THE NEW POLARIS GOLD PROJECT, BRITISH COLUMBIA, CANADA 2023 RESOURCE ESTIMATE UPDATE

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

Effective Date: April 20, 2023
Issue Date: June 30, 2023

Moose Mountain Technical Services
Sue Bird, P.Eng.

#210 1510-2nd St. North
Cranbrook, B.C. V1C 3L2 Canada
Tel: 250.489.1212
Email: sueb@moosemmc.com

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

CERTIFICATE OF QUALIFIED PERSON

I, Sue Bird, P.Eng. am a Geological and 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 2023 Resource Estimate Update"** with an effective date of April 20, 2023 (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 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 due diligence 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 Project on August 25, 2022.

I am responsible for Sections 1.1 through 1.9, 1.11, Sections 2 through 12 and Sections 14 through 24 and Sections 25.1, 25.2, 25.3, 25.5, 26.1, and 26.2.

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

I previously co-authored as QP for Section 14 the 2019 report entitled: "The New Polaris Gold Project, British Columbia, Canada 2019 Preliminary Economic Assessment".

I have read NI43–101 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: 30 June 2023

"Signed and sealed"

____

Sue Bird, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Deepak Malhotra, Ph.D., SME-RM am the Director of Metallurgy with Forte Dynamics, Inc., with an office at 12600 W Colfax Ave., Suite A-540, Lakewood, Colorado 80215.

This certificate applies to the technical report titled "**The New Polaris Gold Project, British Columbia, Canada 2023 Resource Estimate Update"** with an effective date of April 20, 2023 (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 graduated with a Metallurgical Engineering degree (M.Sc.) in 1973 and a Mineral Economics Ph.D. in 1978 from the Colorado School of Mines in Colorado, USA. I am a registered member in a good standing of the Association of Society of Mining and Metallurgical Engineers (SME-RM #2006420) and a member of the Canadian Institute of Mining and Metallurgy (CIM).

I have worked as a metallurgist and mineral processing expert with over 50 years of experience in the mining industry. I have worked on precious metals, base metals and coal mining projects, including processing audits and due diligence 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 Project site.

I am responsible for Sections 1.10, 13, 25.4 and 26.1.

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

I have previously undertaken metallurgical test work for the New Polaris Project.

I have read NI43–101 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: 30 June, 2023

"Signed and sealed"

____

Deepak Malhotra, Ph.D., SME-RM

1 Summary

1.1 Introduction

Moose Mountain Technical Services (MMTS) has prepared a NI43-101 Technical Report (the Report) on the New Polaris Gold Project (the Project) for Canagold Resource Corp (Canagold). The New Polaris Gold Project is in northwestern British Columbia (BC), about 100 km south of Atlin, BC, and 60 km northeast of Juneau, Alaska.

Sue Bird, P.Eng is the QP for Sections 1.1 through 1.9, 1.11, Sections 2 through 12 and Sections 14 through 24 and Sections 25.1, 25.2, 25.3, 25.5, 26.1, and 26.2.

Deepak Malhotra is the QP for Sections 1.10, 13, 25.4 and 26.1.

1.2 Terms of Reference

The Report has been prepared in support of disclosures in Canagold's news release dated May 16th, 2023, entitled "Canagold Increases Indicated resource by 89% in Updated Mineral resource Estimate for new Polaris Gold project, BC".

A Mineral Resource estimate and a Preliminary Economic Assessment (PEA) on the Project was 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 was next done in 2019. An updated Mineral Resource estimate has been completed in 2023 which supersedes the previous studies and has been summarized into this report.

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

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 (**Error! Reference source not found.**). 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 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.

1.4 Mineral Tenure, Royalties and Agreements

The property consists of 61 contiguous Crown-granted mineral claims and one modified grid claim covering 1108 ha. All claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canagold Resource Corp. subject to a 15% net profit interest held by Rembrandt Gold Mines

Ltd. Canagold can reduce this net profit interest to a 10% net profit by issuing 150,000 shares to Rembrandt.

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 (now Canagold) explored the “C” vein system between 1988 and 1997 and carried out infill drilling in 2003 through 2006 and in 2021 and 2022 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 Quality Assurance and Quality Control (QA/QC) program is similar for the above-mentioned programs in that samples were collected by employees of Canagold 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 Canagold 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 2023. The updated resource estimate uses all available drillhole data with historical data compared to and validated with recent drilling. The resource is based on 1,692 assay intercepts from 234 drill holes which intersect the veins within the data set. Inverse distance squared (ID2) has been used to interpolate the gold grade of the veins which were modelled by Moose Mountain Technical Services (MMTS) using Implicit modelling.

The geologic continuity of the "C" vein system has been well established through historic mining and diamond drilling. Grade continuity has been quantified using semi-variograms, which are used to determine the distances (ranges) and directions of maximum continuity in the three principle directions. The ranges are used for Classification.

The classification to Indicated or Inferred required that the true thickness of the vein is at least 2 m. Classification is based primarily on anisotropic distances to drillholes with 50m grid drill spacing being targeted. However, additional adjustments have been made to ensure a cohesive shape of Indicated material is produced, as summarized in Section 14.

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 employed within each confining shape to report and underground Resource 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 generated which targeted material above this grade is used to define the "reasonable prospects of eventual economic extraction" for the Mineral Resource Estimate.

The effective date of this Resource estimate is April 20th, 2023.

Table 1-1 Updated Mineral Resource Estimate and Comparison to the 2019 Resource

Class	Cutoff (Au gpt)	2023 Resource Tonnage (ktonnes)	2023 Resource Au (gpt)	2023 Resource Au (koz)	2019 Resource Tonnage (ktonnes)	2019 Resource Au (gpt)	2019 Resource Au (koz)	Difference as a Percent: (2023-2019)/2019 Tonnage	Difference Au Grade	Difference Au Metal
Indicated	**3**	3,118	11.21	1,124	1,798	10.40	601	73%	8%	87%
	4	2,965	11.61	1,107	1,687	10.80	586	76%	8%	89%
	5	2,769	12.11	1,078	1,556	11.30	565	78%	7%	91%
	6	2,525	12.75	1,035	1,403	12.00	541	80%	6%	91%
	7	2,270	13.45	981	1,260	12.60	510	80%	7%	92%
	8	2,049	14.09	928	1,105	13.30	473	85%	6%	96%
	9	1,814	14.81	864	947	14.10	429	92%	5%	101%
	10	1,594	15.55	797	1,639	9.50	501	-3%	64%	59%
Inferred	**3**	1,061	8.24	281	1,582	9.80	498	-33%	-16%	-44%
	4	926	8.93	266	1,483	10.20	486	-38%	-12%	-45%
	5	817	9.52	250	1,351	10.70	465	-40%	-11%	-46%
	6	706	10.16	231	1,223	11.20	440	-42%	-9%	-48%
	7	603	10.78	209	942	12.50	379	-36%	-14%	-45%
	8	491	11.52	182	753	13.80	334	-35%	-17%	-46%
	9	371	12.51	149	653	14.60	307	-43%	-14%	-51%
	10	291	13.33	125	0	0.00	0			

Notes for Mineral Resource Estimate:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
2. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.

3. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines.
4. The base case Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shape using the following assumptions:
    - Metal prices of US$1,750/oz Au and Forex of 0.75 $US:$CDN;
    - Payable metal of 99% Au;
    - Offsite costs (refining, transport and insurance) of US$7/oz;
    - Mining cost of CDN$82.78/t ,
    - Processing costs of CDN$105.00/t and G&A and site costs of CDN$66.00/t.
    - Metallurgical Au recovery of 90.5%;
5. The resulting Net Smelter Return equation is: NSR (CDN$/t)=Au*90.5%*US$74.72g/t;
6. The specific gravity is 2.81 for the entire deposit;
7. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, 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.

1.10 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%.

1.11 Environmental, Permitting and Social Considerations

The Project is located within the land claim and traditional territory of the Taku River Tlingit First Nation (TRTFN). A collaborative agreement has been signed with TRTFN that formalizes the open and meaningful communications with the Project proponent that have 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 policies and guidelines. Additional baseline data collection work required to meet regulatory requirements will be completed during the Feasibility and Permitting stages of the Project. The proponent will continue to engage and consult with 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 British Columbia Environmental Assessments Office (BCEAO) as its production threshold exceeds the requirements for mining activities at a Provincial Level. Currently, it does not meet the trigger (5,000 tonnes per day) for a federal assessment through the Impact Assessment Agency of Canada (IAAC). Additional consultations and input are also 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 to expect approvals will be received conditional upon acceptance of respective Management Plans and commitments to mitigate impacts from operations.

2 Introduction

Canagold Resource Corp. (Canagold) is engaged in the exploration and advancement of the New Polaris Gold Project (the Project) in northwestern British Columbia, Canada. Sue Bird, P.Eng., of Moose Mountain Technical Services (MMTS) has prepared an updated Resource NI43-101 Technical Report (the Report) on the Project for Canagold.

2.1 Terms of Reference

The Report has been prepared in support of disclosures in Canagold's news release dated May 17th, 2023, entitled "Canagold Increases Indicated resource by 89% in Updated Mineral Resource Estimate (MRE) for the New Polaris Gold project, BC". The Mineral Resource Estimate has an effective date of April 20th, 2023.

A previous MRE and a Preliminary Economic Assessment (PEA) Technical Report on the Project was completed in 2007. The PEA Technical Report was updated in 2009, 2011 with the MRE and the PEA updated again in 2019.

The Mineral Resource Estimate has been performed in accordance with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines and reported in accordance with the 2019 CIM Definition Standards for Mineral Resources and Mineral Reserves (2019 CIM Definition Standards).

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

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:

Sue Bird, P. Eng is the QP for Sections 1.1 through 1.9, 1.11, Sections 2 through 12 and Sections 14 through 24 and Sections 25.1, 25.2, 25.3, 25.5, 26.1, and 26.2.

Deepak Malhotra is the QP for Sections 1.10, 13, 25.4 and 26.1.

2.3 Site Visits and Scope of Personal Inspections

Sue Bird of MMTS conducted a site visit of the property on August 25th, 2022. During the site visit, sufficient opportunity was available to examine core logging procedures, drill core from the 2022 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 her experience, qualifications and review of the site and resulting data, the author, Ms. Bird, 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.

3 Reliance on Other Experts

The QPs of this Report, state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person". The QPs have relied on and believe there is a reasonable basis for this reliance, upon the following expert report, which provide information regarding sections of this Report as noted below.

3.1 Mineral Tenure

The QP has not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements. The QP has fully relied upon, and disclaims responsibility for, information supplied by Canagold Corp. and experts retained for Canagold for this report.

This information was provided by Canagold on June 10, 2022, in the following document:

"CCM Mineral Title Opinion Dentons 20221213" by Dentons Canada LLP, dated December 13th, 2022.

This information is used in Section 4 of the Report, and in support of the Mineral Resource estimate in Section 14. The QP has assumed that the information in this letter is accurate and understands that the information in such letter may not be relied upon by any other party without the consent of Dentons Canada LLP.

4 Property Description and Location

The New Polaris property consists of a group of 61 contiguous crown grants, and one modified grid claim totaling, 1108 ha 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, as illustrated in Figure 4-1

The claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canagold Resources Ltd Rts (Canagold), and subject to a 15% net profit interest held by Rembrandt Gold Mines Ltd. (Rembrandt), which Canagold 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, 2024. The crown granted claims were legally surveyed in 1937. The mineralized areas are shown on **Error! Reference source not found.** 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.

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.

Additional exploration work carried out in 2021 to 2022 was covered by:
Mines Act Permit MX-1-208, Approval # 20-0100054-0911

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 freshwater 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. 6	6106	4472
Polaris No. 7	6145	5223		Snow No. 7	3500	4472
Polaris No. 8	6146	5223		Snow No. 8	6107	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	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, 2024.

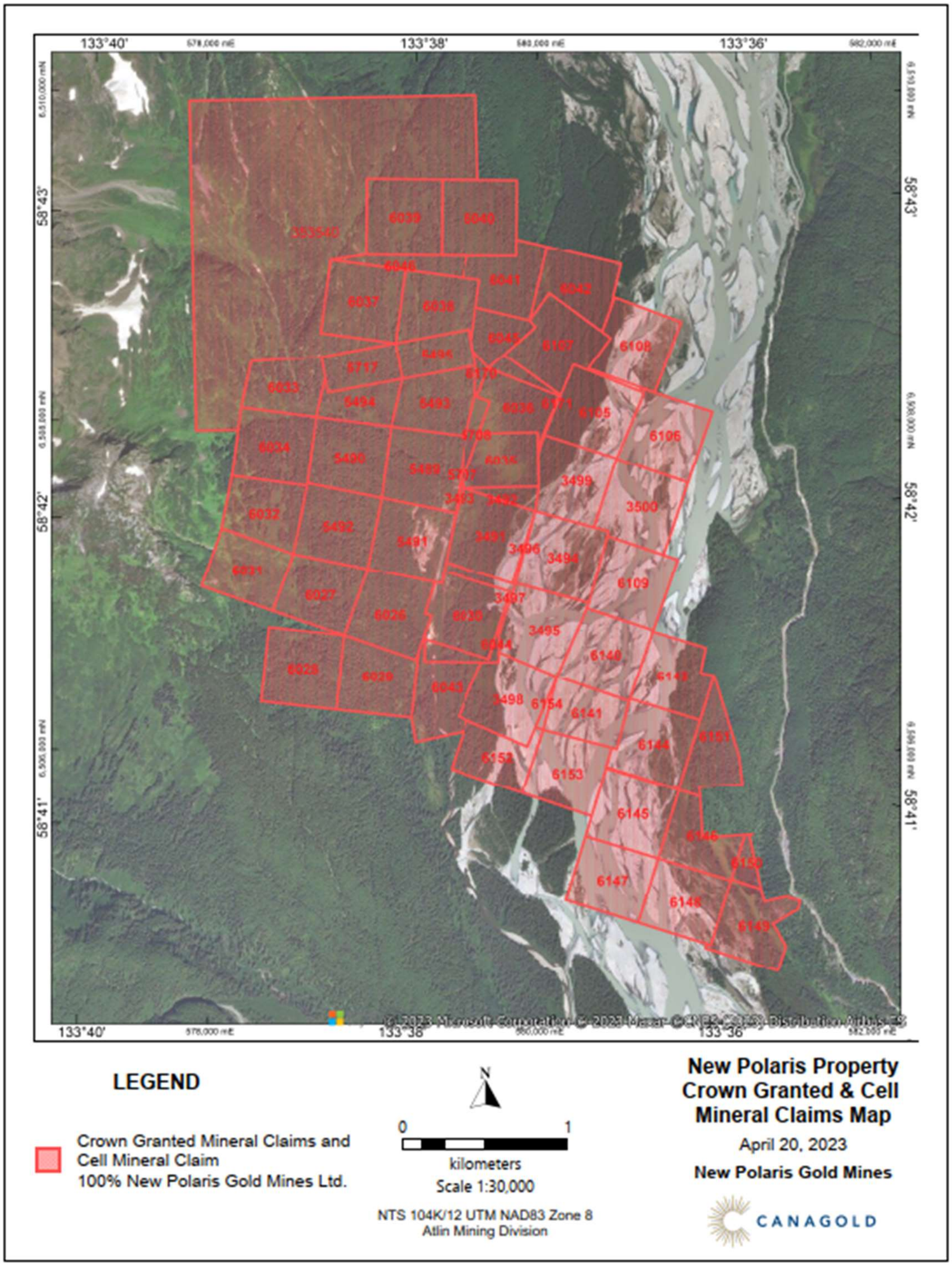


Figure 4-1 Claim Location Map

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 sea level to 2,600 metres.

Extensive glaciation was the dominant factor in topographic development. The Taku and Tulsequah Rivers are the most prominent topographic features with 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 temperate 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 Canagold for installations such as the process plant and related mine infrastructure. Surface rights for the areas covered by the Combined Storage Facility (CSF), and access road to the CSF, lie with the Crown and will need to be obtained from the Province of British Columbia.

6 History

This section has been taken from the previous NI43-101 report done by MMTS (MMTS, 2019).

From 1923 to 1925 the Big Bull and Tulsequah Chief properties were discovered along the east side of the Tulsequah River and opened the Taku River district. In 1930, Noah A. Timmins Corporation optioned some of the claims that make up the New Polaris property and then 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 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 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. Work on the property since 1988 is discussed in later sections of this report.

7 Geological Setting and Mineralization

This section has been taken from the previous NI43-101 report done by MMTS (MMTS, 2019).

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

Canagold 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 anastomosing 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 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.

