. Co-placement creates an integrated disposal facility with very little mixing between the process tails and the waste rock. Benefits include:

- When mine waste rock is potentially acid generating (PAG) and susceptible to metal leaching, the placement of waste rock with the tailings can reduce the access of oxygen to the rock, reducing the potential for acid generation and metal leaching.
- Inclusion of mine waste rock with the tailings improves the stability of the facility and reduces erodibility of the process tails.
- Co-placement reduces the footprint area and simplifies the water management strategy, compared to disposing of waste rock and tailings separately.
- Co-placement accelerates tailings consolidation facilitating earlier closure and generally reduces closure costs.

The underground operations will generate ~800,000 tonnes of waste rock via development ramps, drifts and raises over the planned 8.7-year mine life. This waste rock will be hauled from underground to the CDF and co-disposed of with the process tails.

Ore from the underground operations will be fed into the process plant where it will produce doré bars and tailings products. The process plant will thicken the tailings product from both the flotation and BIOX/CIL circuits and then split the thickened tailings into two streams.

1. 42% of the tails will be pumped underground to a paste plant and dispersal system. These tails will be disposed of in the mined-out voids (stopes) underground.
2. 58% of the tails will be sent through ceramic cyclone filters and filtered to a semi-dry state. These filtered tails will be loaded into haul trucks, hauled to the CDF, and co-disposed of with the mine waste rock.

Some of the planned features of the New Polaris CDF include:

- The surface area of the CDF will be geotechnically prepared ahead of any material placement.
- The entire footprint of the CDF will be lined as required.
- Mine waste rock will be used to consolidate the impoundment with a final slope suitable for final reclamation.
- Waste rock and filtered tails will then be co-disposed within the CDF on suitably sized lifts. Layers can be created by constructing a series of disposal cells utilizing the waste rock and filtered tails.
- Layers will be compacted as required.
- An allowance for additional borrow pit waste rock has been included in the estimated CDF volumes and construction cost.
- The layers will be sloped to manage runoff, and all perimeter berms will be permeable. The free draining nature of the CDF will not allow pore pressures to build up.
- Lined ditching surrounding the CDF will direct water to a lined settling pond where it will eventually report to a water treatment plant.
- The CDF will be designed to be stable in both frozen and unfrozen states.
- During construction, and at closure, a rock, soil, or geosynthetic (or combination) cover will be placed over the entire facility, encapsulating the co-disposed tailings and mine rock. An allowance for borrow pit material for rock or soil has been included.

The conceptual layout of this facility is shown in Figure 18-2 below. Total planned volumes of tails and waste rock are 1,320,000 m^3. Conceptual linework for the perimeter clean water diversion ditches and downstream settling ponds are shown. The haul road from the process plant and underground portal to the CDF is planned to be constructed on the mountain side west of the floodplain.

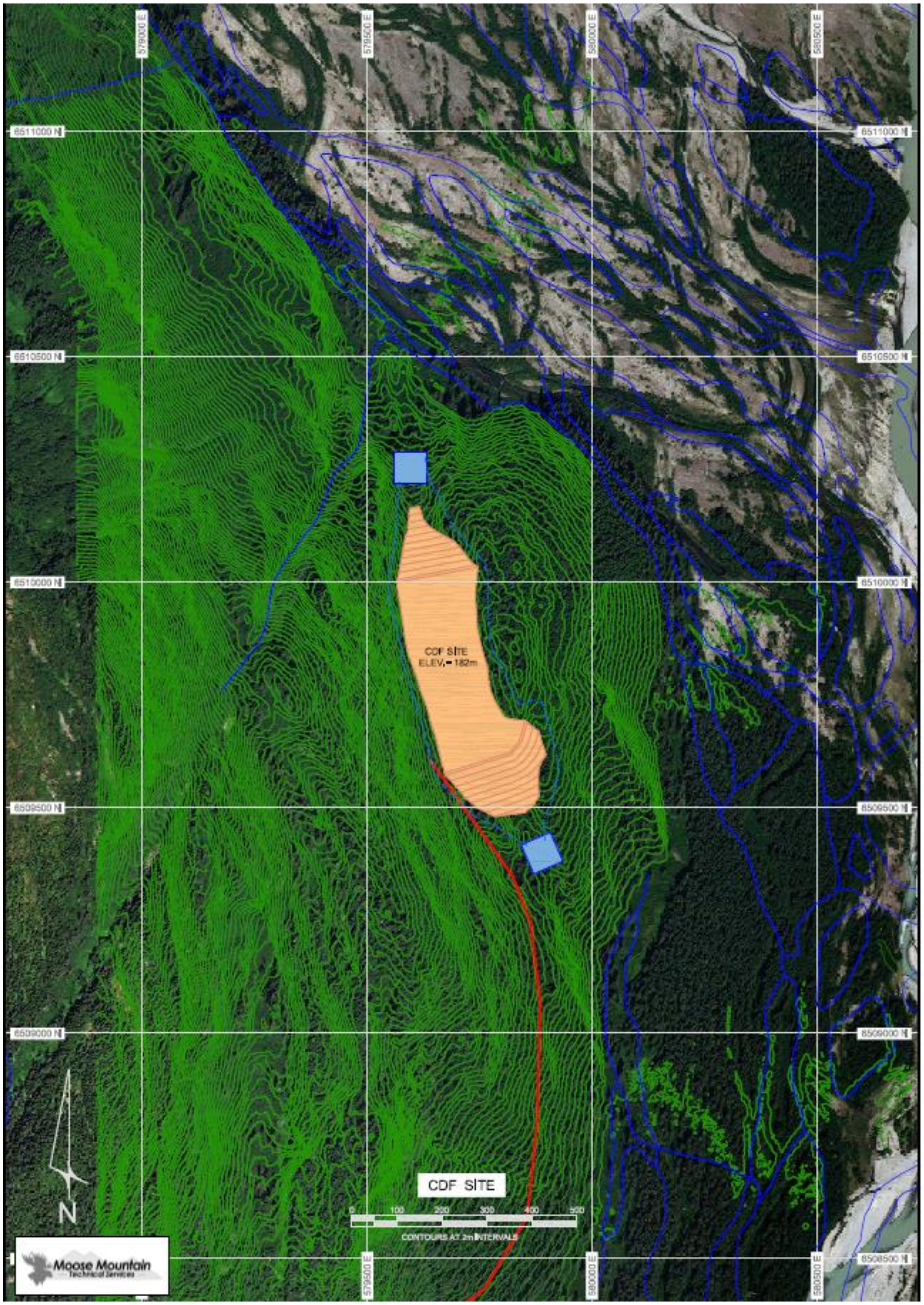


Figure 18-2 Conceptual Layout for New Polaris CDF

19 Market Studies and Contracts

The Project will yield gold doré as its final product, which is expected to be sold on the spot market through marketing experts retained by Canarc. Gold can be readily sold on numerous markets throughout the world; its market price at any time is easily and reliably ascertained. The large number of available gold purchasers, both domestically and internationally, allow for gold production to be sold on a regular and predictable basis, and on a competitive basis with respect to the spot price.

A long-term gold market price of US$1,300/oz is considered by the QP as reasonable with respect to the prevailing market and has been used in the PEA. The QP expects that terms, conditions and charges for doré sales will be typical of similar contracts in the industry. Offsite costs and payable percentages are applied to the gold market price.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental

20.1.1 Aquatic/Terrestrial

Several environmental baseline studies have been initiated and either completed or suspended in 1997, 2007 and again in 2015, however more work is required to meet regulatory requirements. It is possible that a large portion of the dataset will need to be refreshed to reflect current conditions. These include:

- Water quality and quantity (surface and ground)
- Risk Assessment Items for Terrestrial and Aquatic Resources (vegetation, sediment, invertebrates, periphyton, soils etc.)
- Air Quality/Meteorology/Climate

Fish and fish habitat have been characterized in the project area several times beginning in 1997. Spawning and rearing habitat that supports both sea-run and freshwater salmonid species including chinook, coho, pink, and sockeye salmon. The mainstem Tulsequah River has been shown to afford poor fisheries values due to low water temperatures and lack of in-stream cover typical of glacial runoff systems. Based on observations, Whitewater Creek in the project area contains fish rearing and spawning habitat superior to the Tulsequah River and therefore remains critical habitat necessary for sustaining populations (GLL 1997). Knowing this, for a project to be deemed acceptable to government and other stakeholders, future development will have to carefully consider the streams and the riparian habitat which encompasses them.

The project area contains populations of large ungulates and predators. Past studies have identified local species present, however additional work is required to support impact and management plans.

Critical components of impact mitigation include Management Plans for land, water, air, wildlife, fisheries and groundwater. The project will be designed to minimize environmental impacts during the construction and operating phases of the mine and to minimize any long-term environmental impacts. Future work will incorporate and advance the results of all previously completed aquatic, hydrogeology and terrestrial work on site.

20.1.2 Geochemistry

Canarc retained URS Canada Inc. to assess the acid rock drainage and metal leaching potential of major rock units anticipated to be exposed. Studies were completed by static testing conducted on 27 “fresh” rock drill core samples, collected from sections of the C vein, and flotation testing was conducted on 5 test tailings samples. A criterion of 2.0 weight % sulphide Sulphur has been developed to distinguish between non-acid generating and potentially acid generating materials. For waste rock, sampling indicated, generally, that the rock is non-acid generating in nature, due to high carbonate content. It should be noted there were some localized areas that showed some potential for acid generation. For tailings, based on static testing of 5 samples, tailings are not expected to be acid generating. This was indicated by sulphide Sulphur content being typically less than 0.3% and the corresponding carbonate content being greater than 3.5%.

Total metals analysis and leachate extraction test results for hanging wall rock, vein rock and footwall rock indicate a high potential for leaching of arsenic and antimony. Results found in tailings had similarly elevated concentrations. Further studies, although only in draft form at the time of this writing, showed these occurred in low concentrations to the point where it was speculated these lithologies are considered to have low metal leaching potential. More studies, including kinetic testing (humidity cell) are required to further characterize waste rock, tailings and effluent leaching and ARD potential (URS 2007).

Tailings are planned to be disposed of in both underground and surface facilities. Tailings on surface will be thickened and disposed of in a facility that does not store water. Underground disposal will be directed to old workings to minimize the total surface disturbance footprint.

20.1.3 Ore Recovery

The processing of ore will include a typical mill feed system closed off to the environment. It is also important to note that the introduction of bio-leach of concentrates on-site to produce doré bars, is proposed. The use of cyanide, as proposed, to leach metals from rock is a process generally acceptable in BC provided the facility is completely closed off from the environment and the facility meets rigorous environmental standards and practices including transport, handling, storage and deconstruction of the chemical.

20.1.4 Reclamation/Closure

The reclamation of the mine site, its associated infrastructure and any post mining effluent treatment is a requirement of the Mines Act, and accordingly will require a detailed Plan and Reclamation Bond. The reclamation plan and bonding process will require regulatory and First Nations consultation and agreement prior to the start of mining and processing operations.

Much of the design and projected costs will be a result of planned future studies. An allowance of $5M has been included for closure, including removal of all equipment and supplies and reclamation of the site. This estimate reflects professional experience based on examples of similarly sized, remote projects.

20.2 Regulatory Framework

The New Polaris Project will be subject to approvals under BCEAO and CEAA as its production threshold exceeds the requirements for mining activities permittable under those processes. Additional authorizations and input may also be required from United States authorities as the Project area drains into US waters (Transboundary Water). Concerns over water quality and overall ecosystem health will likely drive the discussion from an environmental approval standpoint.

Other requirements of Provincial and Federal Acts and Regulations may also apply, depending upon final design components.

It is possible and desirable to harmonize the required processes. The application process can be developed to meet the requirements of the Canada-British Columbia Environmental Assessment

Delegation Agreement which delegates the federal screening level environmental assessment and preparation of the screening report to the BCEAO, therefore meeting the requirements of both the BCEAA and the CEAA. The intent of the agreement processes is to allow for parallel activities to be conducted, securing all necessary permits and approvals in the most time and cost-effective manner.

A preliminary discussion of the key parameters of permitting and the related timelines are provided below (Figure 20-1), based on known design parameters, and information derived from Provincial and Federal process websites.

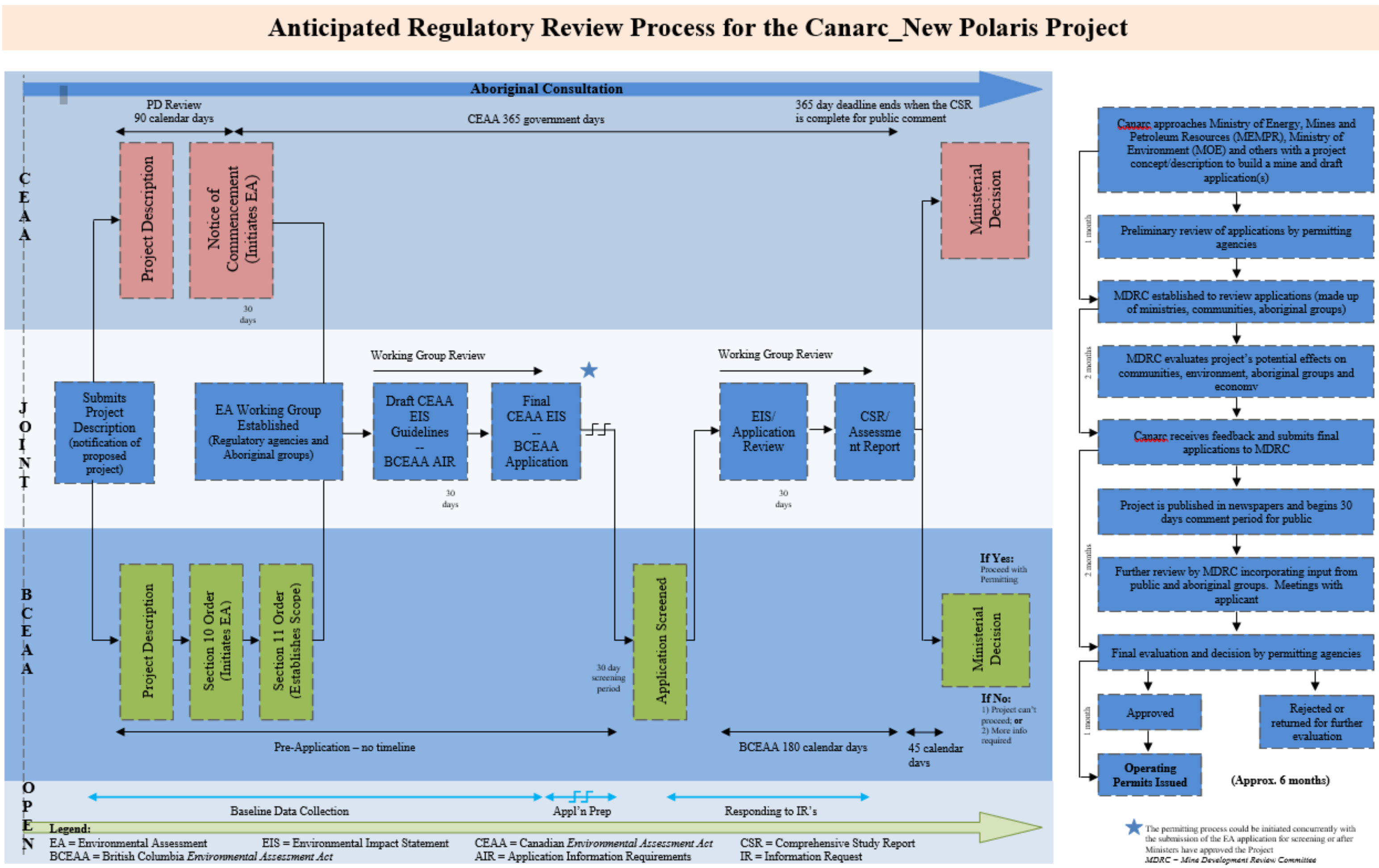


Figure 20-1 Anticipated Regulatory Review Process

20.2.1 Provincial Processes

The EAO works with First Nations, government agencies and the public to ensure major projects are developed in a sustainable manner. The EA process examines major projects for potentially adverse environmental, economic, social, health and heritage effects that may occur during the lifecycle of these projects.