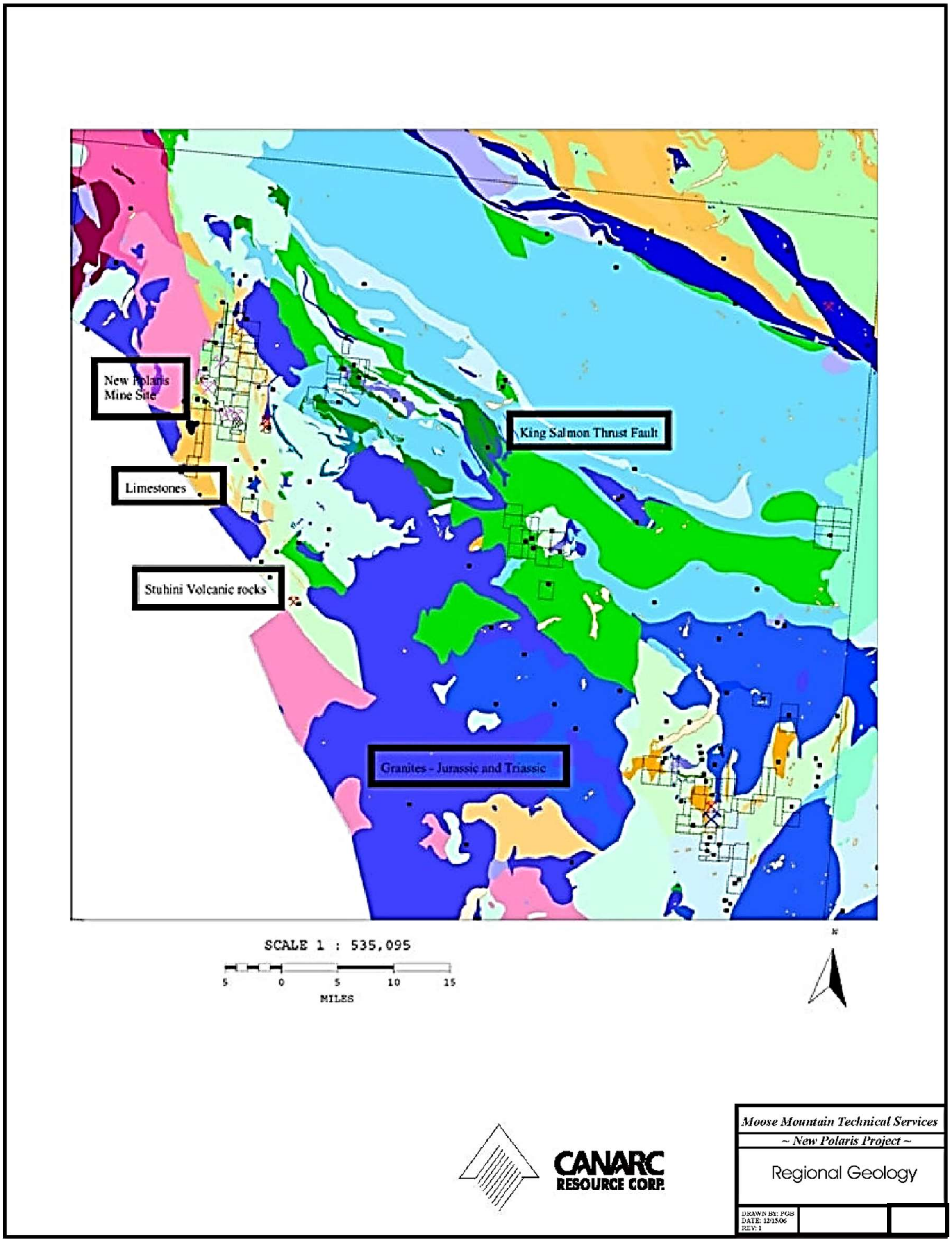


Figure 7-1 Regional Geology

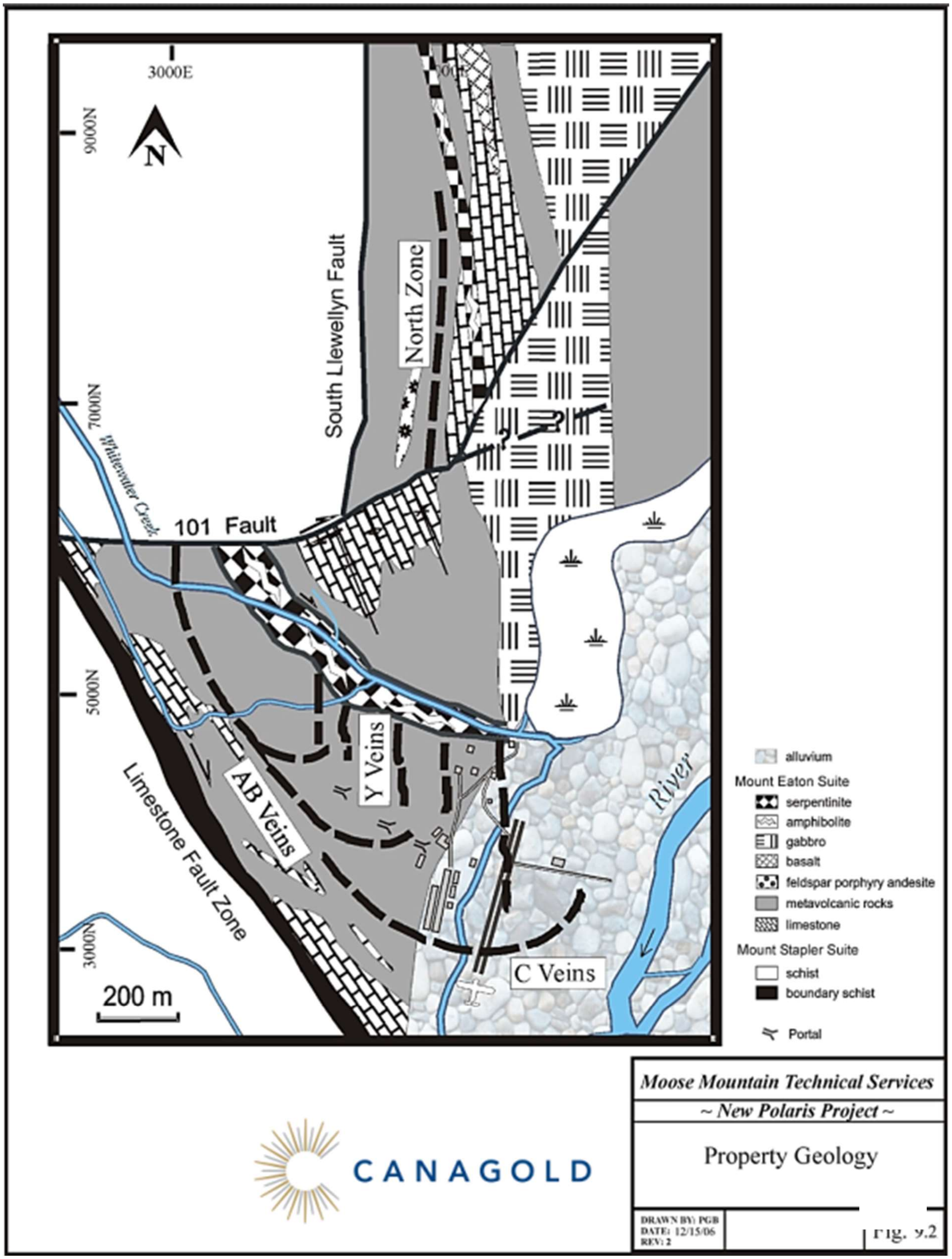


Figure 7-2 Property Geology

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.

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 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.

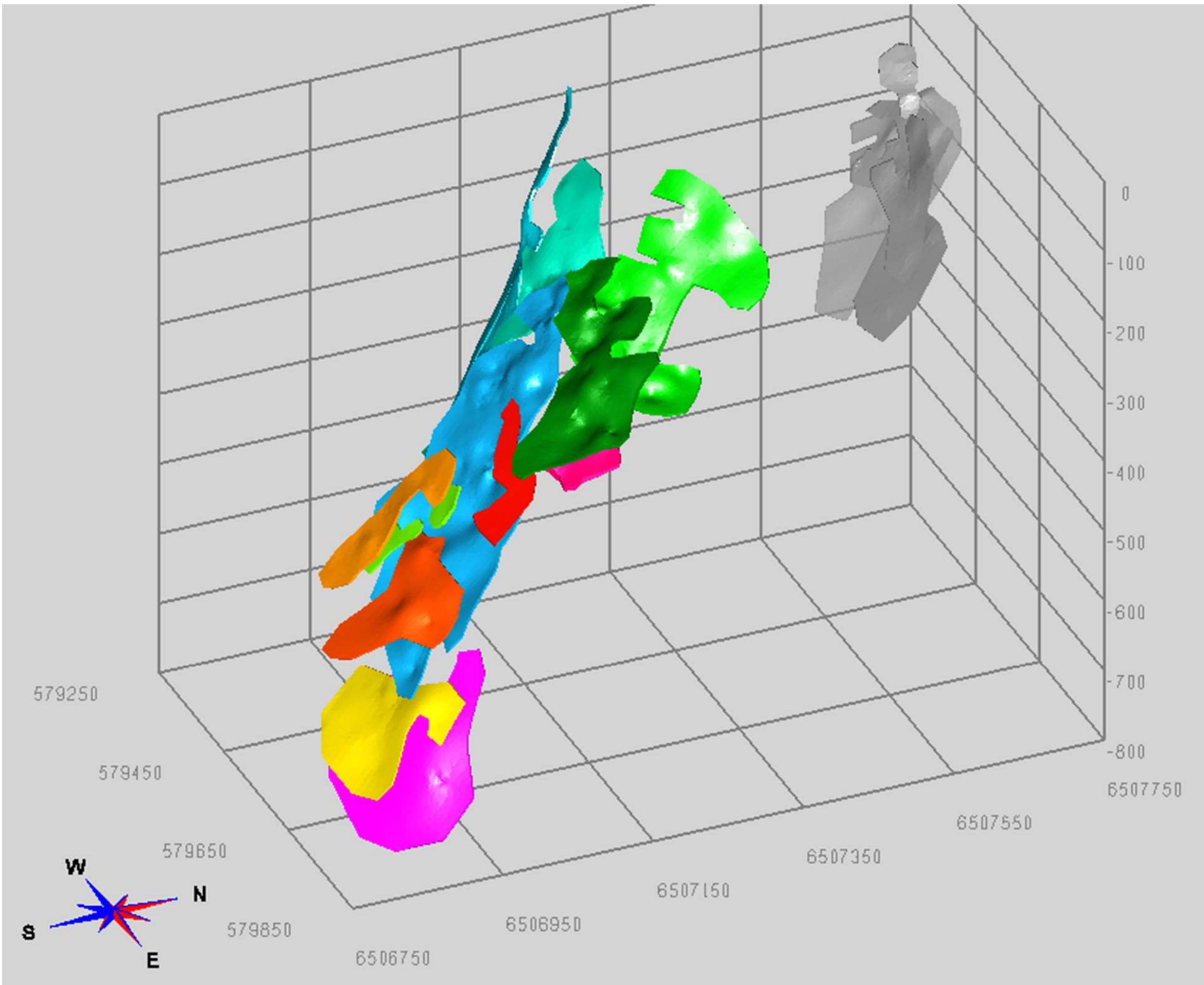
In general, the zones of mineralization ranging from 15 to 250 m in length with widths up to 14 m appear to have been deposited only on the larger and stronger shears. Their walls pinch and swell showing considerable irregularity both vertically and horizontally. Gold values in the veins have remarkable 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.

(Source: MMTS, 2023)

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. Much 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 S. Bird 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.



(Source: MMTS, 2023)

Figure 7-3 3D Vein Model (colour="C" veins, grey = "Y" veins)

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 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 world 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.

Table 9-1 Summary of 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
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	

Surface mapping, geochemistry, and geophysics over the “mine wedge” were completed by Orequest in 1988 and further surface mapping and geochemistry, on the “north grid” were completed in 1993 with results are shown in Figure 7-2.

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.

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.

No additional exploration has occurred on site since 2006, except for drilling discussed in Section 10.

10 Drilling

Diamond drill programs were carried out on the New Polaris Project when the project was reactivated in 1988 until 2022 (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.

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. 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.

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 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 confirmed that the mineralized shoots in the lower most stopes on the Y and C veins were open to depth.

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. Canagold, undertook a review of the Polaris Project and recommended additional drilling 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 out from 2003 to 2006. The 2003 to 2006 exploration programs targeted the “C” vein extensions below the existing mine workings.

In 2021 to 2022 an additional two-phase drilling program was completed. Phase 1 was designed primarily to collect further details on the Inferred Resources of the C vein system, predominately in the West Main domain. The 52 infill drill holes had a target depth ranging between 300 m to 650 m and were designed to provide greater density of drill intercepts at approximately 25 m spacing along section lines. Two drill holes were considered abandoned with no attempt to re-drill. One very deep hole with two additional wedge holes was completed as part of the program to attempt to intersect the down dip projection of the CWM domain. The drilling confirmed the downdip extension of the C veins.

Phase 2 of the 2021 to 2022 drilling program was designed primarily to collect further details on the inferred Resources of the Y Vein system. The 45 infill drill holes had a target depth ranging between 29 m to 749 m and were designed to provide greater density of drill intercepts. One of the drill holes was aimed at intercepting the gap between two of the “C” veins and two holes were intended to infill the C East lenses.

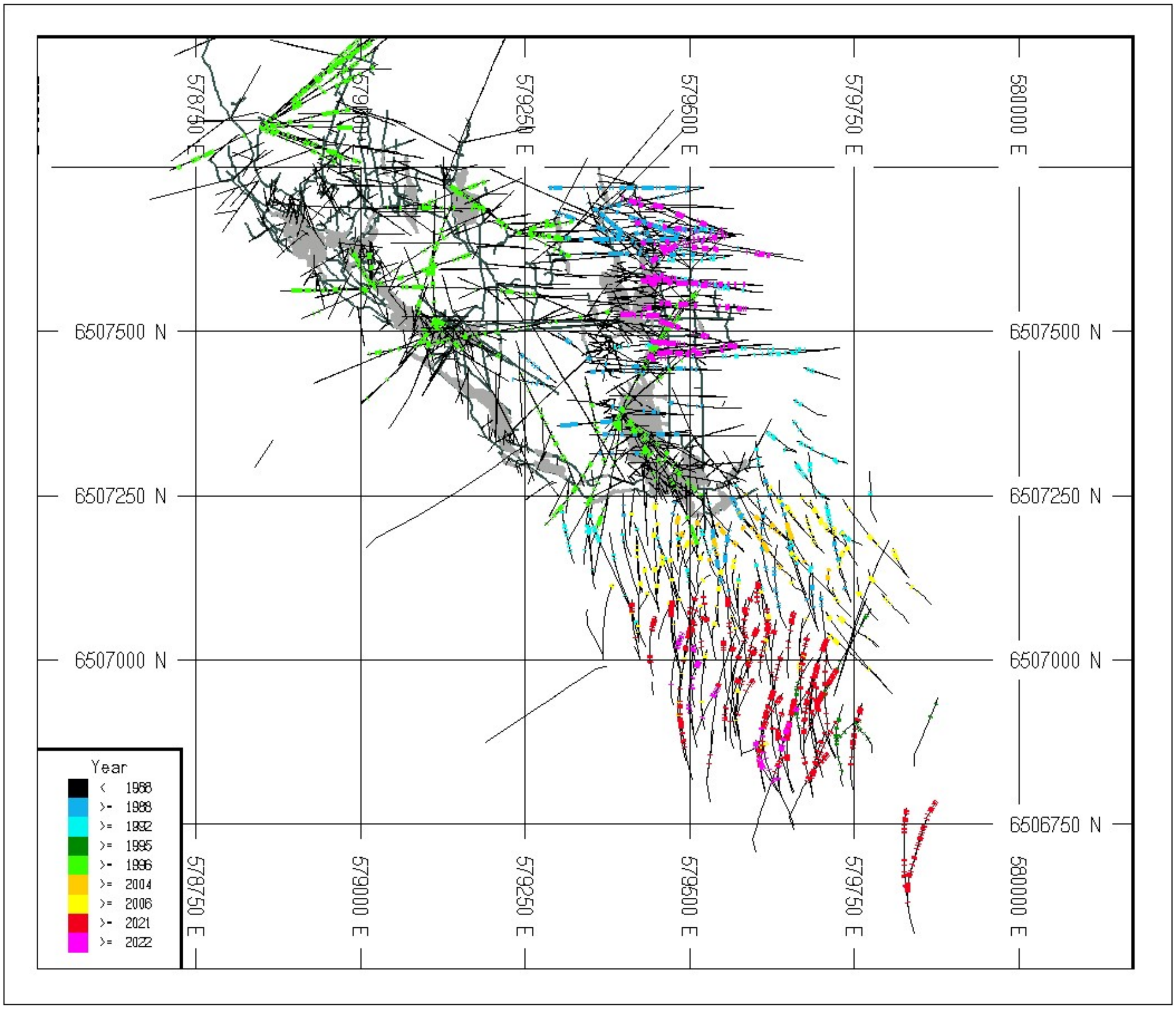
The results of the 2021 to 2022 drilling of the “C” and “Y” vein systems confirmed the continuity of gold mineralization and the vein structure between the earlier drilled holes. As can be seen in the figures below, drill results show the “C” vein system to be an arc-like structure generally oriented east-west with a shift in strike in the west. 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 sigmoid-like feature.

A plan view of the collar locations for the drillhole used for the resource estimate are plotted on Figure 10-2. A representative long section plots showing the pierce points of drillholes, and grades of the relevant drilling is presented in Figure 10-2.

All the holes in this period were drilled from surface and intersected a similar geologic sequence. From the collar, the holes penetrated 15 m to 79 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.