Public participation in the environmental assessment process helps to ensure that community values and public goals for community development are considered in project planning and decision-making, and is an important component of permitting.

The BCEAA process is a two-step process that leads to the issuance of an Environmental Assessment Certificate (EAC), the necessary prerequisite for other approvals. There are two stages of B.C.'s EA review process: pre-application and application review. During pre-application, the focus is on developing project-specific EA procedures, scope of issues, and information requirements that describe what is to be contained in an application for EA certification (Application). During the application review stage, the focus is on the adequacy of the information provided and the significance of potential impacts and permits.

A thorough and complete EA application must provide a wide range of detailed information on a project and its potential effects. The EAO ensures that, in advance of applying for an EAC, a proponent develops an acceptable outline of the EA application through Application Information Requirements (AIR) (formerly "Terms of Reference"), which describes the range of data, analyses and reporting the EA application will contain. It is anticipated that a full range of assessments including environmental, social, economic, heritage and health assessments will be required. The assessment of potential effects on First Nations is also required.

The process of pre-application, application, review and approval may take up to 3 years or more; depending upon technical complexity, consultation requirements, and the significance of potential impacts.

Once issued, the EA certificate remains in effect for the life of the project, unless suspended or cancelled by the Minister of Environment for non-compliance with the BCEAA. All Certificates also contain a deadline of between three and five years from date of issuance to substantial start.

The estimated time to secure all necessary Provincial approvals for The Project to proceed is 1 – 1.5 years from an acceptable Project Description, however complexity of issues and other issues can delay approval significantly, depending upon the complexity of issues and the acceptability of recommended mitigation of proposed impacts.

A significant aspect of permit application for The Project will include the need for an acceptable Mine and Reclamation Plan, an Environmental Management System, a Sediment Control and Water and Waste Management Plan, and a Mine Abandonment Plan. Other specific Environmental Plans may include Fish Habitat Mitigation, Wildlife Habitat Mitigation, Special Waste Management, and others.

Cost and time for major Environmental Plans are not included as the scope of their requirements have not been fully developed. Relevant Provincial Acts and Regulations will apply.

20.2.2 Federal Processes

Federal environmental assessments must be conducted prior to a project proceeding if: a federal authority is the proponent of the project, federal money is involved, the project involves land in which a federal authority has an interest, or some aspect of the project requires federal approval or authorization.

Federal assessments will likely focus on areas of particular interest to the federal authorities such as species at risk, effects of accidents and malfunctions, effects of the environment on the project, effects of the project on the capacity of renewable resources, cumulative effects, and First Nations engagement and consultation.

Project proponents should be diligent in considering mine plan alternatives and details that do not require federal approvals triggering the full CEAA process.

It is not unreasonable in this jurisdiction to expect approvals will be received for the New Polaris Project conditional upon acceptance of respective Management Plans and commitments.

Land Use Planning

The New Polaris Project lies within the area of the Atlin/Taku Land Use Plan (2011). The plan allows for Resource Development, including mining, contingent upon meeting specific requirements of Management Plans for specifies issues, including those of particular interest to the local First Nations. There are 7 objectives set in the affected land use plan concerning mineral extraction/exploration with respect to engaging with stakeholders. Above all, but not to reduce significance from others, Objective #2 states that a proponent must ensure mineral exploration and development are undertaken in a socially and environmentally responsible manner. It will be extremely important to respectfully engage with local First Nations and community stake holders to establish a positive relationship and to understand concerns with the intention to incorporate them into design, operations and closure.

First Nations

The New Polaris Project is situated on the asserted Traditional Territory of the Taku River Tlingit First Nation, the only First Nation recognized as having status in this area to date.

Canarc has developed a good working relationship with the TRTFN, and although no formal agreements are in place, communications between Canarc and TRTFN indicate that the TRTFN are generally in favor of development of projects within their traditional territory, with the condition that an acceptable Management Plan that addresses key activities of particular interest to the TRTFN be developed with them. The Management Plan as required in the Atlin/Taku Land Use Plan provides Implementation Direction for the Resource Management Zone that contains the New Polaris Project site.

The draft Gap Analysis developed by Gartner Lee in 2007 referenced a Conservation Area Design and TRTFN Vision and Management Direction report, which provides some general insight into current and past traditional use and harvesting practices.

In addition, other recently proposed projects in the area, prepared for BCEAO, provide detailed accounts of current land use, social and economic relations, valued components and an account of traditional use and wage based economic activity. A documented oral history is also available that identifies the Tulsequah River as a place where harvesting activity took place.

To meet regulatory requirements, it will be necessary to prepare a detailed Traditional Knowledge Study providing more information on current and historical traditional use specific to the New Polaris Project area.

Archeological and Heritage Resources of the Project area also require further work. Although the database maintained by the Archeology Branch found no conflict between known sites and the Project, it will be necessary to confirm with the TRTFN whether all know sites have been fully registered, as that work was seen to be incomplete. In addition, the Province requires that an Archeological Overview Report be developed for the Project Area, and that Archeological Impact Assessments may be required if archeological resources are identified as having high potential in the Project area footprint of disturbance.

Canarc will be required to develop a Consultation and Engagement Plan with the TRTFN, provide timelines and details of Consultation, and document any agreements reached between them in respect of the New Polaris Project.

Community/Social

The communities of Atlin, Five Mile Point and Unnamed No. 10 have been recognized as being within the area potentially affected by the New Polaris Project, although the very limited data available from Statistics Canada indicate that it will be important to collect current information.

Community and Social Consultation and Engagement will require a Plan and Implementation Strategy developed with the input of identified communities and other potentially affected stakeholders.

20.3 Socio-Economic

The mine is located in an area that has low usage for mining, exploration, hunting, fishing, trapping, and logging activities. The New Polaris site was previously mined between 1938 and 1956 with remnants of the old activities still being present at the site.

The project is located within the land claim and traditional territory of the Taku River Tlingit First Nation (TRTFN).

No formal agreements are in place with the Taku River Tlingit First Nation. Discussions have taken place in previous years but no agreements completed. It is expected that the project will enhance employment opportunities for the people of the TRTFN during the construction, operation and closure of the Project. Operational training as well as trades training opportunities will also be made available for the members of the TRTFN on a preferential basis. A number of other benefits will accrue to Atlin through funding of social events, scholarships for higher education, and community enhancement programs.

During the exploration phase of the project a high percentage of employees have been from Atlin and the surrounding area. It is the intention of Canarc to continue to operate in a fashion that ensures the local community and its citizens continue to benefit from the construction and operation of the mine.