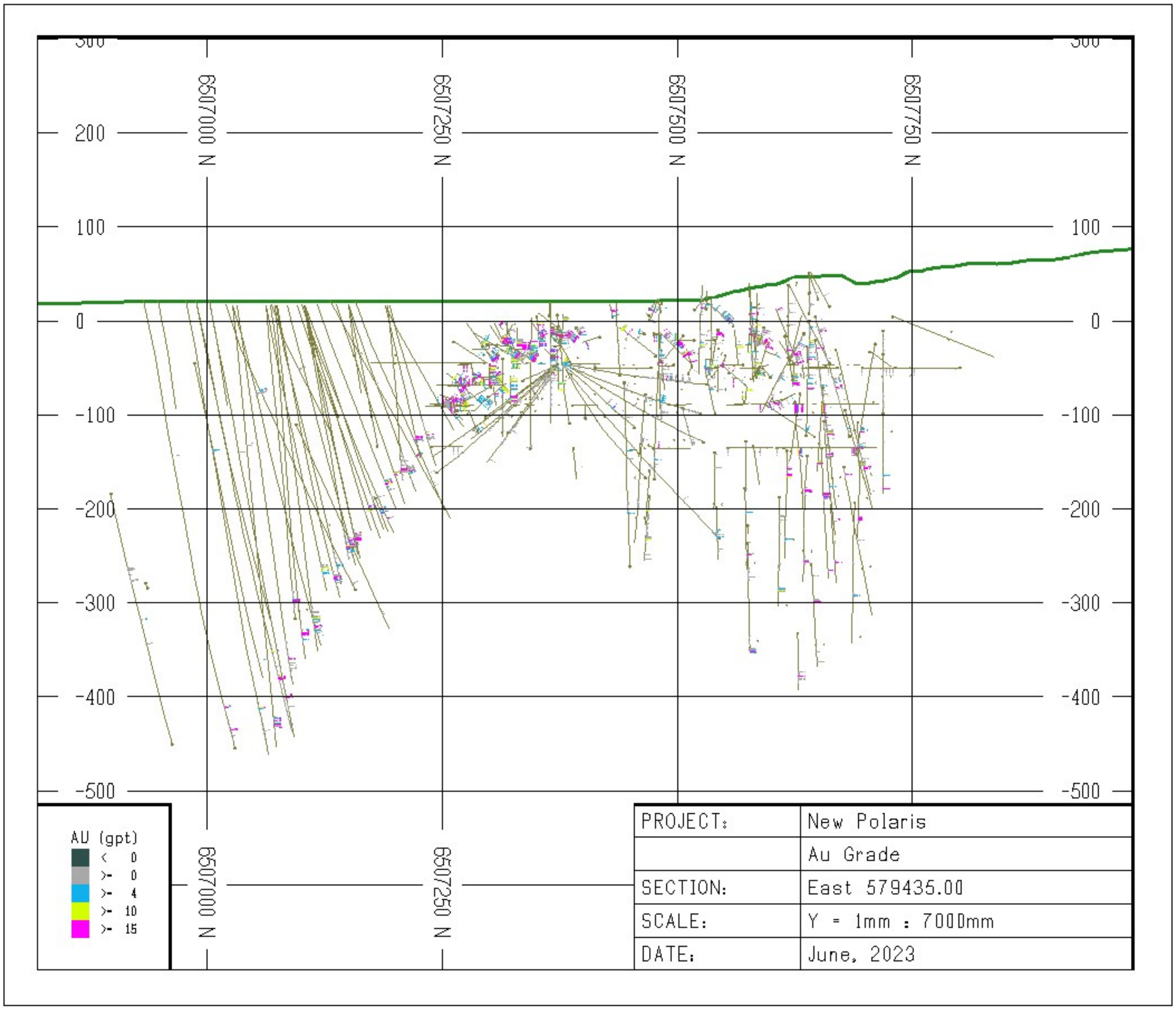
Table 10-1 Diamond Drillhole Summary (1988 to 2022)

Year	Supervisor	Drilling Contractors	Laboratory	Number of Collars	Total Length Drilled (m)
<1988				778	32,483. 05
1988	Cloutier		TSL	8	1,027.79
1989	Cloutier		TSL	21	4,490.75
1990	Cloutier		TSL	9	2,862.46
1991	Marriott	Arctic	Min En/Chemex	11	3,436.77
1992	Marriott	Arctic	Chemex	21	5695.40
1993	Marriott/Moors	Arctic	Chemex	7	1,142.39
1994	Moors	Arctic/Falcon	Chemex	29	5,719.89
1995	Moors	Arctic/Falcon	Chemex	17	7,572.76
1996	Karelse/Watkinson	Advanced	Northern	24	3,204.69
1997	Karelse/Watkinson	Advanced	Northern	43	7,210.84
2003	Moors	Hy-Tech	ALS Chemex	3	1,560.89
2004	Moors/Aspinall	Hy-Tech	ALS Chemex	11	2,766.99
2005	Moors/Aspinall	Hy-Tech	ALS Chemex	9	2,389.63
2006	Moors/Cote	Hy-Tech	ALS Chemex	72	24,801.60
2021	McLaughlin/Dupuis	ITL Diamond Drilling Ltd	ALS Geochemistry	52	25,512.28
2022	Dupuis/Smith	ITL Diamond Drilling Ltd	ALS Geochemistry	45	14,324.40



(Source: MMTS, 2023)

Figure 10-1 Plan view of Drillhole Locations by Year



(Source: MMTS, 2023)

Figure 10-2 Section of the Drillholes and Vein Modelling (+/- 50m)

11 Sample Preparation, Analyses and Security

11.1 Sample Method and Approach

11.1.1 Year 2003 - 2006

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.

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.

11.1.2 Years 2021-2022

During the 2021 to 2022 drilling program the core was geologically logged to identify the gold mineralized zones that were allocated unique sample number tickets and marked for cutting using a purpose-built diamond blade rock saw. Half core samples were collected in labelled bags and the other half remains in the original core box stored on site. Quality control (QC) samples including certified reference material standards, blanks and duplicates were inserted into the sample sequence at intervals of one in ten on a rotating basis to monitor laboratory performance and provide quality assurance (QA) of the assay results. Several sample bags were transported together in rice bags with unique numbered security tags attached and labelled with company and lab contact information to ensure sample security and chain of custody during shipment to the lab.

The samples were submitted to the ALS Geochemistry lab in Whitehorse, YT in 2021 and Yellowknife, NT 2022 for preparation and assaying. The entire sample was crushed to 70% passing -2 millimeters and a 250-gram aliquot is split and pulverized to 85% passing -75 microns. Analysis for gold was by 30-gram fire assay and gravimetric finish. A suite of 30 other elements including arsenic, antimony, sulfur and iron is analyzed by aqua-regia digestion Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES). ALS Canada Ltd. is accredited by the Standards Council of Canada and is an ISO/IEC 9001:2015 and 17025:2017 certified analytical laboratory in North America.

The procedures for sample preparation, analysis and security procedures follow accepted engineering standards and the quality of gold analytical data collected by Canagold is sufficiently reliable to support Mineral Resource estimation.

11.2 QAQC – 2003-2006

11.2.1 Blanks – 2003-2006

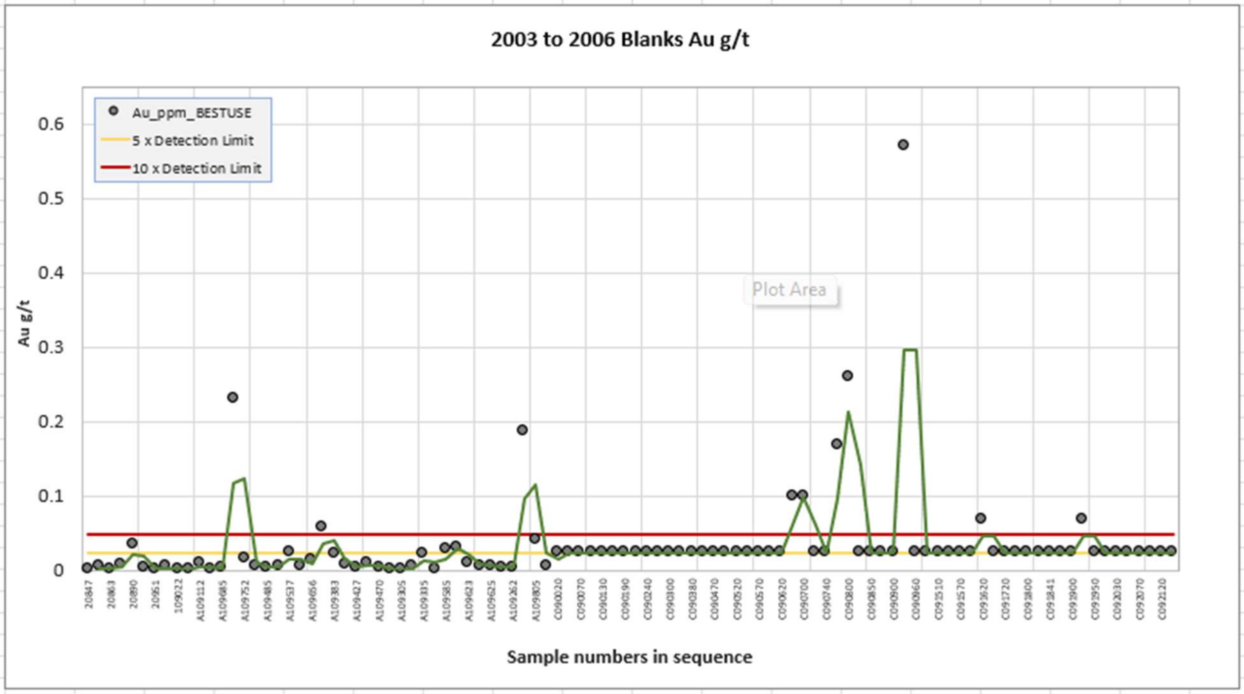
For the QA/QC program in these years, the samples were collected by employees of Canagold on site and shipped to ALS Chemex laboratory in Vancouver. For quality control and quality assurance, core samples were regularly mixed with blanks, duplicates, and standards. The program in the field was run in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

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	98
Minimum value	0.005
Maximum value	0.57
Mean value	0.035
Standard Deviation	0.07
CV	1.94

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). (Source: MMTS, 2023) Figure 11-1 shows the test results of the blank samples (excluding the highest value sample).



(Source: MMTS, 2023)

Figure 11-1 Blank Samples, 2006

11.2.2 Duplicates – 2003-2006

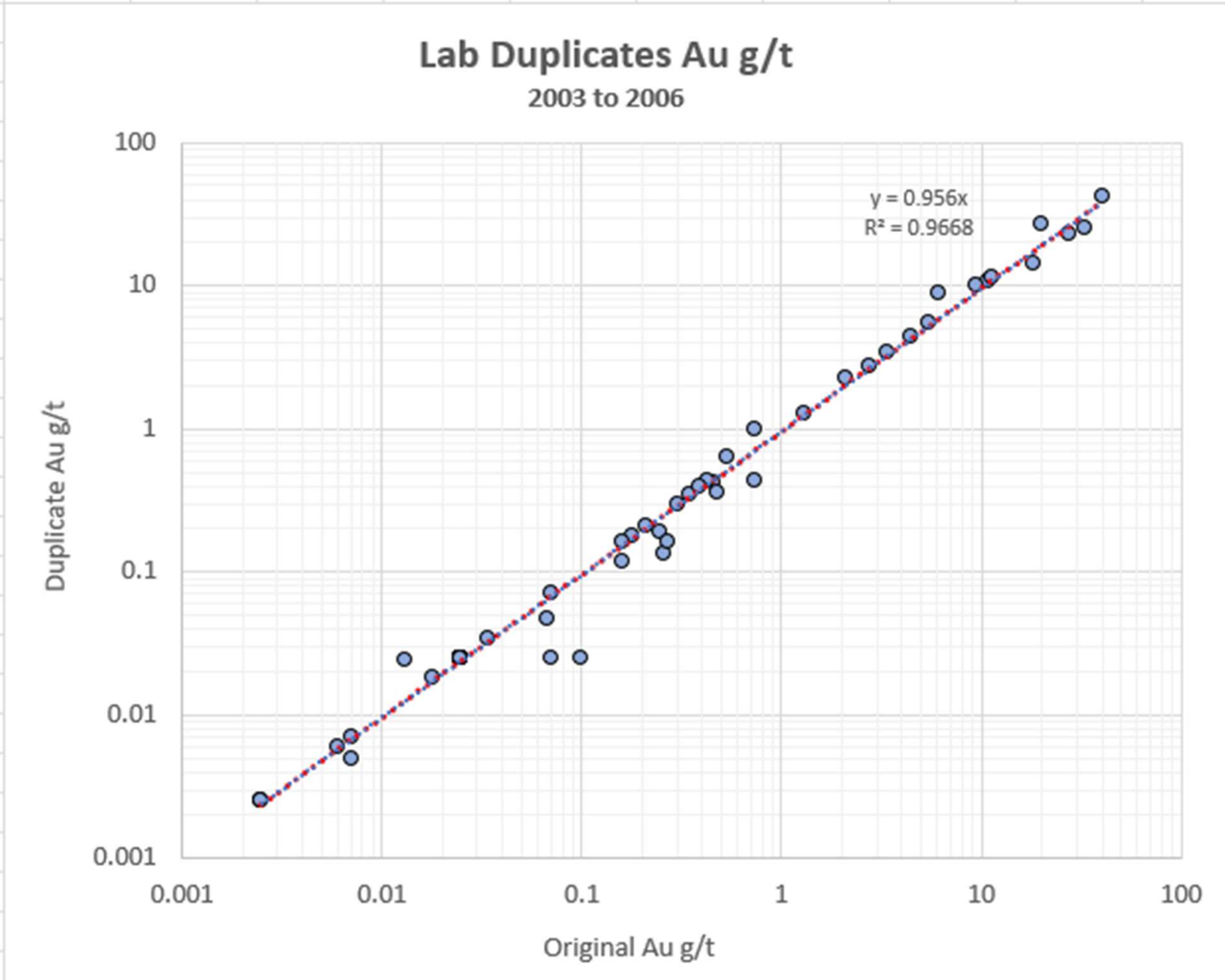
Duplicate samples were made by cutting ½ of the drill core into ¼ core and submitting the quarters as two different samples. In total 45 duplicate samples were assayed. The sample statistics are shown in Table 11-2.

Table 11-2 Univariate Statistics, Duplicate Samples

Parameter	Result, First Sample	Result, Duplicate Sample	Result, Sample Difference
Population	67	67	67
Minimum value	0.025	0.025	-7.25
Maximum value	40.1	41.7	7.5
Mean value	2.00	1.93	0.009
Standard Deviation	5.82	5.71	0.62
CV	2.91	2.96	67.42

(Source: MMTS, 2023)

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 0.97.



(Source: MMTS, 2023)

Figure 11-2 Duplicate Samples – 2004 through 2006

11.2.3 Standards – 2005-2006

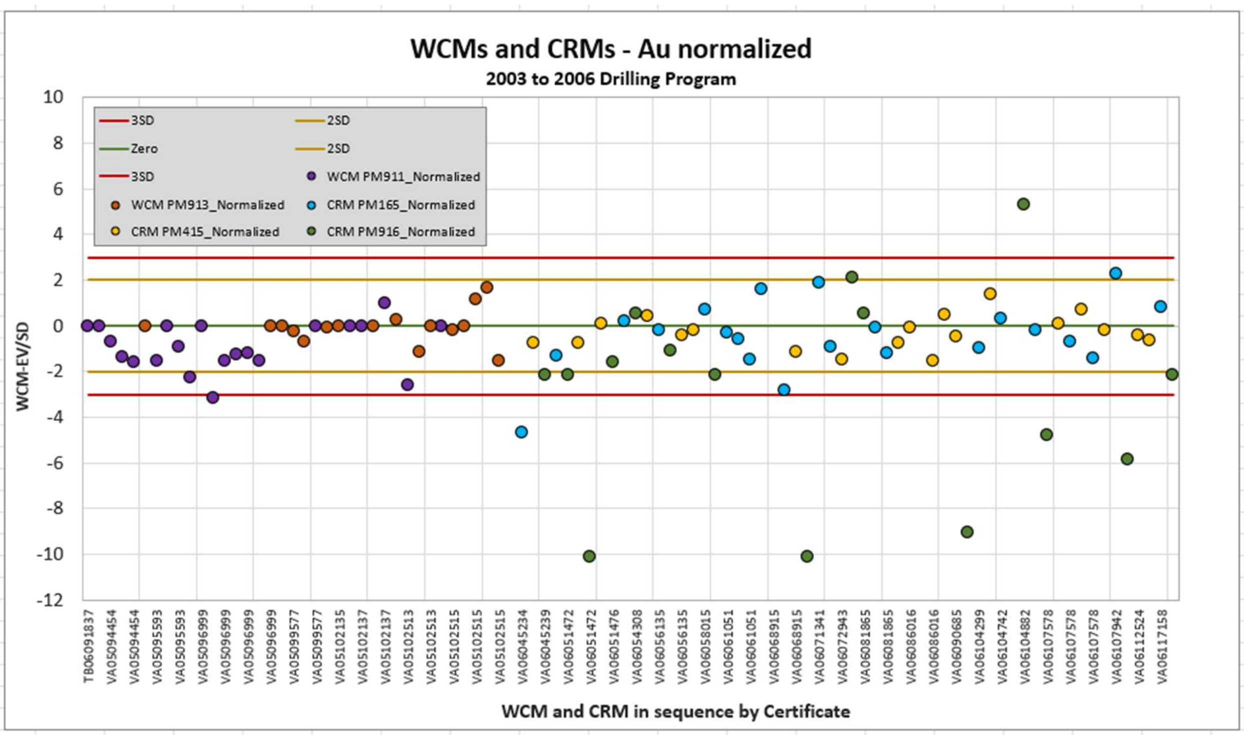
Five different standard samples were submitted during the years 2005 and 2006 throughout the program to test the accuracy of the laboratory, as summarized in Table 11-3.

(Source: MMTS, 2023)

Figure 11-3 shows the results for standards listed above. As shown, there are seven samples with lower than acceptable values and one with a higher than acceptable value. These samples, and others in the same batch, should be re-assayed.

Table 11-3 Standard Samples – 2005-2006

Standard	Year	Mean (Au g/t)	Standard Deviation	Upper Range	Lower Range
PM 911	2005	15.38	0.59	15.97	14.79
PM 913	2005	5.45	0.12	5.33	5.57
PM 916	2006	12.7	0.09	12.9	12.5
PM 165	2006	6.51	0.10	6.71	6.31
PM 415	2006	2.37	0.12	2.49	2.13



(Source: MMTS, 2023)

Figure 11-3 Standard for 2005-2006 Drilling

11.2.4 Re-Assays - 2006

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. A complete analysis of the round-robin analyses is provided in the previous PEA report (MMTS, 2019).

11.3 2021 to 2022 Drill Program Sampling

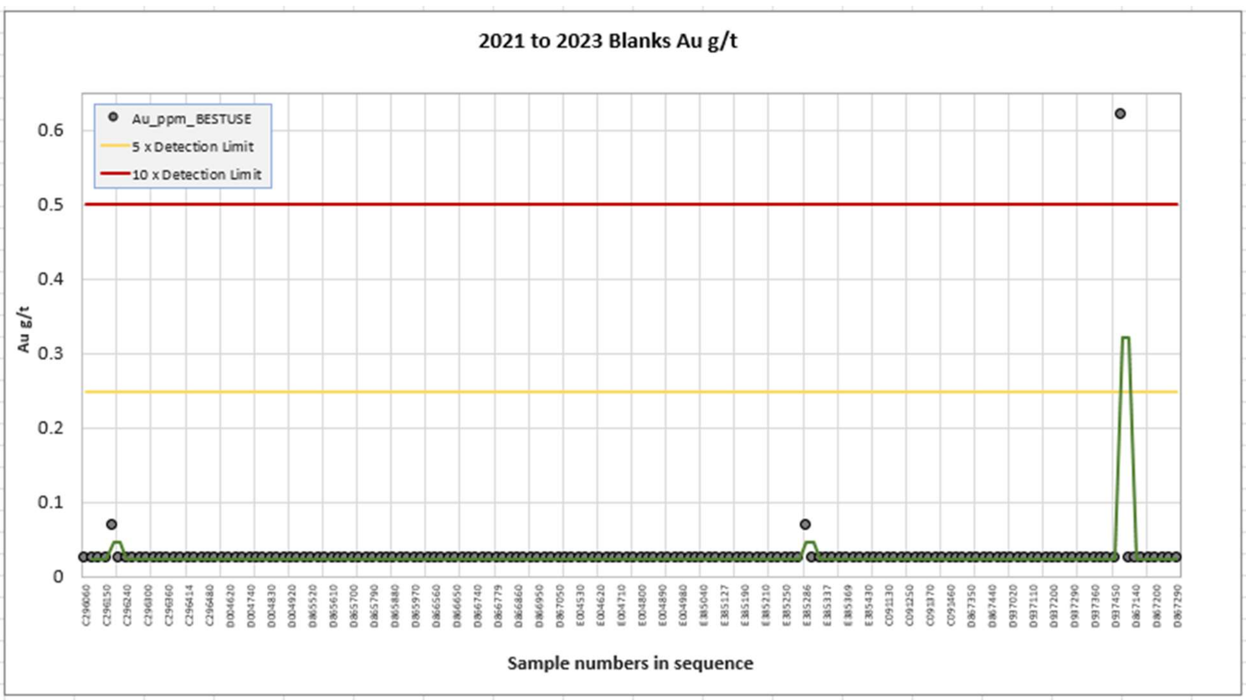
The 2021-2022 QA/QC program was like the previous programs in that samples were collected by employees of Canagold on site and shipped to ALS Chemex laboratory in Whitehorse, YT in 2021 and Yellowknife, NT 2022. For quality control and quality assurance, core samples were regularly mixed with blanks, duplicates, and standards at a rate of one in ten in a rotating sequence. 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 11-4.

Table 11-4 Blank Samples – 2021-2022

Parameter	Result
Population	160
Minimum value	0.025
Maximum value	0.62
Mean value	0.029
Standard Deviation	0.047
CV	1.6

One sample was greater than ten times the detection limit. The noted sample was D937480, with a value of 0.62. Figure 11-9 shows the test results of the blank samples.



(Source: MMTS, 2023)

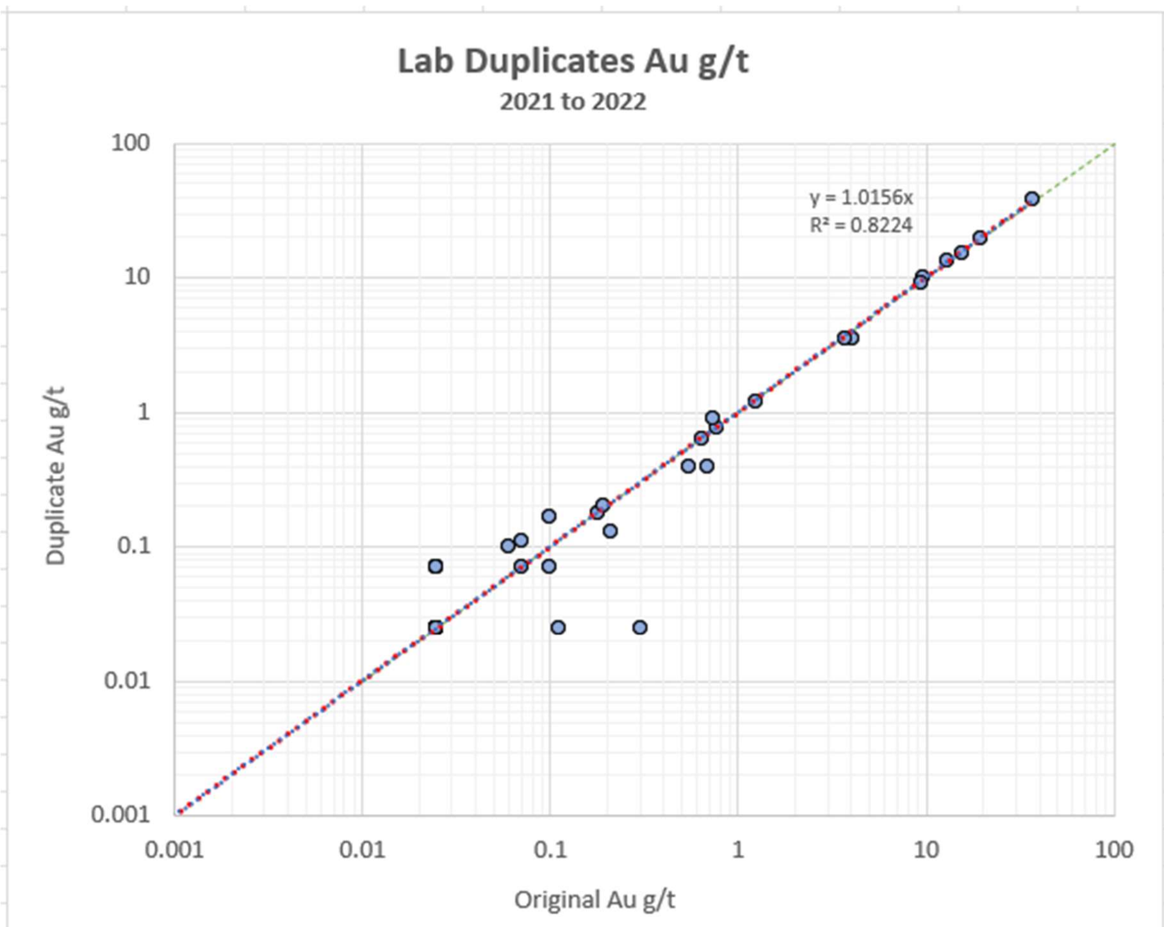
Figure 11-7 Blank Samples, 2021 to 2022

Duplicate samples were made by cutting ½ of the drill core into ¼ core and submitting the quarters as two different samples. In total 143 duplicate samples were assayed. The sample statistics are shown in Table 11-5.

Table 11-5 Duplicate Samples – 2021-2022

Parameter	Result, First Sample	Result, Duplicate Sample	Result, Sample Difference
Population	143	143	93
Minimum value	0.0025	0.0025	0
Maximum value	38.4	37.3	1.1
Mean value	1.44	0.70	0.74
Standard Deviation	4.56	4.15	0.41
CV	3.18	4.51	1.33

Figure 11-8 shows the duplicate sample results. As shown, there is a strong correlation between the two sample sets, with an overall coefficient of correlation of 0.82. the correlation is particularly good for values in the range of interest (above 4.0 g/t Au).



(Source: MMTS, 2023)

Figure 11-8 Duplicate Samples, 2021 - 2022

12 Data Verification

The QP, Sue Bird visited the property on the 25th of August 2022. At the time of the site visit core logging procedures from the 2022 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.

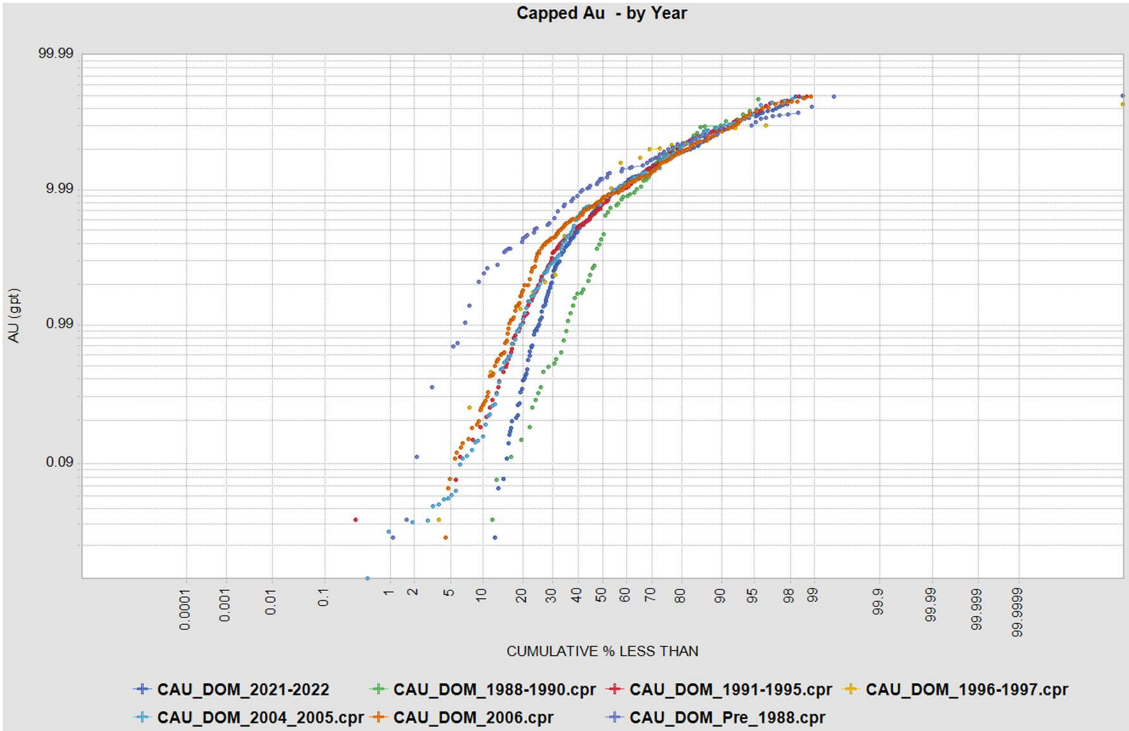
While on the property, the author along with the chief geologist, Troy Gill, P.Geo. examined underground workings to confirm the nature of the mineralization, as well as the dimensions and extent of the vein system. The author also viewed a selection of core from key holes drilled from the early 1990's 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 out to help separate variance caused 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. Also, the input of the database into the geology and resource modeling software was also checked.

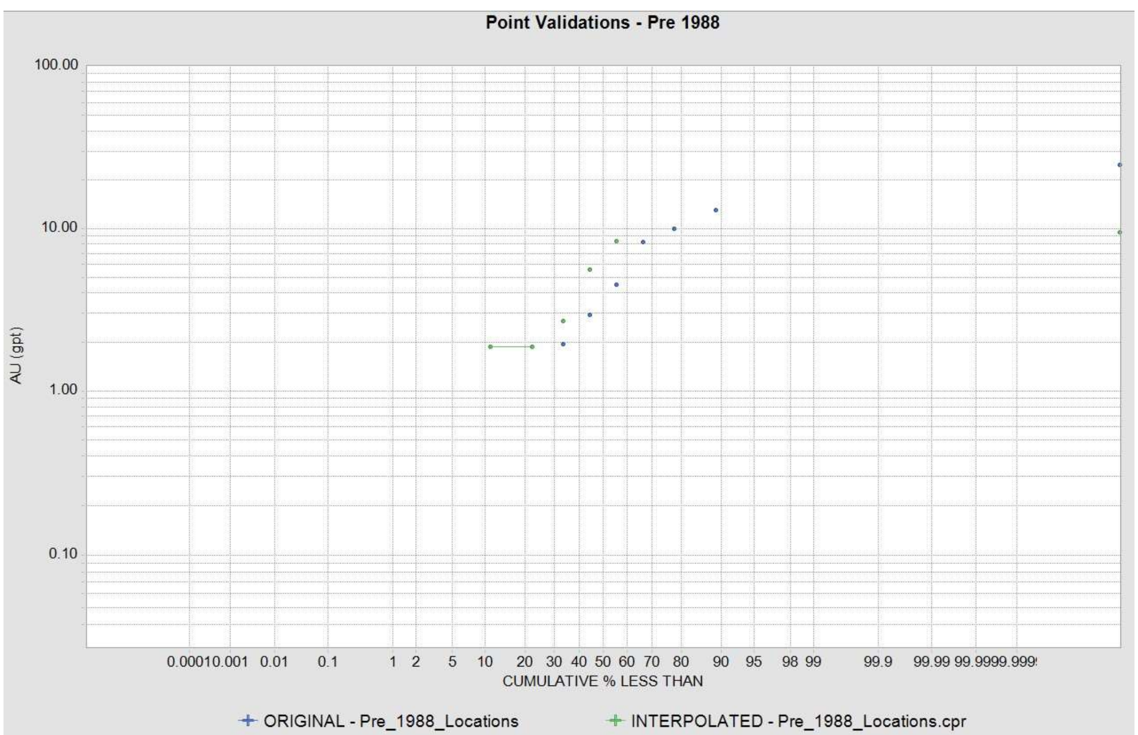
The quality control systems in place prior to the 2003 program are poorly documented but seem to follow the norms of that period. Of concern is the way the collar locations of drillholes were determined. Most of the holes were located using Brunton compass and chaining. Also, the down hole surveying was not consistently done. As a result, the exact location of the vein intersections is not as certain as those from the drilling after 2003. 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.

To further validate the drillhole database, comparisons have been made of the results by year. As illustrated in (Source: MMTS, 2023)
Figure 12-1, the capped Au grades do not indicate any bias based on year drilled except possibly the pre-1988 holes. However, these holes are all within the high grade central portion of the deposit and drilled from underground so are expected to be higher grade. Therefore, a point validation has been completed to compare drillhole assays only within 20m of the 1988 data. This analysis, illustrated in (Source: MMTS, 2023)
Figure 12-2, which shows that the 1988 grade distribution is below the grade distribution of the surrounding data. (Source: MMTS, 2023)
Figure 12-1 also shows that the high Au grades (>30g/t) are lower for the pre-1988 than the other years. Both plots show that the pre-1988 data is somewhat conservative.



(Source: MMTS, 2023)

Figure 12-1 Comparison of Capped Au Grades by Year



(Source: MMTS, 2023)

Figure 12-2 Point Validation of Pre-1988 Data

The procedures used in the development of the database are considered by the QP to follow accepted industry standards. There is confidence that selected drillholes for the Mineral Resource Estimate are error-free data and are suitable 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.
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.

Laboratory	Sample Type	Tests	Comments
		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 Leach	Poor results: 10.6% Au recovery, 31 kg/t NaCN consumption.
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™.

Laboratory	Sample Type	Tests	Comments
	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

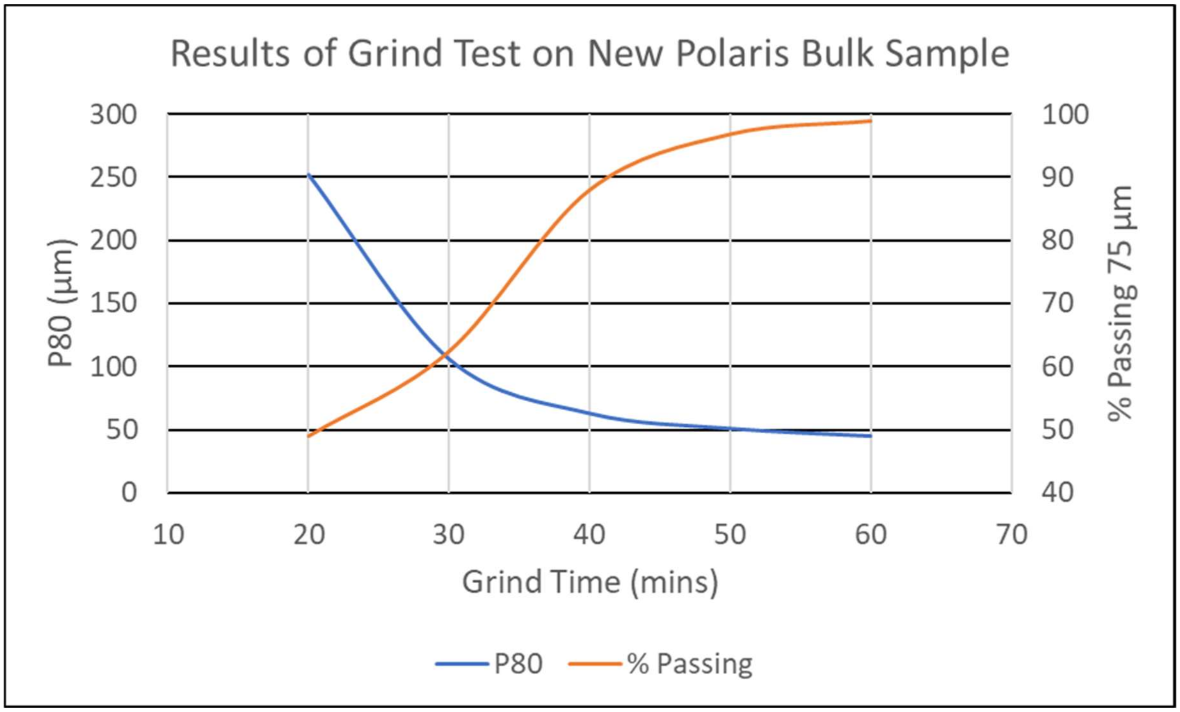
Gold mineralization is associated primarily with the sulphides arsenopyrite, pyrite, and stibnite. Gold is disseminated throughout these minerals along with gangue minerals including quartz and carbonate material. Approximately 10% of the ore material consist of sulphides of which less than 0.1% is Stibnite. 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.