The sourcing of qualified and experienced underground miners, process personnel, and tradesmen is a concern, particularly with the current labour shortages in Western Canada. Canarc will continue to source appropriate personnel as the project advances. A contract mining company will also be pursued.

Since access to the site for major supplies will use shallow barges up the Taku River it is likely that a large number of supplies may be purchased in Juneau. Supplies not available in Juneau will be purchased in Southern BC, Canada or the United States and shipped to site through Seattle and Juneau. Barging will done using independent contractors who have the required equipment and have the necessary experience with this type of service. The major items needed for the operation will be diesel fuel, ground support supplies, mill reagents & supplies, explosives and a variety of components for equipment maintenance.

The current plan is based on a fly-in fly-out rotation and an onsite camp. Air transportation will be used for transporting employees and perishable items or small items needed to sustain the operation. As other projects in the area are developed the opportunity may arise to use access and infrastructure developed for those other projects.

Continued work in negotiations with First Nations, labour force planning, and the impact on local infrastructure is recommended.

It is not unreasonable to expect support from local communities contingent upon the Project proponents developing acceptable plans for mitigation and final reclamation of project impacts.

21 Capital and Operating Costs

21.1 Capital Cost Estimate

Capital cost estimates are derived from a combination of MMTS experience in similar projects and consultation with contractors and equipment suppliers. The estimated capital cost breakdown is shown in Table 21-1 below.

All dollar amounts are expressed in Q1 2019 Canadian dollars, unless specified otherwise.

The accuracy range of the capital cost estimate is +/- 30 %.

Table 21-1 Project Capital Cost Estimate

Area	Capital Estimate (M$)
Overall Site	$12.8
Mining	$20.0
Process Plant	$39.8
Co-Disposal Facility	$6.8
Site Services	$4.3
Surface Mobile Fleet	$2.5
Buildings and Facilities	$8.4
Total Direct Costs	$94.5
Total Indirect Costs	$25.2
Contingency (20%)	$23.9
Total	**$143.7**

Sustaining capital is estimated as $56.4 million over the LOM, comprised of extensions to underground development, mobile equipment purchases, CDF construction, water treatment expenses and a nominal allowance for infrastructure maintenance and upgrades.

The listed capital costs include delivery to the site and assembly but do not include the following:

- Force majeure
- schedule delays such as those caused by
    - major scope changes
    - unidentified ground conditions
    - labour disputes
    - environmental permitting activities
    - abnormally adverse weather conditions
- cost of financing (including interests incurred during construction)
- GST
- royalties

- cost of this study
- sunk costs for exploration, technical studies, and permitting.

21.1.1 Direct Capital Costs

21.1.2 Overall Site

Descriptions of items included in the overall site are included in Section 18.1. Cost estimates are shown in Table 21-2.

Table 21-2 Overall Site Capital Cost Estimate

Item	Cost Estimate (M$)
General Site Work	$0.2
Airstrip Improvements	$0.5
Roads and Access	$1.2
Barge Landing and Dock	$1.4
Power Supply and Distribution	$9.3
Communications	$0.2

21.1.3 Mining

Capital for mining is comprised of the following items:

- Operations Infrastructure
- Mobile Fleet
- Underground Development
- Mine operating costs incurred ahead of commercial production (Pre-Production Operating Costs)

Descriptions for these items can be found in Section 16 of the Report. A breakdown of the mine capital estimate is shown in Table 21-3.

Table 21-3 Mining Capital Cost Estimate

Item	Initial Capital Cost Estimate (M$)	Sustaining Capital Cost Estimate (M$)
Operations Infrastructure	$3.4	$1.0
Mobile Fleet	$4.1	$11.5
Underground Development	$8.8	$31.3
Pre-Production Operating Costs	$3.7	

Operations Infrastructure includes estimates for the following items:

- Compressed Air Plant
- Tailings Paste Backfill Plant and Distribution System
- Communication Systems and Leaky Feeder Cable
- Refuge Stations
- Mine Rescue Gear
- Underground Shop Equipment and Tools
- Underground Electrical Distribution
- Warehouse Spares
- Pumps
- Ventilation Fans
- Jacklegs
- ANFO Loader
- Battery Chargers and Cap Lamps
- Hoses and Fittings

Most of the mine mobile fleet is assumed to be purchased under a 60-month lease arrangement carrying an 8% interest rate. The lease arrangements assume 10% down payment and 0% residual value. The following equipment is included in the mine mobile fleet:

- Drill Jumbo (2 x 2-Boom Production Jumbos and 1 x Bolting Jumbo)
- Longhole Drill
- Scoops (1 x 2 yd^3 bucket LHD and 3 x 3.5 yd^3 bucket LHD)
- Haul trucks (3 x 28 t payload)
- Scissor Truck
- Loading Truck
- Exploration Diamond Drill
- Lube/Service Truck
- Fuel Truck
- Raise Bore Machine
- Dozer
- Grader
- Supply Haulage
- Personnel Carriers (2)
- Light Vehicles (4)

Underground Development includes the costs to drive the main ramp from surface down to the identified production levels, drive the footwall drifts and crosscuts along each production level, develop sumps on each level, and drive vertical service raises between the production levels and to surface for ventilation. Initial capital costs include development to Level 2. Development down to Level 9 is included in the sustaining capital cost estimate. Direct face development costs average $2,250 per metre.

During the Pre-Production period some mine operating costs associated with stope production are incurred. These costs have been capitalized.

21.1.4 Process Plant

Descriptions for the process plant are included in Section 17 for the Report. The process plant capital costs are summarized in Table 21-4 below.

Table 21-4 Process Plant Capital Cost Estimate

Item	Cost Estimate (M$)
Crushing and Grinding	$8.6
Flotation	$3.4
BIOX Plant	$21.2
CIL Circuit	$3.4
Detox Circuit	$1.6
Filtration	$1.7

21.1.5 Co-Disposal Facility

Descriptions for the CDF are included in Section 18.4.5 of the Report. A breakdown of the CDF capital estimate is shown in Table 21-5.

Table 21-5 CDF Capital Cost Estimate

Item	Initial Capital Cost Estimate (M$)	Sustaining Capital Cost Estimate (M$)
Foundation Preparation	$0.3	$0.2
Liner and Bedding	$1.6	$1.1
Rock Cover	$1.2	$2.8
Settling Ponds	$0.6	
Ditching and Water Management	$0.6	
Water Treatment Plant	$2.5	$4.5

Conceptual layouts for the CDF cover 13.5 ha. Settling ponds are assumed to be 50,000 m^3. Ditching for water management is assumed to be 2 km long. Most of the CDF is comprised of development rock from the underground operations and tails from the process plant; a nominal 200,000 m^3 of additional rock cover is assumed to be sourced from a nearby borrow pit.

Sustaining capital costs for the water treatment plant include the operating costs for the plant.

21.1.6 Site Services

Descriptions of items included in site services are included in Section 18.2. Cost estimates are shown in Table 21-6.

Table 21-6 Site Services Capital Cost Estimate

Item	Cost Estimate (M$)
Compressed Air Plant and Compressors	$0.2
Fresh, Fire and Potable Water	$0.3
Sewage/Septic	$0.6
Waste Disposal	$0.4
Fuel Storage and Distribution	$2.8
Explosives Storage	$0.1
Portal Rehabilitation	$0.1

21.1.7 Surface Mobile Fleet

Descriptions of items included in the surface mobile fleet are included in Section 18.3. Cost of the fleet is estimated to be $2,500,000.

21.1.8 Buildings and Facilities

Descriptions of items included in buildings and facilities are included in Section 18.4. Cost estimates are shown in Table 21-7.