(Source: MMTS, 2023)

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 Figure 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 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 with out 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 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 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

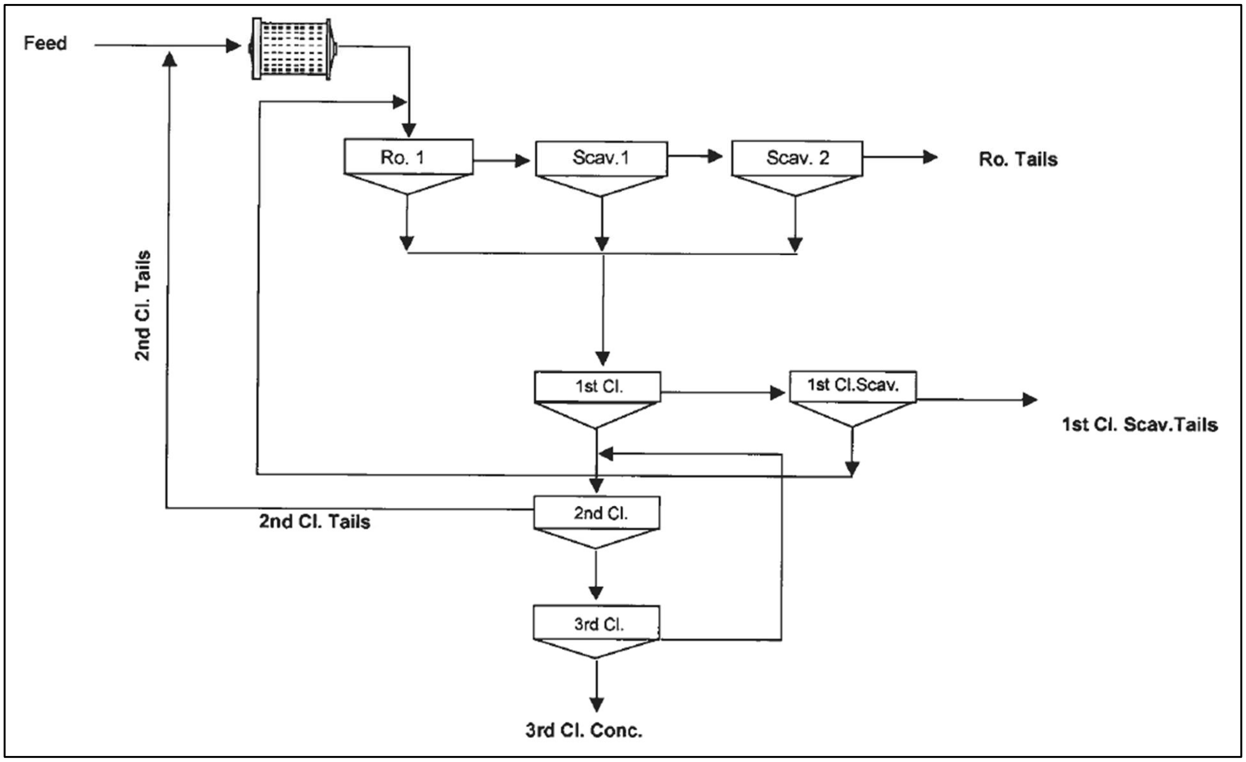
Table 13-19 Test F8 Flotation Test Results – Adapted from PRA 2007

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

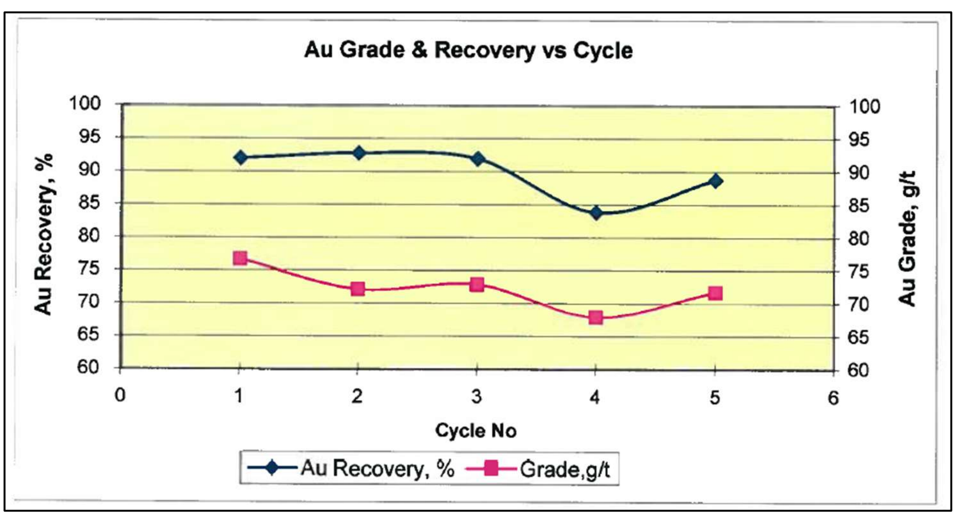
Locked cycle tests were conducted with multiple recycle streams, as shown in the Figure 13-2, taken from the PRA 2007 report.



(Source: PRA, 2007)

Figure 13-2 Locked Cycle Test Schematic Diagram

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.



(Source: PRA, 2007)

Figure 13-3 Locked Cycle Test Results

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 grade of the final concentrate steadily decreases from approximately 76% to 71% after 5 cycles. Results are given in Table 13-20.

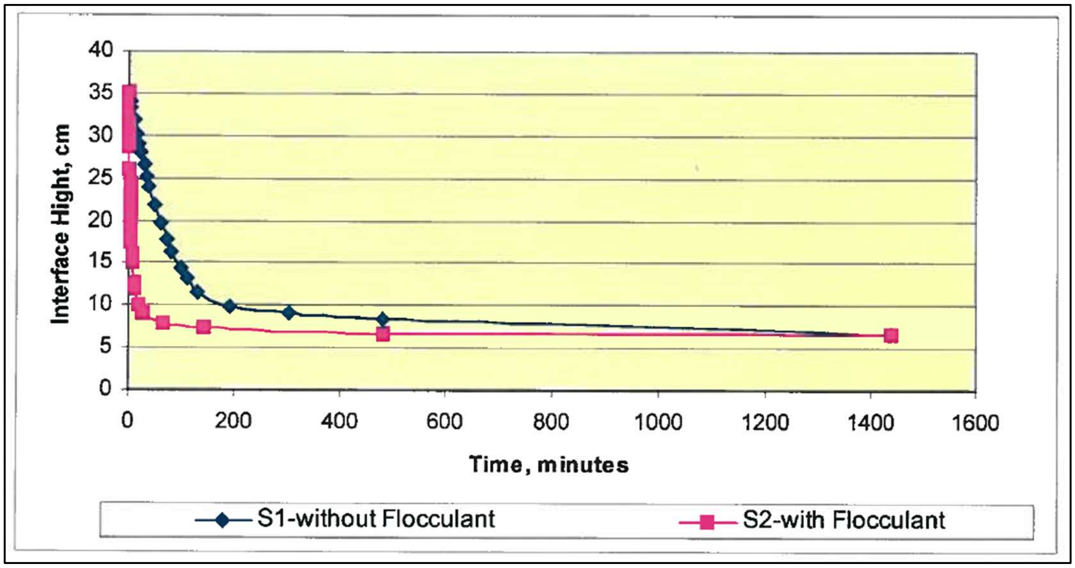
Table 13-20 Locked Cycle Flotation Test Results – Adapted from PRA 2007

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.



(Source: PRA, 2007)

Figure 13-4 Slurry Settling Test Results

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.

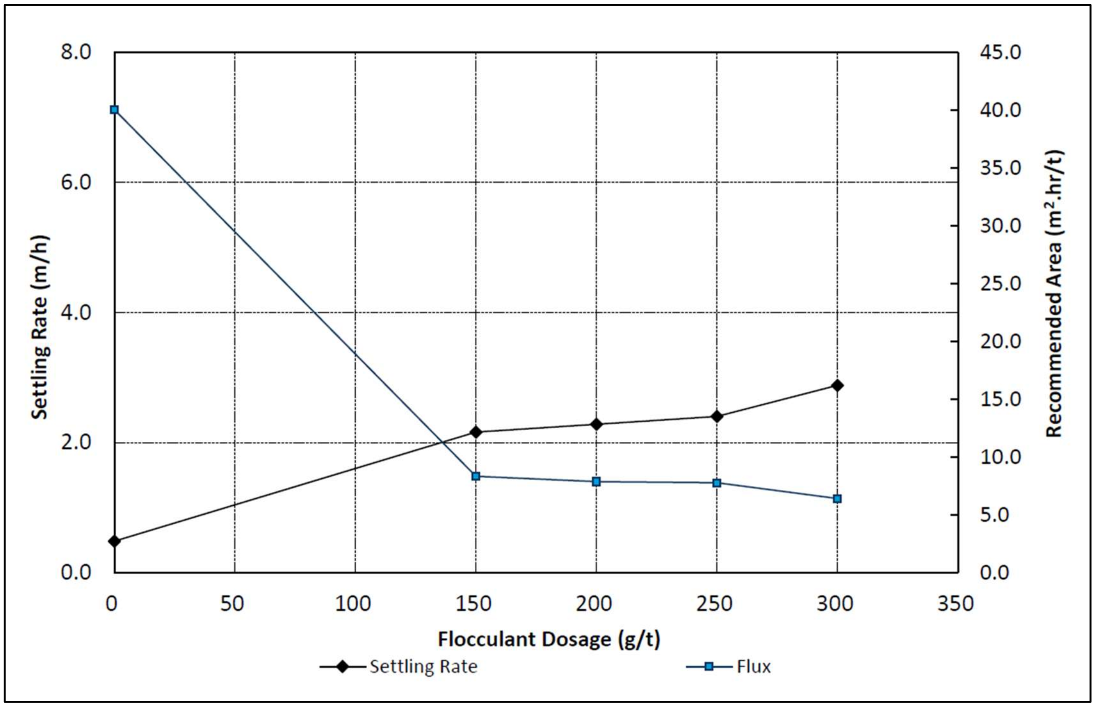
13.10.4 BIOX™ Slurry Settling Tests

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.



(Source: MMTS, 2019)

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

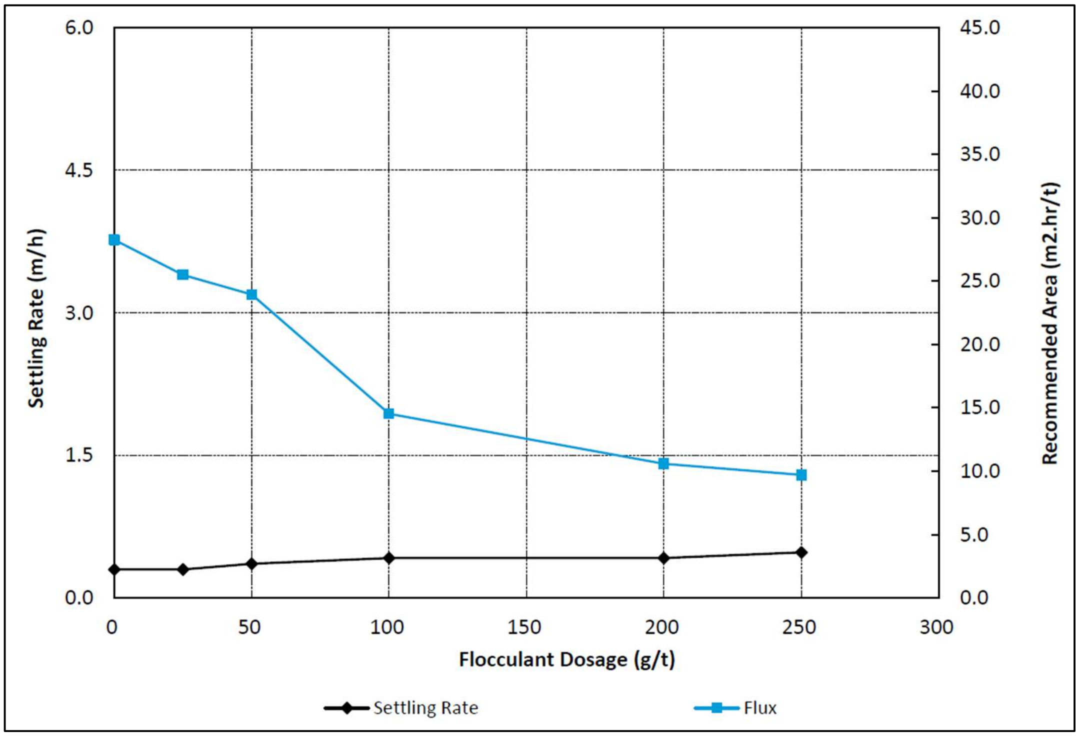
Results indicate that 150 – 250 g/t flocculant dosage were enough to produce a settling rate of 2.3 m/h, and a thickener area flux of 8 m^2h/t.

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.



(Source: MMTS, 2019)

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.

13.12 Metallurgical Performance Projections

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

Table 13-25 New Polaris Projected Metallurgical Recoveries

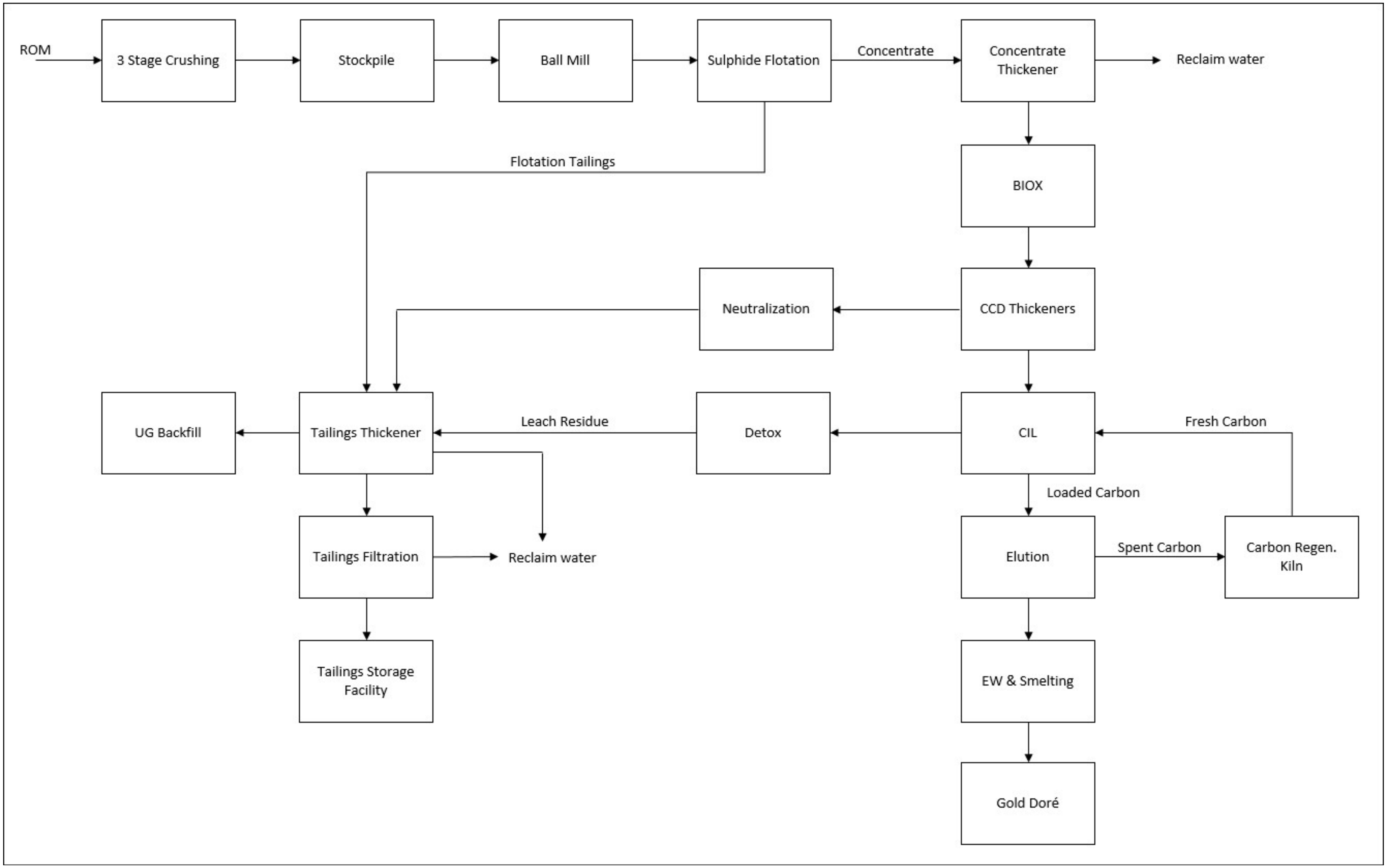
Area	Recovery (%)
Sulphide Flotation	94.9
BIOX and CIL Leach	95.6
Carbon Loss	0.1
EW	99.9

An overall gold recovery for the process flowsheet in (Source: MMTS, 2019)

Figure 13-7 is estimated at 90.5%.

13.12.1 Deleterious Elements and Other Factors

To the extent known there are no additional process 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.



(Source: MMTS, 2019)

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 (APEGBC #25007) of MMTS in accordance with updated Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (CIM 2014) and were estimated using the 2019 CIM Best Practices Guidelines. Updates from the previous model include additional drilling in 2021 and 2022, updated mineralized vein interpretations, and updated modelling methodology.

14.2 Mineral Resource Estimate

The Resource Estimate for the New Polaris deposit is summarized in Table 14-1. A comparison with the previous (2019) resource estimate is also summarized. 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 is 4 g/t Au and the Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shapes using the following assumptions:

- Metal prices of US$1,750/oz Au and Forex of 0.75 $US:$CDN;
- Payable metal of 99% Au;
- Offsite costs (refining, transport and insurance) of US$7/oz;
- Mining cost of CDN$82.78/t, Processing costs of CDN$105.00/t and G&A and site costs of CDN$66.00/t;
- Metallurgical Au recovery of 90.5%;

The “reasonable prospects for eventual economic extraction” confining shape also considers a minimum mining width of 2.0m, and removes shapes considered too small and separated from the primary mining volumes. Previous underground mining has been accounted for by restricting the modelled veins away from the mined-out shapes.

The effective date of this Resource estimate is April 20th, 2023.

Table 14-1 Updated Mineral Resource Estimate and Comparison to the 2019 Resource

Class	Cutoff (Au gpt)	2023 Resource Tonnage (ktonnes)	2023 Resource Au (gpt)	2023 Resource Au (koz)	2019 Resource Tonnage (ktonnes)	2019 Resource Au (gpt)	2019 Resource Au (koz)	Difference as a Percent: (2023-2019)/2019 Tonnage	Au Grade	Au Metal
Indicated	**3**	3,118	11.21	1,124	1,798	10.40	601	73%	8%	87%
	4	2,965	11.61	1,107	1,687	10.80	586	76%	8%	89%
	5	2,769	12.11	1,078	1,556	11.30	565	78%	7%	91%
	6	2,525	12.75	1,035	1,403	12.00	541	80%	6%	91%
	7	2,270	13.45	981	1,260	12.60	510	80%	7%	92%
	8	2,049	14.09	928	1,105	13.30	473	85%	6%	96%
	9	1,814	14.81	864	947	14.10	429	92%	5%	101%
	10	1,594	15.55	797	1,639	9.50	501	-3%	64%	59%
Inferred	**3**	1,061	8.24	281	1,582	9.80	498	-33%	-16%	-44%
	4	926	8.93	266	1,483	10.20	486	-38%	-12%	-45%
	5	817	9.52	250	1,351	10.70	465	-40%	-11%	-46%
	6	706	10.16	231	1,223	11.20	440	-42%	-9%	-48%
	7	603	10.78	209	942	12.50	379	-36%	-14%	-45%
	8	491	11.52	182	753	13.80	334	-35%	-17%	-46%
	9	371	12.51	149	653	14.60	307	-43%	-14%	-51%
	10	291	13.33	125	0	0.00	0			

Notes for Mineral Resource Estimate:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
2. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.
3. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines.
4. The base case Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shape using the following assumptions:
   - Metal prices of US$1,750/oz Au and Forex of 0.75 $US:$CDN;
   - Payable metal of 99% Au;
   - Offsite costs (refining, transport and insurance) of US$7/oz;
   - Mining cost of CDN$82.78/t ,
   - Processing costs of CDN$105.00/t and G&A and site costs of CDN$66.00/t.
   - Metallurgical Au recovery of 90.5%;
5. The resulting Net Smelter Return equation is: NSR (CDN$/t)=Au*90.5%*US$74.72g/t;
6. The specific gravity is 2.81 for the entire deposit;
7. Numbers may not add due to rounding.

The QP for the resource estimate is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant 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.

14.3 Key Assumptions and Data used in the Resource Estimate

14.4 Underground Workings and Topography

Underground workings and topography have been provided as solids and surfaces. They are the same as those used for the MMTS 2019 Resource Estimate with the exception that they have been converted to UTM coordinates (Underhill, 2021). The underground workings consist primarily of mined out stopes and stoping development levels used for historic long hole mining with side hill ramp access.

The topography and bottom of the overburden are both above any veins considered for the Resource Estimate, and the underground workings have been considered in the modelling procedure by clipping the modeled veins away from them. The underground workings and topography are shown in **Error! Reference source not found.**.

To the knowledge of the QP for the Resource Estimate and based on the tour of the property and discussion with the site geologist, there are no additional underground workings that have not been included in the shapes provided in the areas of the 2023 resource estimate. The topography and underground workings are shown in (Source: MMTS, 2023)
Figure 14-1.

14.4.1 Database

A summary of the total number of drillholes used for the Resource Estimate is found in Table 14-2 below. All zero value assays and missing assays values within the modeled vein shapes have been treated as zero on the assumption that they represent un-mineralized dilution.

Table 14-2 Summary of Drillholes and Assays used in the Resource Estimate

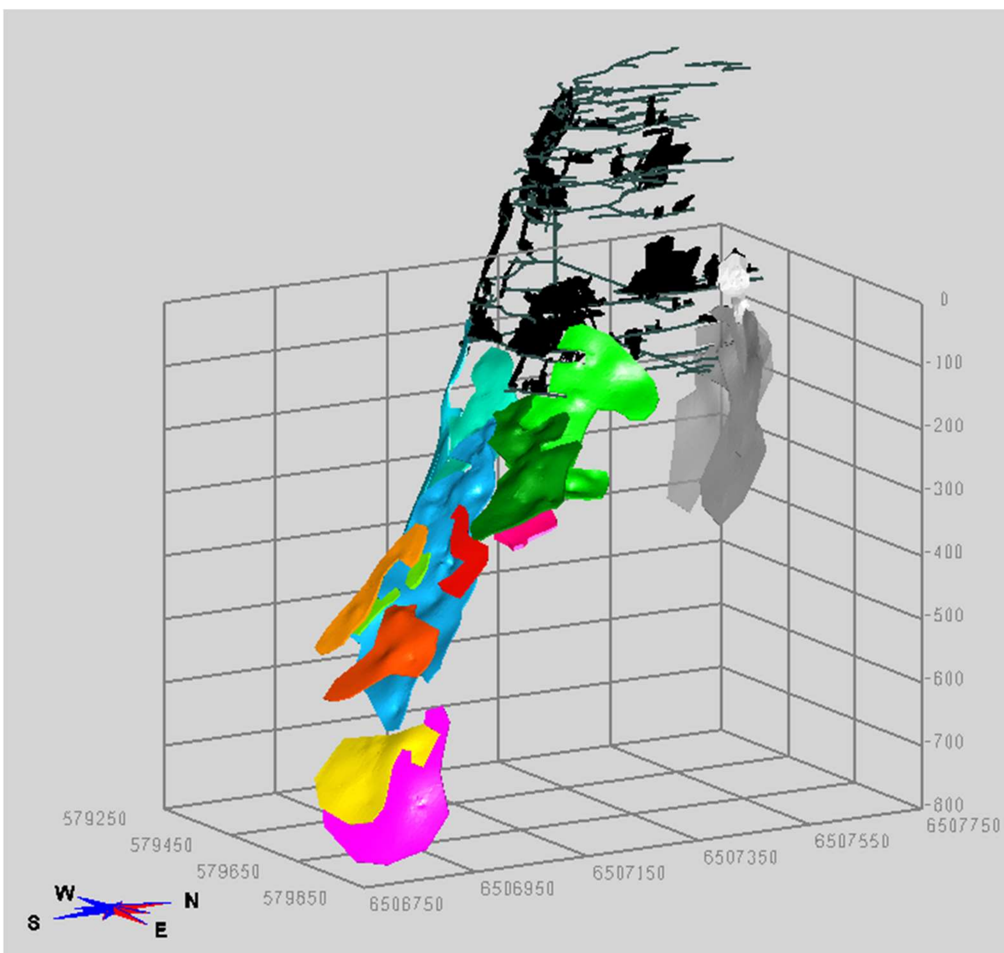
Year	Number of Drill Holes Intercepting Veins	Total Length of drilled (m)	Assayed Length in Veins (m)	Number of Assays	% Length Assayed
<1989	48	1,430.41	253.69	211	18%
1989	7	1,801.05	50.17	62	3%
1990	6	1,923.58	41.37	50	2%
1991	10	3,161.84	99.5	137	3%
1992	14	3,897.08	79.24	77	2%
1993	4	691.90	21.96	17	3%
1994	7	2,019.00	35.99	45	2%
1995	5	3,753.92	50.26	63	1%
1997	3	756.21	31.66	26	4%
2004	11	2,766.99	87.07	89	3%
2005	8	2,295.14	96.21	124	4%
2006	54	18,581.60	372.2	364	2%
2021	40	19,477.00	249.35	297	1%
2022	17	7,687.00	118.16	130	2%

Total	**234**	**70,242.72**	**1586.83**	**1692**	**2%**

14.5 Geologic Models

The mineralized veins discussed in Section 7 have been modeled in 3D using the Hexagon MinePlan Implicit Modeler® tool which uses the radial basis function (RBF like other industry software) to define surfaces based on user constraints.

A total of 17 veins (domains) have been modeled compared to 6 in the previous modelling. The increase in number of veins is due to the additional drilling, additional fault modeling and tighter control on dilution resulting in veins being split. The modeled shapes generally target a 4.0 g/t Au cutoff, while also honouring the known geological trends. Lower grade dilution is occasionally included to maintain continuity and to ensure the veins are a minimum of 2.0 meters in true thickness to align with the expected mining selectivity. The modeled shapes have also been clipped away from mined-out areas. An illustration of the veins is provided in Figure 14-1, with the "C" veins coloured and the "Y" veins to north shown in grey, and the existing underground workings shown in black.

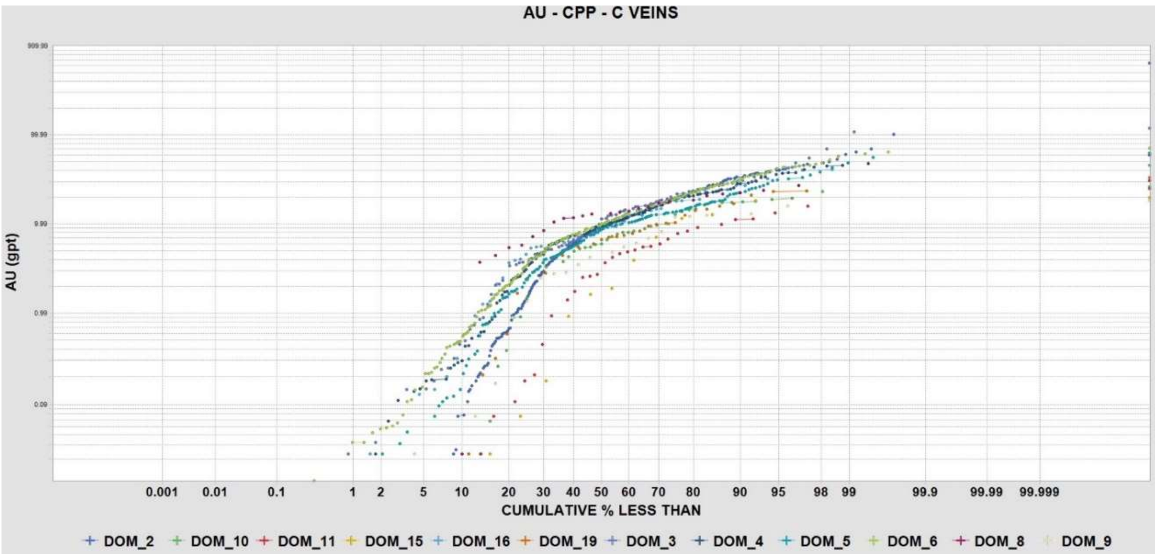


(Source: MMTS, 2023)

Figure 14-1 Three-dimension View – Veins and Existing Underground

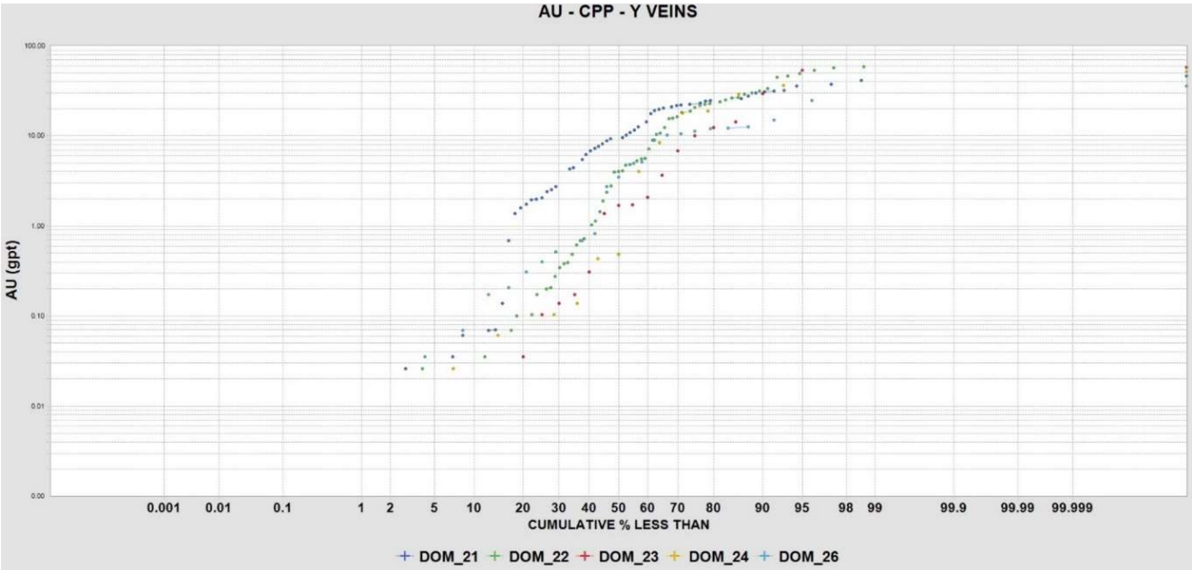
14.6 Assay Statistics, Capping and Compositing

MMTS has examined the sample assays in the veins 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 has been done. Table 14-3 summarizes the capping done on the assays prior to compositing. For clarity, also summarized in the table is the Outlier Restrictions which has been applied to the composites during interpolation. For composite grades above the Outlier value provided, and at distances greater than 4m from the data, the value is essentially capped to the outlier.



(Source: MMTS, 2023)

Figure 14-2 Gold Grade (gpt) by Domain – "C" Veins



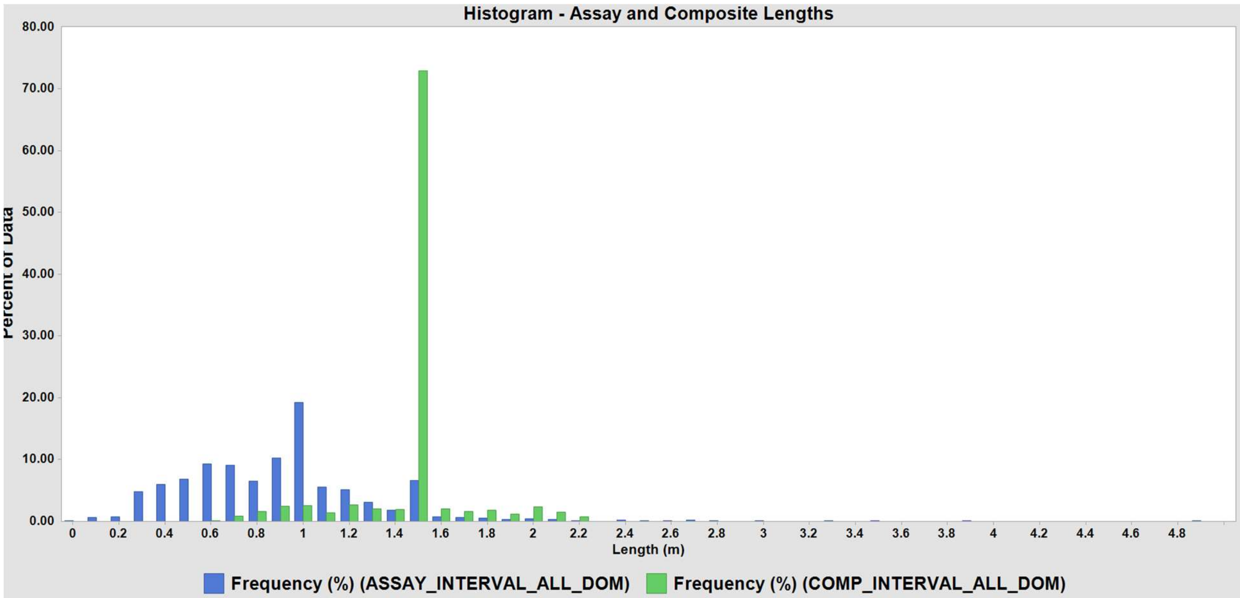
(Source: MMTS, 2023)

Figure 14-3 Gold Grade (gpt) by Domain – "Y" Veins

Table 14-3 Capping of Assays and Outlier Restriction of Composites by Domain

Domain	Au Cap (gpt)	Au Outlier (gpt)	Outlier Distance
2	50		
3	50		
4	50	25	4
5	50	25	4
6	50	35	4
8	25	20	4
9	13	10	4
10	20	15	4
11	10		
15	12		
16	30	20	4
19	20		
21	30	20	4
22	50	35	4
23	30		
24	30		
26	15		

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. Most 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.



(Source: MMTS, 2023)

Figure 14-4 Histograms of Assay and Composite Lengths within Mineralized Domains

The assay and composite basic statistics within the modelled veins are summarized in Table 14-4. The Coefficient of Variation before capping is 1.6 indicating that linear interpolation is appropriate.

Table 14-4 Summary Statistics of Assays and Composites within the Domains

	Source		
Parameter	**Assays**	**Capped Assays**	**Capped Composites**
Num Samples	1616.00	1692.00	1069.00
Num Missing Samples	76.00	0.00	0.00
Min (gpt)	0.01	0.00	0.00
Max (gpt)	646.00	50.00	50.00
Weighted SD	20.61	11.78	9.76
Weighted CV	1.63	1.05	0.91

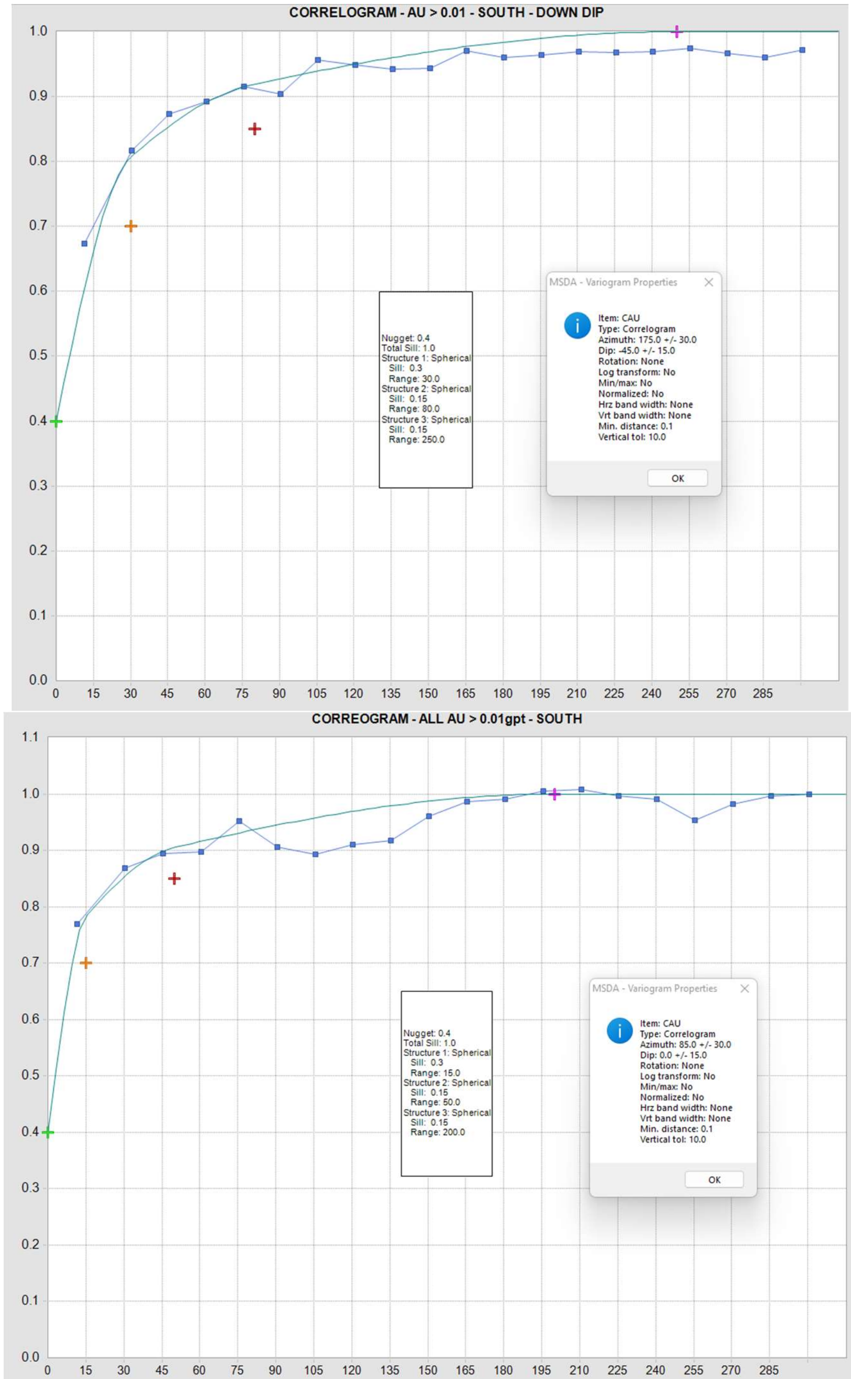
To ensure correct compositing the length and weighted mean grades of the composite have been compared to the original assay data in Microsoft Excel with the results summarized in Table 14-5 below illustrating that there is no bias introduced in the compositing process.