Table 21-7 Buildings and Facilities Capital Cost Estimate

Item	Cost Estimate (M$)
Camp	$3.9
Administration	$0.7
Mine Dry	$0.8
Assay Lab	$0.5
Maintenance Facilities	$1.2
Warehouses	$1.3

Sustaining capital of $4,000,000 is estimated for infrastructure maintenance and upgrades over the LOM. This sustaining capital is assumed to cover the items listed in site services, surface mobile equipment and buildings and facilities.

21.1.9 Indirect Capital Costs

Total indirect costs are shown in Table 21-8 below.

Table 21-8 Indirect Capital Cost Estimate

Item	Cost Estimate (M$)
Construction Indirects (10%)	$7.2
EPCM (10%)	$7.2
Spares (2%)	$1.1
Initial Fills (2%)	$1.1
Freight and Logistics (8%)	$3.5
Commissioning	$2.0
Vendor Representatives	$0.6
Owner's Costs	$2.5

Construction indirect costs are estimated as a percentage of the direct construction costs. Construction indirect costs are not applied to surface mobile fleet or mining capital items. An indirect factor of 10% of direct construction capital costs is used. This cost allows for charges contractors would apply or include in their rates, including but not limited to:

- Temporary facilities and structures, support systems, and utilities,
- Mobilization and de-mobilization,
- Construction consumables,
- Safety training, safety officers and inspections, other medical and safety requirements,
- QA/QC,
- Contractor margin, contractor supervision and staff support.

EPCM costs are incurred past project Feasibility to produce engineered construction drawings, hire a general contractor, and procure all equipment and facilities. Additionally, a construction management firm or general contractor will manage and oversee all construction activities and contractors. EPCM costs are estimated at 10% of direct construction capital costs based on project complexity.

A spare parts allowance of 2% of capital for mining fleet, processing equipment, and the surface mobile fleet is estimated. This cost allows for a critical inventory of spare parts to be available during commissioning and start-up.

An initial fill allowance of 2% of capital for mining fleet, processing equipment, and the surface mobile fleet is estimated.

A freight allowance of 8% of direct capital costs is estimated for all equipment and material. This factor is applied only to the equipment or material costs, not labour.

An allowance is included to cover engineering and operational costs for testing and startup of equipment.

Vendor representatives and engineers are on-site for commissioning, startup, and initial training of operators and maintenance personnel. An allowance based on estimated man-days (50 days with 8 representatives) and costs ($1,500/day) is included.

An allowance for estimated owner's costs is included and covers all overhead costs incurred during the construction period.

21.1.10 Contingency

Project contingency is applied to cover undefined or unknown scopes of work within the scope of the estimate. Contingency does not cover uncontrolled risk factors such as labour disputes, force majeure, currency fluctuations or escalation. It does cover areas of unknown cost factors due to lack of data and engineering. Therefore, at lower levels of engineering, contingency percentages are generally higher.

A 20% contingency is applied to the direct and indirect capital cost estimates.

21.1.11 Reclamation Costs

An allowance of $5,000,000 is included for reclamation activities at the end of mine life.

Reclamation of all the disturbed areas at the New Polaris mine site will require re-vegetation to achieve previous land uses on a property average basis. It is expected that reclamation can be achieved at the site with topsoil salvage and replacement. Costs of re-sloping of the CDF are included in sustaining capital and G&A operating costs.

Underground working will be abandoned at the end of mine life.

Upon closure, all buildings and infrastructure will be removed, and the all disturbed areas fully reclaimed. Profits from sales of used and no longer need equipment and supplies will assist to offset reclamation costs.

21.2 Operating Cost Estimate

Operating costs are estimated based on a combination of MMTS experience in similar projects and consultation with contractors and equipment suppliers. The estimated operating costs are shown in Table 21-9.

All dollar amounts are expressed in Q1 2019 Canadian dollars, unless specified otherwise.

The accuracy range of the capital cost estimate is +/- 30 %.

Table 21-9 Project Operating Cost Estimate

Area	Operating Estimate	Units
Mining	$65.25	$/t milled
Processing	$62.70	$/t milled
G&A	$37.00	$/t milled
Total	**$164.95**	**$/t milled**

21.2.1 Mine Operating Costs

Mine operations descriptions are included in Section 16 of the Report. Mine operating costs are shown in Table 21-10 below. They are incurred once the mill commences production. They do not include ongoing capital development nor equipment purchase or replacements.

Table 21-10 Mine Operating Cost Estimates

Area	$/t
Mining (Direct)	
Longhole Mining	6.00
Cut and Fill Mining (Conventional)	2.00
Drawpoints	2.50
Development Ore	11.75
Backfill	7.50
Subtotal:	**29.75**
General Mine Services	
Labour	
Technical Services	6.00
Mine Operations and Maintenance Supervision	4.50
General Operating Labour	14.00
Subtotal:	**24.50**
Miscellaneous	
Power	4.50
Equipment	3.00
Supplies	1.25
Hauling & Remucking	2.25
Subtotal:	**11.00**
Total:	**65.25**

21.2.2 Process Operating Costs

Process operations are described in Section 17 of the Report. They include operations of the crushing, grinding, flotation, bio-oxidation, CIL, detoxification, and filtration circuits. Mill feed is 750 tonnes/day, producing 100 tonnes of concentrate per day to the bio-oxidation and CIL circuits. Process operating costs are shown in Table 21-11 below.

Table 21-11 Process Operating Cost Estimate

Area	$/t
Power	26.65
Consumables	2.70
Labour	16.00
Maintenance	2.70
Reagents	14.65
Total:	**62.70**

21.2.3 G&A Costs

An estimate of the general and administration (G&A) operating costs is shown in Table 21-12 below. Descriptions of these items, including assumed surface operations, are included below.

Table 21-12 G&A Cost Estimate

Area	Operating $/year (x1,000,000)	$/t
Camp	$1.9	6.80
Office Costs	$1.6	5.85
Employee Travel	$1.3	4.75
Surface Mobile Fleet	$0.7	2.40
Freight	$1.1	4.05
Surface Staff	$1.1	4.15
Administration Staff	$0.8	3.00
HSE and HR Staff	$0.9	3.15
Power (Site)	$0.8	2.85
Total:	**$10.1**	**37.00**

Camp operating costs are built up in Table 21-13.

Table 21-13 Build-Up of Camp Operating Costs

Employees on Site	85
Charge ($/person-day)	$60
Total ($/day)	$5,100
Cost ($/t)	**$6.80**

Estimated office costs to cover office equipment and supplies, software, training, outside consultants, etc. for the following departments:

- Environmental
- Administration
- Health, Safety and Environment
- Human Resources
- IT
- External Technical Consultants
- Insurance
- Legal
- Accounting
- Regulatory and Audit
- Janitorial

Employee travel costs are based on 4 crews operating a 4 week on / 4 week off rotation, with 25% of the employees on local flights and 75% on provincial flights.

Surface Mobile Fleet costs cover the operations of the plant and surface mobile equipment fleet, as described in Section 18.3. This includes the operating cost of mobile equipment hauling, placing and compacting filtered tailings on the CDF, as well as re-sloping the CDF for reclamation.

Freight costs are based on an estimation of 5,500 tonnes of inbound freight per year, at a cost of $200/tonne of freight. Freight consists of trucking materials and equipment to Prince Rupert Port where low draft barges pick up the materials and equipment at the mouth of the Taku River and deliver to site. Supplies transported by air will come with passenger planes (costed in employee transport). Fuel freight costs are included in the total cost of fuel (surcharge of $0.04/L).

Surface, Administration, HSE and HR staff costs include fully burdened salaries for the following functions:

- General Manager
- Accounts Payable and Receivable
- Purchasing and Warehouse Supervision
- Warehouse Technicians
- Surface Mobile Equipment Operators
- Power Plant Operators / Mechanics
- Carpenters
- General Site Labourers
- Health and Safety Manager
- Environmental Technicians
- First Aid
- Nursing
- Security

Power Requirements are described in Section 18.1.9. Power costs for the mine and process plant are included in their respective operating costs described above. Costs for generating power for the site and camp (2.6 MW/year) are included in the G&A operating costs. Costs for power generation are summarized in Table 21-14.

Table 21-14 Power Generation Cost Estimates

Diesel Price ($/Litre)	$1.04
kWh/Litre	3.7
$/kWh, Fuel	$0.28
$/kWh, Maintenance (parts & labour)	$0.015
Total ($/kWh)	**$0.30**

22 Economic Analysis

All dollar amounts in this analysis are expressed in Q1 2019 Canadian dollars, unless specified otherwise.

The economic evaluation of the New Polaris Project includes one year of construction and 8.7-years of mining and milling, as described in Table 16-2. The valuation date on which the Net Present Value (NPV) and Internal Rate of Return (IRR) are measured is the commencement of construction in Year -1. Corporate sunk costs to that point in time, including costs for exploration, technical studies, and permitting, are not included in cash flow; except when determining the project's royalty estimate. The IRR assumes 100% equity financing.

The net profit royalties associated with Rembrandt, as described in Section 4, have been accounted for in the cashflow assuming that 150,000 Canarc shares with be issued to Rembrandt in advance of operations.

A simplified tax regime has been incorporated to estimate federal, provincial and other taxes. The simplified tax regime does not consider the benefits of Capital Cost Allowance, Canadian Development Expense, Canadian Exploration Expense, Cumulative Tax Credit Accounts and Cumulative Expenditures Accounts.

The basis of the project economic analysis is summarized in Table 22-1. Details of the capital and operating cost estimates are described in Section 21. The production schedule used for the economic analysis is described in Section 16.

Table 22-1 Inputs for Economic Analysis

Parameter	Value	Units
Gold Price	$1,300	US$/oz
Currency Exchange Rate	1.30	US$:C$
Gold Payable from refinery	99.9%	
Gold Offsite Costs*	$7	$/oz
Royalty**	10%	
Gold Process Recovery	90.5%	
Mining Cost***	$65.25	$/t milled
Processing Costs	$62.70	$/t milled
General & Administration Costs	$37.00	$/t milled
Federal income tax rate	15%	
BC Provincial income tax rate	11%	

**Offsite costs cover refining charges, doré transport, and insurance costs.*
***It is anticipated that Rembrandt NSR obligations, described in Section 4, will be lowered to 10% in advance of commercial production.*
****Variable annual mining costs based on scheduled underground production, LOM average of $65.25/t.*

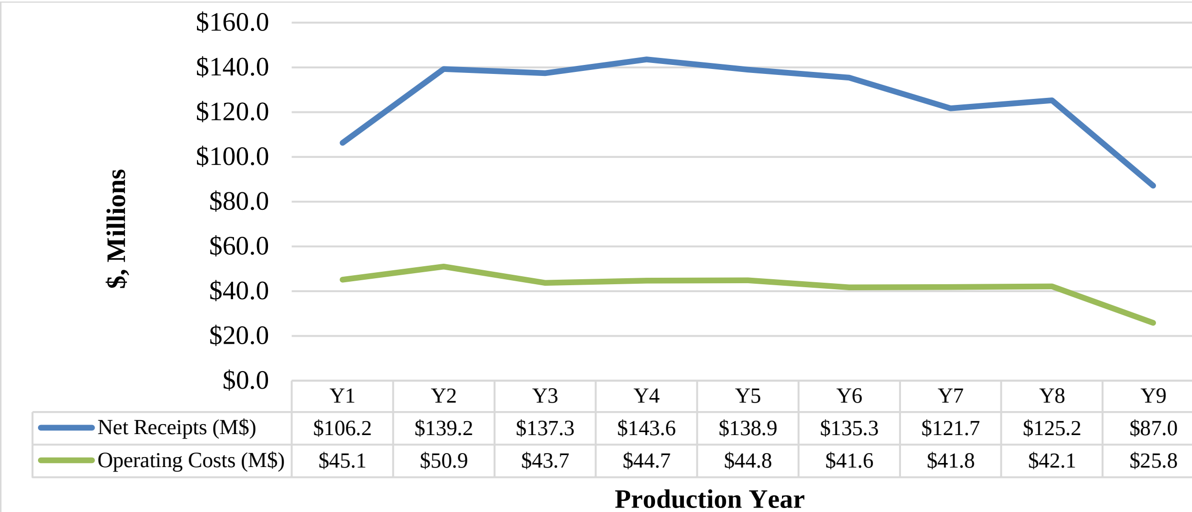
The preliminary economic assessment is based on resources, not reserves. Resources are considered too speculative geologically to have economic considerations applied to them, so the project does not yet have proven economic viability.

Table 22-2 below summarizes the results of the economic analysis for the Project, both the Pre-Tax and Post-Tax results are shown. The following graph, Figure 22-1, shows by year:

- the estimated net gold receipts
  - gross gold receipts minus offsite charges: refining, transport, insurance and royalty charges
- the estimated operating costs
  - mining, processing, G&A costs

Table 22-2 Summary of Economic Analysis

	Value	Units
Mill Feed	2,306,000	t
Au Grade	10.3	g/t
Au Produced	693,000	oz.
Initial Capital	144	$M
Sustaining Capital	56	$M
Cash Cost	433	US$/oz
AISC	510	US$/oz
Net Cash Flow	554	$M
Pre-Tax		
NPV, 5%	385	$M
IRR	47%	%
Payback	2.3	Years
Post-Tax		
NPV, 5%	280	$M
IRR	38%	%
Payback	2.7	Years



	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Net Receipts (M$)	$106.2	$139.2	$137.3	$143.6	$138.9	$135.3	$121.7	$125.2	$87.0
Operating Costs (M$)	$45.1	$50.9	$43.7	$44.7	$44.8	$41.6	$41.8	$42.1	$25.8

Figure 22-1 Net Receipts vs. Operating Costs

22.1 Economic Sensitivity

Figure 22-2 and Figure 22-3 below show the economic result sensitivities to:

- Gold Price
- Foreign Exchange Rate
- Project Capital Costs and
- Operating Costs (mining, processing, and G&A costs)