Table 14-5 Compositing Validation

	Grade X Length	Length (m)	Weight Average Grade (g/t)
All Au Assays in Database	53,704.4	16,903.4	3.177
All Au Assays in MineSight Assay Table	53,704.4	16,903.4	3.177
Capped Au in MineSight Assay Table	17,035.9	1,586.8	10.736
Capped Au in MineSight Composite Table	17,035.9	1,586.8	10.736

14.7 Variography

The interpolations have been done by inverse distance squared. However, variograms have been used to determine drillholes spacing necessary to classify the material to Measured, Indicated, and Inferred. The figure below illustrates variograms in the approximate strike and dip directions of the "C" veins.



(Source: MMTS, 2023)

Figure 14-5 Sample Variography for the "C" Veins

14.8 Specific Gravity

A total of 76 specific gravity determinations are available for examination. All measurements are from core drilled in 1996 and 1997. Unfortunately, most of these samples are from parts of the deposit not included in this resource estimate (higher up on the C Vein). A total of 16 samples are from C Vein Domain 4. These 16 samples had an average specific gravity of 2.81 and this is the value used for all estimated blocks. The average value of all the gold bearing samples is 2.88.

14.9 Block Model Estimation Parameters

Block dimensions are 5m x 5m x 5m with the extent of the block model summarized in UTM coordinates, in Table 14-6.

Table 14-6 New Polaris Model Extents

Direction	Minimum	Maximum	Size (m)	# Blocks
Easting	579,200	579,900	5	140
Northing	6,506,700	6,507,800	5	220
Elevation	-850	130	5	196

Modelling has been accomplished using inverse distance squared (ID2), which is a change from Ordinary Kriging (OK) which had been used previously. However, variograms were created on a global basis to aid in determination of Classification parameters.

Domain matching for each of the 17 domains has been used, and a search ellipse with axis in the same orientation as the primary axes of each domain has been used to select samples for interpolation. Composite values have also been restricted at higher cutoffs to reduce the impact of high-grade samples. Table 14-7 and Table 14-8 below summarize the interpolation search parameters.

Table 14-7 Summary of Search Parameters

Search Parameter		Pass1	Pass2	Pass3	Pass4
Search Distance (m)	Major	30	60	90	150
	Minor	30	60	90	150
	Vertical	10	20	30	30
Number of Composites	Minimum	4	4	4	4
	Maximum	12	12	12	12
	Maximum/DH	2	2	2	2
	Maximum/Quadrant	2	2	2	2

Table 14-8 Summary of Rotation by Vein

Domain	Rot-Y	Rot-X	Rot-Z
2	267	0	46
3	290	0	67
4	265	0	46
5	249	0	32
6	210	0	50
8	270	0	47
9	265	0	51
10	265	0	53
11	257	0	53
15	247	0	16
16	240	0	42
19	243	0	57
21	350	0	87
22	350	0	87
23	3	0	89
24	8	0	88
26	346	0	87

14.10 Classification

Classification is based primarily on anisotropic distances to drillholes with 50m grid drill spacing being targeted. However, additional adjustments have been made to ensure a cohesive shape of Indicated material is produced. All blocks not classed as Indicated are classed as Inferred. Table 14-9 below summarizes the classification parameters.

Table 14-9 Summary of Requirements for Classification to Indicated

Criteria		Distance (m)
1	Average Distance to closest 2 DHs	35
	Max. distance to closest 2 DH	50
	Minimum # Quadrants	2
2	Average Distance to closest 3 DHs	50
	Max. distance to closest 3 DH	70
	Minimum # Quadrants	2
3	Distance to closest DH	10
	Max. distance used	50
	Minimum # Drillholes used	3

14.11 Cutoff Grade and Reasonable Prospects of Eventual Economic Extraction

The base case cutoff grade is 4gpt Au, based on the following economic assumptions and preliminary production rate estimates.

- Metal prices of US$1,750/oz Au and Forex of 0.75 $US:$CDN;
- Payable metal of 99% Au;
- Offsite costs (refining, transport and insurance) of US$7/oz;
- Mining cost of CDN$82.78/t, Processing costs of CDN$105.00/t and G&A and site costs of CDN$66.00/t.
- Metallurgical Au recovery of 90.5%;
- NSR (CDN$/t)=Au*90.5%*US$74.72g/t;

The "reasonable prospects of eventual economic extraction" has been further confirmed by:

- Ensuring all Resources are within modeled domains having minimum 2m true thickness
- Creating a resource a gradeshell for each reported cutoff and removing any outlying blocks too small to be considered “reasonably minable”

14.12 Block Model Validation

The model has been validated by comparison of the Au grade and tonnage above cutoffs with the de-clustered grades and total metal. Further validations have been done by visually comparing the modelled grades to the assay data, as presented below.

14.12.1 Comparison of Modelled Grades to De-clustered Composites

To validate the block model, a Nearest Neighbor model has been created (NN) to compare the de-clustered composite data to the interpolated grades. The following Table 14-10 compares the relative metal content across grade-bins of Indicated Resources. Decreasing relative metal content as the grade increases confirms that interpolated grades are conservative compared to de-clustered composite values.

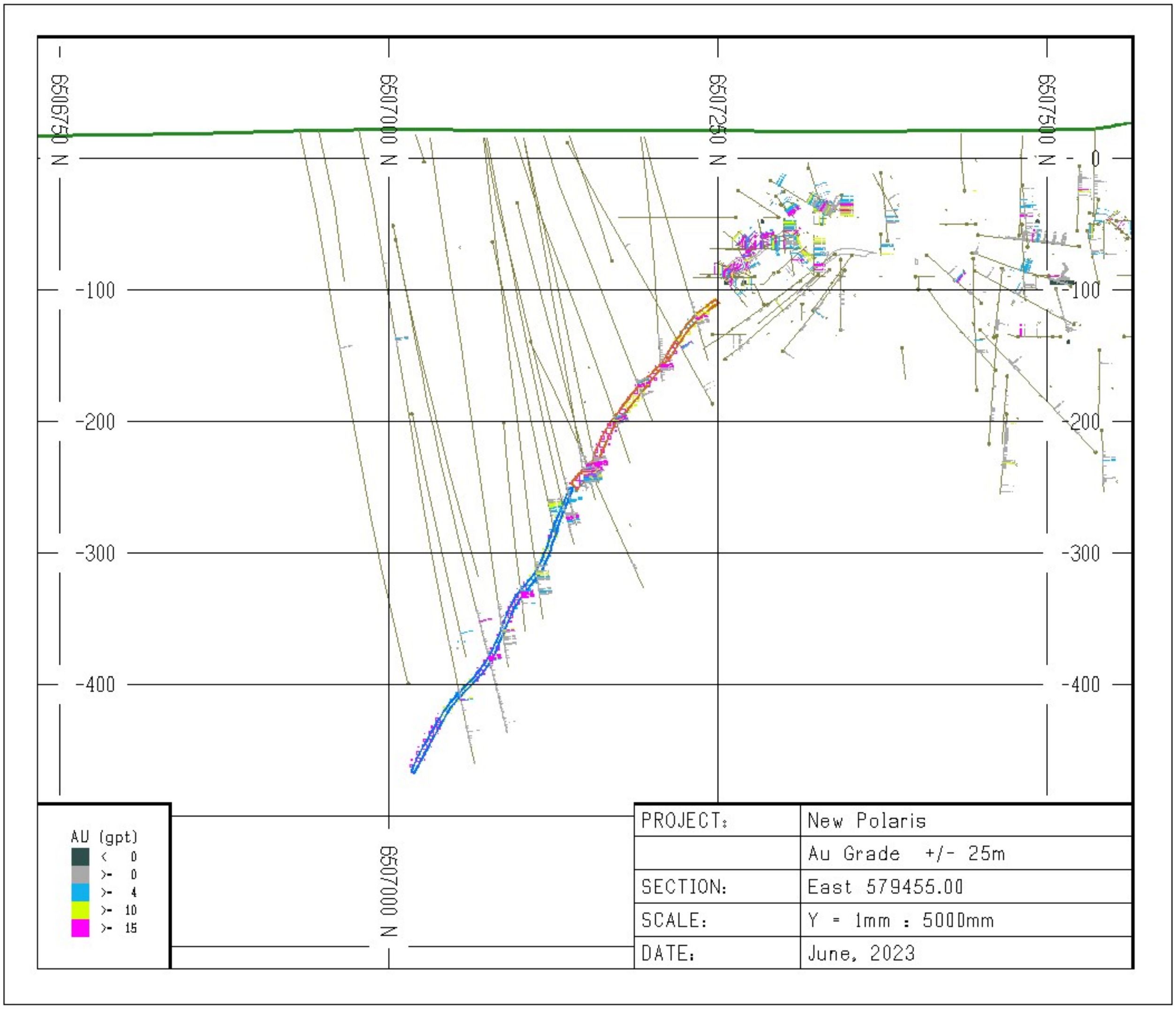
Although Domains 8 and 9 do not have a good comparison between modelled and NN model grades, they are very small domains that contributes little to the overall Resource. It is also established that Domain 9 contains assays at the peripheries of the solids, but still within the domains that are below cutoff, contributing to the values of lower de-clustered composite (NN) grades.

Table 14-10 Block Model and Declustered Composite Comparison, Au, Indicated Resources

Domain	ID2 / NN Contained Metal, Indicated Blocks only					
	Cuf off, g/t					
	1	4	6	9	15	% Total at 4 g/t Cut Off
2	92%	92%	92%	90%	68%	25.9%
3	96%	98%	98%	92%	70%	10.5%
4	96%	96%	98%	96%	77%	8.5%
5	96%	97%	94%	90%	85%	13.3%
6	97%	96%	93%	96%	90%	17.6%
8	95%	95%	93%	114%	106%	1.9%
9	131%	134%	113%	0%	0%	0.2%
10	96%	95%	84%	97%	0%	3.7%
11	91%	87%	54%	0%	0%	1.2%
15	76%	68%	58%	41%	0%	0.4%
16	90%	91%	91%	74%	59%	4.0%
19	102%	102%	100%	99%	0%	1.4%
21	84%	83%	82%	82%	45%	5.1%
22	95%	95%	95%	99%	69%	4.5%
23	78%	77%	70%	38%	6%	0.4%
24	72%	69%	86%	39%	29%	0.3%
26	90%	86%	91%	71%	0%	1.1%
All Domains	**94%**	**94%**	**93%**	**91%**	**72%**	**100%**

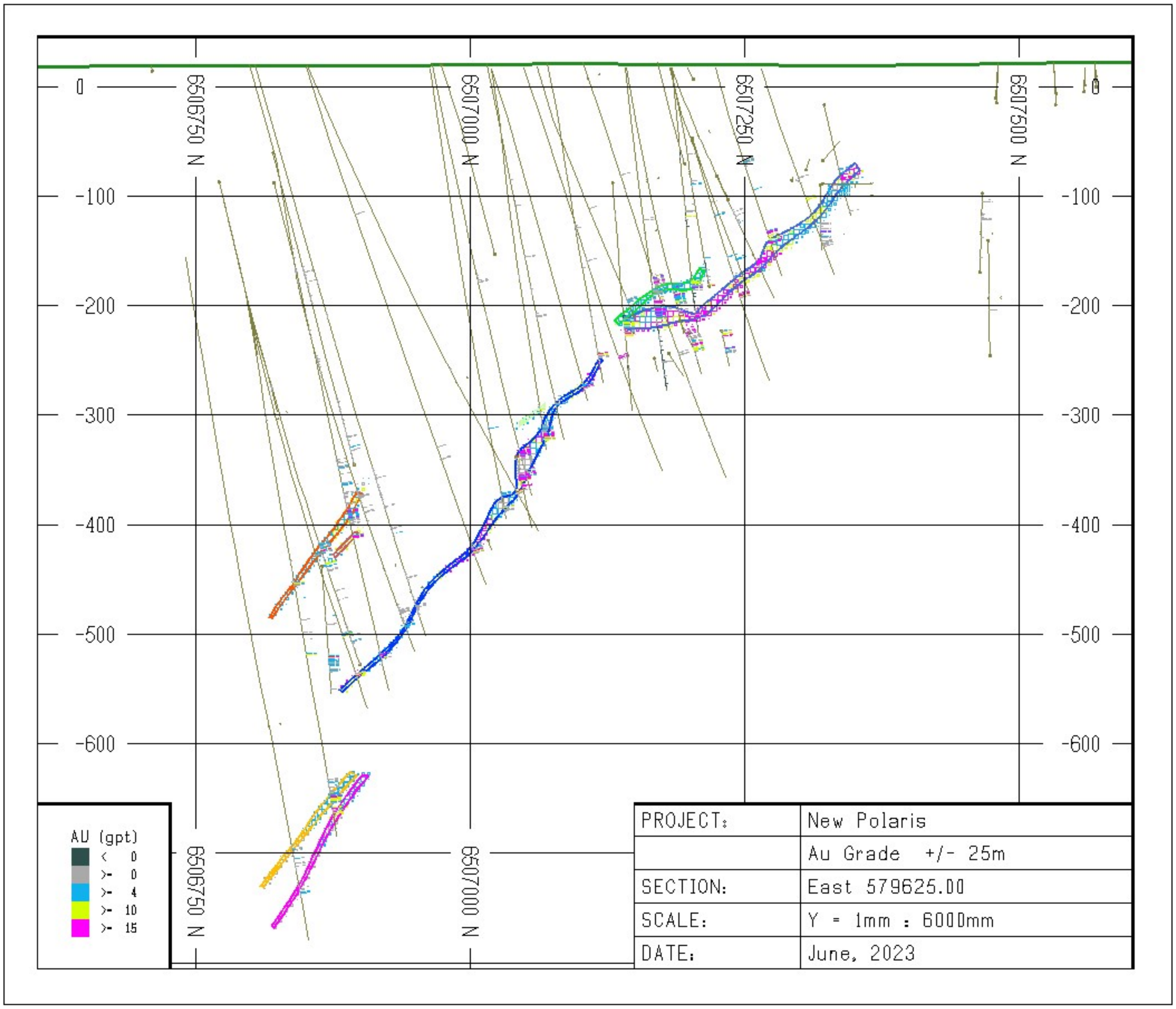
14.13 Visual Validation

The modelled 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 14-6 and Figure 14-7 for the “C” veins and in Figure 14-8 for the “Y” veins. The assay data has been projected +/- 25m 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.



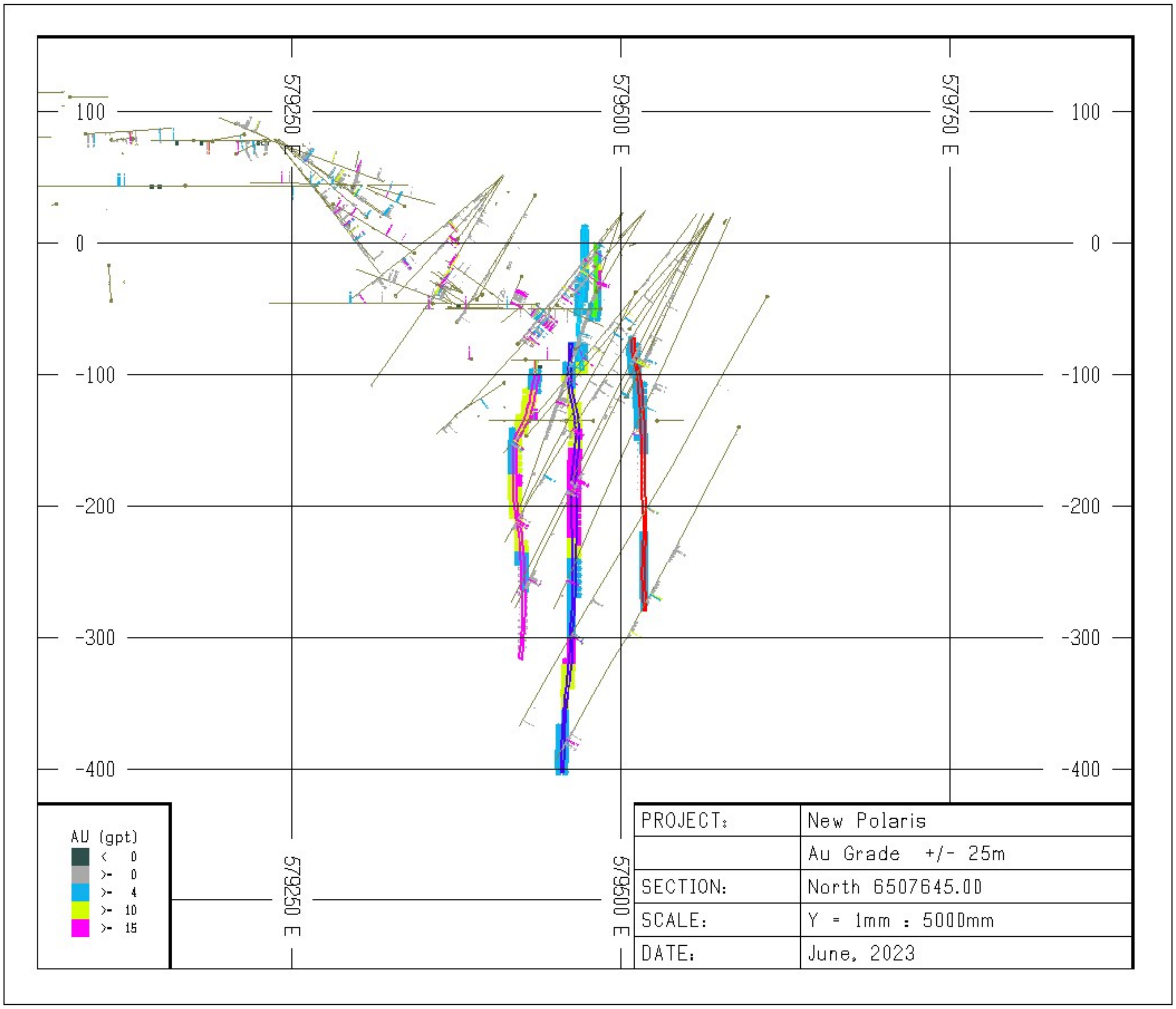
(Source: MMTS, 2023)

Figure 14-6 Comparison of Assay Au Grades and OK block grades – Section 579455E



(Source: MMTS, 2023)

Figure 14-7 Comparison of Assay Au Grades and OK block grades – Section 579625E



(Source: MMTS, 2023)

Figure 14-8 Comparison of Assay Au Grades and OK block grades – Section 6507645N

14.14 Independent Checks

An independent check on the modelling has been done by George Dermer, P.Eng of MMTS who checked:

- the resource shapes
- the model coding
- the "reasonable prospects of eventual economic extraction" shapes and inputs
- the interpolation runs
- The Nearest Neighbour Validations

14.15 Risk Assessment

#	Description	Justification/Mitigation
1	Classification Criteria	Based on variography
2	Geologic Model	Geologic interpretations and orientations of previous underground working considered when creating new geologic confining shapes for the resource interpolations. Faults used to define changes in orientations/offsets
3	Metal Price Assumptions	Cutoff is based on US$1750/oz Au, which is below the current prices and based on 3-year trailing average.
4	High Grade Outliers	Capping and outlier restriction applied to ensure modelled mean grade matches data. Grade-tonnage curves show modelled metal validates well with de-clustered composite data throughout the grade distribution.
5	Processing and Mining Costs	Assumed from comparables and based on mining method.
6	Previous underground mining	Underground workings examined during site visit. 2021 drilling did not hit un-expected voids.