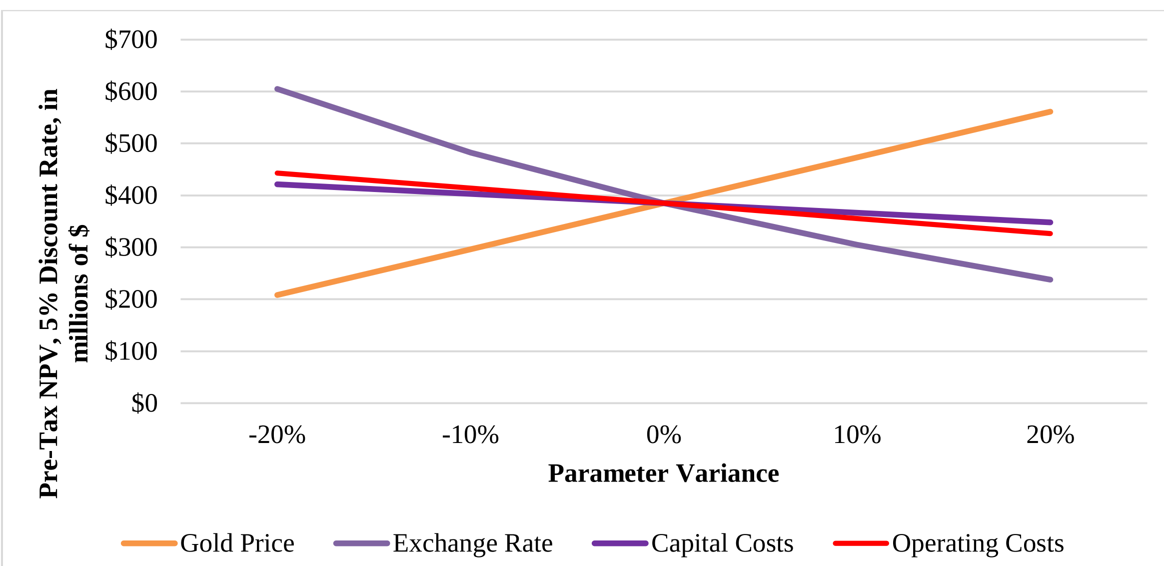


Figure 22-2 Pre-Tax Economic Sensitivity, NPV 5%

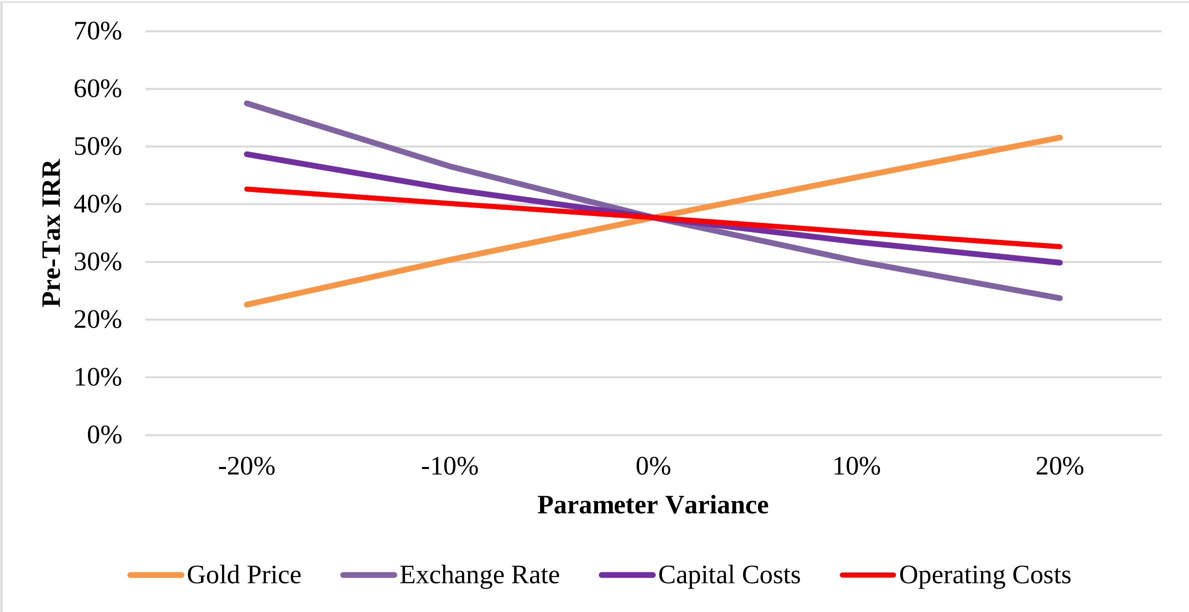


Figure 22-3 Pre-Tax Economic Sensitivity, IRR

The Project is most sensitive to fluctuations in gold price and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs. Sensitivity to metal grades is reflected by the sensitivity to price assumptions.

22.2 Cautionary Statement

The results of the economic analyses discussed in this section represent forward looking information as defined under Canadian securities law. The results depend on inputs that are subject to several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates;
- Assumed commodity prices and exchange rates;
- Mine production plans;
- Projected recovery rates;
- Sustaining and operating cost estimates;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade, or recovery rates;

- Geotechnical and hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment, or processes to operate as anticipated;
- Accidents, labour disputes and other risks of the mining industry.

23 Adjacent Properties

There are no adjacent properties that impact the results of this PEA.

The Tulsequah project is less than five kilometers north of the New Polaris project, up the Tulsequah River, while the Big Bull deposit is approximately six kilometers to the southeast.

24 Other Relevant Data and Information

No additional relevant information or data to disclose.

25 Interpretation and Conclusions

A PEA has been conducted based on a 750 tonne per day underground mine and surface mill plan using an updated Mineral Resource Estimate for the New Polaris Project. It is the opinion of the QP's that the PEA shows positive economic viability, and that the Project warrants further study.

25.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

- The mineral tenure held is valid and sufficient to support the Mineral Resources.
- Surface rights will be required from the Crown before operations.
- Royalties are payable to third parties.
- There are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property that have not been discussed in this report.

25.2 Geology, Mineralization, Exploration

- The deposit is considered to be an example of Mesothermal lode-gold mineralization.
- Knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization is sufficient to support the Mineral Resource estimation.
- The quantity and quality of the lithological, collar and downhole survey data collected in the drill programs are sufficient to support Mineral Resource estimation.
- Canarc has been drilling on the Property since 1988. To date total drilling totals about 106,000 m in more than 1,000 drillholes.
- The sample security, sample preparation and analytical procedures during the exploration programs by Canarc followed accepted industry practice appropriate for the stage of mineral exploration undertaken.
- Data verification has been conducted by Canarc, and no material issues have been identified by those programs.
- Data collected have been sufficiently verified for post 1988 drilling that these drillholes can support Mineral Resource estimation and can be used for mine planning purposes.

25.3 Mineral Resource

- The Mineral Resource has been updated for the "C" and "Y" veins of the New Polaris deposit.
- In total, 286 core drillholes (73,898 m) from 1988 to 2006 inclusive have been used to determine the Resource Estimate. Prior data could not be verified and was not used.
- The updated resource further accounts for limitation in QA/QC by down-grading the Classification in the "Y" veins to Inferred.
- The updated resource accounts for "reasonable prospects of eventual economic extraction" by removing any portions of the mineralization that have a true thickness of less than two metres, and by removing discontinuous mineralization that is not of sufficient volume to have potential underground mining at the current state of knowledge of the deposit.

- The updated resource also differs from the previous resource estimate since all the material within the potential mining shape is reported rather than only material above a cutoff grade. This method of reporting is more in keeping with underground mining and expected recoverable grades and tonnage.
- The base case Mineral Resource is confined by an underground shape that targets material above a 4.0 g/t Au cutoff grade, which provides suitable economics for a PEA study.
- All material within the potential mining shape is reported rather than only material above a cutoff grade. This method of reporting represents the planned underground mining methods and expected recoverable grades and tonnage.
- The base case Mineral Resource contains 1.7 Mt of 10.8 g/t Au in the Indicated category; 1.5 Mt of 10.2 g/t Au in the Inferred category.

25.4 Mining

- The mine plan supports the cash flow model and financials developed for the PEA.
- A subset of the Mineral Resources, utilizing a confining shape targeting material above a 6.0 g/t gold cutoff grade, has been used to develop a 750 tpd underground mine plan feeding a surface mill over an 8.7-year period.
- 6.0 g/t gold grade covers the mining, process, and G&A costs, the total project capital costs spread over the production tonnes and provides some margin to future changes to prices and costs.
- The total delineated resource included in the mine plan is 2.3 Mt at an average gold grade of 10.3 g/t.
- The mine plan uses a combination of conventional cut and fill mining on 24% of the deposit and longhole stoping on 58% of the deposit, depending on mineralization thickness and continuity. Development in ore makes up the remaining 18% of production.
- Development will include a decline from surface, extraction drifts on sublevels across the footwall of the orebody, and ventilation raises to the surface.
- The planned underground unit operations are proven to be effective in similar underground mine operations. The level sequencing and mine design provide a reasonable basis for the production schedule with adequate operating rates to meet the targeted mill feed rate of 750 tpd.