15 Mineral Reserve Estimates

This section is not relevant to the Technical Report.

16 Mining Methods

This section is not relevant to the Technical Report.

17 Recovery Methods

This section is not relevant to the Technical Report.

18 Project Infrastructure

This section is not relevant to the Technical Report.

19 Market Studies and Contracts

The Project is expected to yield gold doré as its final product, which is expected to be sold on the spot market through marketing experts retained by Canagold. 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 price of US$1,750/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 and conditions for gold sales will be typical of similar contracts in the industry for the sale of doré.

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 supplementary work was started in 2019 to facilitate a formal environmental assessment with the British Columbia Environmental Assessment Office (BCEAO) to review and refresh data. These include but are not limited to:

- 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
- Wildlife

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 include 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). 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, and additional studies have been performed to update previous information.

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

20.1.2 Geochemistry

Canagold 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 generated in nature, due to high carbonate content. It should be noted there were some localized areas that showed some potential for acid generation (URS Canada, 2007b). 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 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 being done 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 drystack facility and will not be designed in a conventional fashion to store water. Underground disposal will be directed to old workings to minimize the total surface disturbance footprint.

20.1.3 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 seek regulatory, and First Nations input and approval prior to the start of mining and processing operations.

Reclamation costing will be developed as part of the mine planning and feasibility study stages. At this time, no assessment as to closure amounts has been completed.

20.2 Regulatory Framework

The New Polaris Project will be subject to Provincial and Federal approvals to proceed to construction and operations phases. A federal environmental assessment is not expected as the production threshold does not meet the 5,000 tpd threshold. Additional authorizations and input may also be required from United States authorities as the Project area drains into US waters (Transboundary Water).

20.2.1 Provincial Processes

When a major project is proposed in British Columbia, it must undergo an environmental assessment. In B.C. environmental assessments are managed by the Environmental Assessment Office (EAO), a neutral regulatory agency within the provincial government.

The BCEAO 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.

More information can be found on the BC government website:
https://www2.gov.bc.ca/gov/content/environment/natural-resource-stewardship/environmental-assessments

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.

The estimated time to secure all necessary Provincial approvals for The Project to proceed is 1 – 3 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 Closure 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 has not been fully developed.

Some of the Provincial Acts and Regulations that may apply to The Project include the;

- Environmental Assessment Act
- Drinking Water Protection Act
- Energy Efficiency Act
- Environmental Management Act (including wastes and contaminated sites authorizations)
- Fish Protection Act
- Forest and Range Practices Act
- Forest Practices Code of BC Act
- Geothermal Resources Act
- Health, Safety and Reclamation Code for Mines in BC
- Heritage Conservation Act
- Archaeological permitting and registries
- Hydro and Power Authority Act
- Land Act
- Crown Land tenure applications

- Local Government Act
- Mineral Tenure Act
- Mines Act (including Mining and Reclamation Permits and Bonding)
- Transport of Dangerous Goods Act
- Transportation Act
- Water Act
- Water rights
- Water license application
- Water Protection Act
- Water Utility Act
- Water utilities
- Wildlife Act
- Wildlife permits and commercial licenses

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.

Federal permits and approvals that still may be required are included in the following;

- Canadian Environmental Protection Act
- Transboundary Waters Protection Act
- Fisheries Act
- Migratory Birds Convention Act
- Navigable Waters Protection Act
- Species at Risk Act
- Transportation of Dangerous Goods Act
- Canadian Environmental Assessment Agency Registry
- Canadian Transportation Agency
- Environment Canada, including the Canadian Wildlife Service and Species at Risk requirements
- Indian and Northern Affairs Canada
- Fisheries and Oceans Canada Species at Risk, including Species at Risk requirements
- Health Canada requirements
- Natural Resources Canada

- Transport Canada requirements
- Explosives Act

It is not unreasonable to expect approvals will be received for the New Polaris Project to be 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 is 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.

Canagold has developed a good working relationship with the TRTFN and maintains a collaborative agreement to formalize their commitment for open, honest communications that were already ongoing between Canagold and TRTFN. TRTFN has a Mining Policy that relates to the development of projects within their traditional territory. A 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 have been updated as part of the baseline studies to support the BC Environmental Assessment. A Chance Find Procedure will need to be developed to ensure the identification and protection of additional areas.

Canagold is required to develop and maintain a Consultation and Engagement Plan with TRTFN, provide timelines and details of any engagement, and document any agreements reached between them with respect to 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 requires a Plan and Implementation Strategy to be developed and maintained with the input of identified communities and other potentially affected stakeholders. This engagement has started and will be ongoing.

20.3 Socio-Economic

The mine is 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).

It is expected that the project will enhance employment opportunities for the people of 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. Several 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 Canagold 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. Canagold 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 many 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 be 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

This section is not relevant to the Technical Report.

22 Economic Analysis

This section is not relevant to the Technical Report.

23 Adjacent Properties

The Tulsequah Chief 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

An updated Mineral Resource Estimate for the New Polaris Project has been completed. It is the opinion of the QP's that the resource is of sufficient quality to proceed with engineering studies and further drilling to upgrade the Classification of additional resources and potentially extend the mineralization.

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 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.
- Canagold has been drilling on the Property since 1988. To date total drilling totals about 145km in approximately 1100 drillholes.
- The sample security, sample preparation and analytical procedures during the exploration programs by Canagold followed accepted industry practice appropriate for the stage of mineral exploration undertaken.
- Data verification has been conducted by Canagold, 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 base case Mineral Resource contains an Indicated resource of 2.97 Mt of 11.61 g/t Au for 1.1Moz of Au and an Inferred resource of 0.93 Mt of 8.93 g/t Au for an additional 0.27 Moz of Au.
- The base case Mineral Resource is confined by an underground shape that targets material above a 4.0 g/t Au cutoff grade which shows suitable economics for further studies.
- The Mineral Resource has been updated for the "C" and "Y" veins of the New Polaris deposit.
- In total, 234 core drillholes from 1988 to 2022 inclusive have been used to determine the Resource estimate. A limited amount of historical data has been used and was validated.
- 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 for underground mining at the current state of knowledge of the deposit.

25.4 Metallurgy and Process

- The completion of a several metallurgical test work campaigns has 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 is expected to 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.
- An overall gold recovery of 90.5% is projected at this point in the project analyses.

25.5 Environmental, Permitting and Social Impact

- It is not unreasonable 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.

26 Recommendations

The following programs and studies are recommended and are currently ongoing as part of the requirements for the feasibility Study that is in process. The estimated budget for the Feasibility Study is summarized in Table 26-1.

26.1 Metallurgical Recommendations

- Additional metallurgical testwork 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.

26.2 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. 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.

Table 26-1 Exploration Budget Estimate

Item	Description	Estimated Budget (M$)
1	Geotechnical Study and Rock Mechanics Analysis	$0.2
2	Mining Studies	$0.7
3	Metallurgical Studies and Process Refinement	$0.5
4	Site Plan GA Refinement	$0.3
5	Geotechnical Site Investigation	$1.0
6	Water Quality Testing	$0.6
7	Hydrology	$0.3
8	Hydrogeology	$0.5
9	Fish and Fish Habitat Studies	$0.1
10	Air, Noise and Climate Studies	$0.1
11	Vegetation Studies	$0.1
12	Wildlife Studies	$0.2
13	Soil Quality Testing	$0.2
14	Terrain and Seismic Studies	$0.2
15	Archaeology	$0.2
16	First Nations and Community Engagement	$0.3
17	Traditional Knowledge/Use Study	$0.2
18	Transportation and Logistics	$0.2
19	Order of Magnitude and Trade Off Studies	$0.1
20	Estimating and Financial Modeling	$0.1
Total		**$6.1**

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.

27 References

Atlas Copco (2003): Underground Mining Methods, First Edition, August 2003

BC Ministry of Mines Annual Reports 1936 to 1953

Beacon Hill Consultants Ltd. (1989): Conceptual Study by RJ Morris, M.Sc.; BM Briggs, P.Eng., and WR Stokes, P.Eng, Suntac Minerals Corporation; internal company report

Beattie, M. (1993): Report on the metallurgical testing of gold ore from the Polaris-Taku Deposit, Golden Angus Mines Ltd. (Canarc Resource Corp.), internal company report.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2014): CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines, Canadian Institute of Mining, Metallurgy and Petroleum

Canadian Securities Administrators (CSA) (2019): National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators

Canarc Resource Corp. (1996 to 2019) News Releases

Canarc Resource Corp. (2018): Correspondence Records with TRTFM, internal company document

Canarc Resource Corp. (2018): Correspondence Records with EAO, internal company document

Caterpillar (2016): Caterpillar Performance Handbook, Edition 46, January 2016

Chamberlin Ken (2007): New Polaris Gold Mine Ltd. Surface Infrastructure Cost Estimate Internal company report

Dentons Canada LLP (2023): 'CCM Mineral Title Opinion Dentons 20221213.pdf', inter-company memo

Dewonck, B., Brucciani, P. (1991): Geological, Geochemical and Geophysical Report on Suntac Minerals Corporations Polaris-Taku Project, 1990 Surface Exploration Program, Orequest Consultants Ltd., internal company report

Gartner Lee Limited (1997): Summary report – Environmental and Site Development Update – Golden Angus Mines, internal company report

Gartner Lee Limited (2007): New Polaris Project Technical Report, internal company report

Gartner Lee Limited (2007): Regulatory Risk Management and Gap Analysis Report for the New Polaris Project, internal company report

Giroux, G.H. (1991): 1991 Geostatistical Study of the Geological Resource contained within the Polaris-Taku Gold Deposit for Suntac Minerals Corporation; amended Feb 1995; internal company report

Giroux, G.H. (1991): Geostatistical Study of the Geological Resource contained within the Polaris-Taku Gold Deposit, Montgomery Consultants Ltd.

Giroux, G.H, (1995): Update of Resource Estimate for Polaris-Taku "C" Vein System and NI Vein System (1995)

Giroux G. H. and Morris R. J. (2007): Resource Potential New Polaris Project Technical Report

Giroux G.H., Gray J.H. and Morris R.J. (2007): New Polaris Project - Preliminary Assessment

Giroux G.H., Gray J.H. and Morris R.J. (2009): New Polaris Project - Preliminary Assessment

Giroux G.H., Gray J.H. and Morris R.J. (2011): New Polaris Project – Preliminary Assessment Update

Harrison, I and Moors, J.G. (1995): Report on the 1995 North Grid Trenching and Diamond Drilling Programs

Hatch Ltd. (2018): Canarc Resources Corporation, New Polaris Project – Sulphide Flotation Concentrate Pressure Oxidation Assessment, Scoping Study Final Report, Internal company report

Hill, A.R. (1992): Structural Analysis and Discussion of Exploration Targets, Polaris-Taku Project; Suntac Minerals Corporation, internal company report

Hodgson, C.J., (1993): Mesothermal lode-gold deposits, in Kirkham, R.V., Sinclair, W.D., Thorpe, R.I. and Duke, J.M., eds., Mineral Deposit Modeling: GAC Special Paper 40

Kerr, F.A. (1948): Taku River area, B.C.; Geological Survey of Canada, Memoir 248

Lewis, P.D. (1994): Polaris-Taku Project, North Grid Mapping and Structural Analysis - Preliminary Report; Golden Angus Mines Ltd. (Canarc Resource Corp.), internal company report

Makarenko, Doerksen, Klingmann, McLeod, Arseneau, Marsland, McLeod, West, Palkovits, Krys (2014): Feasibility Study Technical report, Tulsequah Chief Project, Northern British Columbia, Canada, Chieftain Metals Corp.

Marriott, C.C. (1991): Report on the 1991 Diamond Drilling Program and Geological Review of the Polaris-Taku Gold Property; Suntac Minerals Corporation, internal company report

Marriott, C.C. (1992): Report on the 1992 Diamond Drilling Program of the Polaris-Taku Gold Property; Suntac Minerals Corporation, internal company report

McClintock, J.A. (2006): Report on the 2003 to 2005 Exploration Program on the New Polaris Mine Site

McKay D.L. (1996): Technical and Economic Review of the Polaris-Taku Project.

McKay D.L. (1997): Update Report on the New Polaris Project

Ministry of Energy, Mines and Petroleum Resources, Mining and Minerals Division (2008): Health, Safety and Reclamation Code for Mines in British Columbia

Monger, J. W. H. (1977): Upper Pa1aeozoic rocks of the western Canadian Cordillera and their bearing on Cordilleran evolution; Canadian Journal of Earth Science, V. 14, pp. 1832-1859

Monger, J. W. H. (1984): Cordilleran Tectonics: a Canadian Perspective; Bull. Soc. Geol. France, 1984(7) t. XXVI, N. 2, pp. 255-278

Moors, J.G. (1995): Report on the 1994 Exploration and Diamond Drilling Programs

Moors, J.G. (1995): Report on the 1995 Mine Wedge Deep Drilling Program

Moose Mountain Technical Services (2019): 2019 The New Polaris Gold Project British Columbia, Canada – 2019 Preliminary Economic Assessment

Mountjoy, Keith, MASc, PGeo. (2007): Ongoing ARD/ML Kinetic Testing by URS Canada Inc. – Internal company progress reports of ongoing test work

Neale, J. and Gericke, M. (2003): Diagnostic and Bioleach Testing of a Sample of Refractory Gold Concentrate from the New Polaris Project, Located in Northwestern British Columbia, Canada. Mintek Biotechnology Division.

Outotec RSA (PTY) LTD (2018): BIOX Batch Amenability Test Work Report on the New Polaris Sample, Internal Company Report

Outotec RSA (PTY) LTD (2019): Capital and Operating Cost Estimations for the New Polaris Preliminary Economic Assessment, Internal Company Report

Oxidor Corporation (2004): Biooxidation Amenability Testing of New Polaris Concentrate, Internal Company Memo

Palmer, Rick, M.Sc., R.P. Bio, 2011: Work Plan for the Environmental Assessment Certificate Application, New Polaris Project, BC, Palmer Environmental Consulting Group Inc.

Palmer, Rick, M.Sc., R.P. Bio, 2015: Work Plan for the Environmental Assessment Certificate Application, New Polaris Project, BC, Palmer Environmental Consulting Group Inc.

Palmer Environmental Group and Morrison Hershfield (2015): New Polaris Mine Draft Project Description, internal company report

Parliament, H. (1949): The Geology and Ore Deposits of Polaris-Taku Mine with Particular Reference to the Lower Levels; Theses, B.C. Professional Engineers

Process Research Associated Ltd. (PRA). (2007): Metallurgical Testwork on Gold-Bearing Sulphide Sample Originating from the New Polaris Project, internal company report

Process Research Associates (PRA). (2007): Flotation Test Metallurgical Balance, Internal Company Report

Province of British Columbia, Taku River Tlingit First Nation (2011): Wóoshtin wudidaa, Atlin Taku, Land Use Plan, July 2011

Resource Development Inc. (2002): Review of Historical Metallurgy Data from Polaris Taku Project, internal company memo

Resource Development Inc. (2003): Metallurgical Testing of Polaris Taku Sample, internal company memo

Resource Development Inc. (2004): Pressure Oxidation Testing of Polaris Taku Flotation Concentrates, internal company memo

Rhys, D.A. (1992): Structural Study of the Polaris-Taku Mine and Potential Exploration Applications; internal company report

Smith, A. (1938): Report on the Surface Geology of the Polaris-Taku Mine and Adjacent Area, Tulsequah, B.C.; internal company report

Smith, A. (1948): Taku River Area; In C.I.M. Symposium Volume, Structural Control of Canadian Ore Deposits

Smith, A. (1950): Report on Polaris-Taku Mine, Geology and Exploratory Development Program; Taku River Gold Mines Ltd., internal company report

Souther, J. G., and Armstrong, J. E. (1966): North Central Belt of the Cordil1era of British Columbia, in Tectonic History and Mineral Deposits of the Western Cordillera; Canadian Institute of Mining and Metallurgy, Special Vol. 8, pp. 171-184

Souther, J. G. (1969): Tulsequah and Juneau, Cassiar District, British Columbia; Geological Survey of Canada, Map 1262A

Souther, J. G., and Symons, D. T. A. (1974): Stratigraphy and Palaeomagnetism of the Mount Edziza volcanic complex, northwestern British Columbia; Geological Survey of Canada Paper 73-32, pp. 48

URS Canada Inc. (2007a): Geochemical Assessment Underground Exploration Development, New Polaris Project, Atlin Mining Division, British Columbia, internal company report

URS Canada Inc. (2007b): Phase 1 ARD/ML Assessment, New Polaris Project, Atlin Mining Division, British Columbia, internal company report

URS Canada Inc. (2007c): Phase 2 Geochemical Assessment, Vein Sample Static Tests, New Polaris Project, Atlin Mining Division, British Columbia, internal company report

Watts, Griffis, and McOuat Ltd (1992): A Report on the Polaris-Taku Gold Project for Canarc Resource Corp., internal company report

Watkinson, D., (1997): Technical Overview

Walton, G.J., (2002): Report on the 1996 and 1997 Exploration program on the New Polaris Mine site, Northwestern British Columbia

Yee, Jasmin, (2007): Metallurgical Report, Canarc Resource Corp. New Polaris Project Atlin, BC

Yee, Jasmin, (2007): Metallurgical Testwork on Gold Bearing Sulphide Sample Originating from the New Polaris Project