25.5 Metallurgy and Process

- The completion of a several metallurgical test work campaigns have developed a process flowsheet for the economic extraction of gold from New Polaris ore.
- Gold is associated with arsenopyrite and is refractory.
- An onsite process plant will consist of crushing, grinding and flotation to produce a flotation concentrate.

- The flotation concentrate will be treated using bio-oxidation and a CIL plant, followed by carbon stripping, Electrowinning and refining steps to produce doré gold bars.
- Leach tails will be detoxified using the ASTER™ and SO_2/Air treatment processes before being pumped to the final tailings thickener. Test work has demonstrated the successful destruction of cyanide and long-term stability of arsenic in leach tails.
- An overall gold recovery of 90.5% is projected.
- Process tails will be thickened with 42% pumped underground to a paste plant and distribution system into mined out voids and the remaining 58% filtered and hauled to a co-disposal facility (CDF) along with mined waste rock from underground development.

25.6 Environmental, Permitting and Social Impact

- It is not unreasonable in this jurisdiction to expect the New Polaris Project will be approved by regulators, with the inclusion of acceptable Management and Mitigation Plans for potential impacts.
- It is also not unreasonable that the Project will enjoy community support with the inclusion of acceptable Management plans and commitments from the Proponent.

25.7 Infrastructure and Costing

- Capital and Operating cost estimates include the cost to purchase, transport, construct and operate onsite infrastructure, equipment, supplies, and personnel to support the operation.
- The project will be fly-in fly-out, with an onsite camp and airstrip. Major supplies shipping will be barged along the Taku River on a seasonal basis.
- The CDF is located 2.5 km north of the deposit and has a design capacity of approximately 1.3 Mm^3 of tails and waste rock. The CDF will not store water.
- Initial capital cost for construction of the Project are estimated to be $144 million. Sustaining capital requirements over the 8.7-year mine life are estimated to be $56 million.
- Capital cost estimates have been benchmarked to similar construction projects in Western Canada and are reasonable.
- Total operating costs for the Project are estimated at a unit cost $165/t ore. Mining, process, G&A and surface operating cost estimates are based on operating statistics from similar operations utilizing similar infrastructure and equipment fleets. The resultant operating costs are reasonable.

25.8 Costs and Economic Analysis

- Economics are based on a market gold price of US$1,300/oz. and a 1.3 C$:US$ exchange rate.
- The LOM gold production is estimated at 693 koz.
- The Post-tax NPV5% is $280 million, and Post-tax IRR is 38%. The projected capital payback is 2.7 years.
- The all-in sustaining cost is US$510/oz. Au.

- The project is most sensitive to fluctuations in gold price and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs.

26 Recommendations

The PEA indicates that the New Polaris Project has positive results and therefore further work is recommended to gather additional site baseline data, optimize the project, and complete a Preliminary Feasibility Study.

The following programs and studies listed in Table 26-1 are directly recommended following this PEA, to lead to a decision point on whether to complete a PFS. A more detailed scope of work will need to be developed when the programs listed below are completed.

Table 26-1 Recommended Programs for Next Level of Study

Item	Description	Estimated Budget (M$)
1	Diamond Drilling Program	$4.5
2	Updated Geological Interpretation and Resource Estimate	$0.2
3	Geotechnical Drilling and Analysis	$1.0
4	Mining Studies	$0.1
5	Metallurgical Studies and Process Refinement	$0.5
6	Site Plan Refinement	$0.2
7	CDF Specific Site Investigations	$1.0
8	Water Quality Testing	$0.6
9	Hydrology	$0.3
10	Hydrogeology	$0.5
11	Fish and Fish Habitat Studies	$0.1
12	Air, Noise and Climate Studies	$0.2
13	Vegetation Studies	$0.1
14	Wildlife Studies	$0.1
15	Soil Quality Testing	$0.2
16	Terrain and Seismic Studies	$0.2
17	Archaeology	$0.1
18	First Nations and Community Engagement	$0.2
19	Traditional Knowledge/Use Study	$0.1

Note: Although care has been taken in the preparation of these estimates, the authors do not guarantee that the above described programs can be completed for the estimated costs. Additional quotes and budgeting should be done when financing is in place prior to the start of the program, when quotes can be obtained for supplies and services.

The estimated dollar amounts for these items are not included in the Project Capital Estimate or Economic Analysis conducted for this PEA.

The following recommendations are intended for consideration in the test work above and for the eventual PFS work to follow.

26.1 Resource Recommendations

- Additional geological interpretation, drilling, and QA/QC is recommended.
- Further investigation should include a program to laterally extend known mineralization and to test the down dip extension of mineralization.
- An infill drilling program to increase the confidence and Classification of mineralization. The program should include additional QA/QC and potentially twinning of existing holes prior to 2006 to further validate older drill data.
- It is recommended to diamond drill 20,000 m to the 450 m level for both the expansion and infill goals listed above.
- A suitable program of specific gravity determination should be undertaken to increase the number of measurements in the current areas of expected mining – to be done on existing core and any new drilling on the property.
- Update the geologic interpretation and Resource estimate based on the new drilling and to aid in further drill targets.
- Items 1 and 2 in Table 26-1.

26.2 Mining Recommendations

- A geotechnical study, including core logging, should be carried out for the proposed underground development and production mining areas.
- A mining method trade-off study should be carried out to confirm which mining methods work best for the deposit. Along with that an analysis of the optimum mining rate should be carried out.
- If sublevel stoping is to be used, a detailed analysis of sublevel intervals should be carried out.
- UCS testing should be carried out on the potential paste backfill tails.
- Items 3 and 4 in Table 26-1.

26.3 Process Recommendations

- Additional metallurgical test work using samples from new drill core to finalize the process flowsheet, develop recovery projections, mass balance, and design assumptions.
- Complete preliminary process engineering and plant design.
- Examine higher mill throughput potential.
- Item 5 in Table 26-1.

26.4 Infrastructure Recommendations

- Optimize the site general arrangement.
- Initiate geotechnical site investigations to identify suitable borrow sources for construction materials.
- Initiate engineering studies for the water balance, water quality and water management on site, including hydrogeology investigations into the areas proposed for underground operations.
- A surficial geology study and geotechnical site investigation in the CDF area, including ground truthing and laboratory testing, to refine the assumptions made for the PEA cost estimate.
- Items 6 to 10 in Table 26-1.

26.5 Environmental, Permitting and Social Impact

- Update environmental background/baseline studies for aquatic, terrestrial components as well as studies for wildlife, groundwater, geochemical, archeology, seismic, and related environmental issues.
- Further develop an understanding of geochemistry characteristics with respect to waste rock and tailings; begin including analysis and predictions involving bio-leach processes.
- Sampling of site water and the existing tailings to determine if there is any acid water or contaminants draining out. As well, sampling of water downstream from the site to determine if drainage from the existing tailings and waste rock is affecting the water quality of Whitewater Creek or the Tulsequah River. A mitigation plan will be required ahead of any further exploration or development work if there is any contamination of the downstream water. The cost of the mitigation will depend upon the level of contamination and has not been included in Table 26-1.
- Commission Traditional Knowledge Studies in consultation with the TRTFN.
- Develop and implement a Consultation and Implementation Plan with the TRTFN.
- Commission Social and Community Studies and develop Consultation and Engagement plans with affected stakeholders.
- Items 11 to 19 in Table 26-1.

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