Eskay Creek Project

NI 43-101 Technical Report and Prefeasibility Study

British Columbia, Canada

Effective Date: 22 July, 2021

Prepared for: Skeena Resources Ltd.

Prepared by: Ausenco Engineering Canada Inc.

List of Qualified Persons: • Tommaso Roberto Raponi, P. Eng., Ausenco • Scott Elfin, P.E., Ausenco • Sheila Ulansky, P. Geo., SRK Consulting • Rolf Schmitt, P. Geo., ERM • Adrian Dance, P. Eng., SRK Consulting • Willie Hamilton, P. Eng., AGP Mining Consultants • Roland Tosney, P. Eng., AGP Mining Consultants

CERTIFICATE OF QUALIFIED PERSON

Adrian Dance, P. Eng.

I, Adrian Dance, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Principal Metallurgist, with SRK Consulting, with an office at 1066 W Hastings St, Vancouver, BC V6E 3X2.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").
3. I am a graduate of the University of British Columbia in 1987 where I obtained a Bachelor of Applied Science and a graduate of the University of Queensland in 1992 where I obtained a Doctorate.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 37151.
5. I have practiced my profession for 29 years since graduation including 19 years as a consultant and have experience working in a number of gold operations around the world.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Eskay Creek property.
8. I am responsible for sections 1.1, 1.2, 1.9, 1.9.1, 1.9.2, 1.24, 1.25, 2.2, 2.3, 2.4, 2.5, 2.6, 2.8, 2.9, 3, 13, 26, and 27 of the Technical Report.
9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.
10. I have been involved with the Eskay Creek property since 2019 during the preparation of the Preliminary Economic Assessment report as well as supervised metallurgical testwork programs in 2019 and 2020.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 1 September 2021

"Signed and Sealed"

Adrian Dance, P. Eng.
APEGBC # 37151

CERTIFICATE OF QUALIFIED PERSON

Tommaso Roberto Raponi, P. Eng.

I, Tommaso Roberto Raponi, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Principal Metallurgist, with Ausenco Engineering Canada Inc. (Canada), with an office at Suite 1550 11 King St West, Toronto, ONT M5H 4C7.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").
3. I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering, with specialization in Mineral Processing in 1984.
4. I am a Professional Engineer registered with the Professional Engineers Ontario (No. 90225970), Engineers and Geoscientists British Columbia (No. 23536) and NWT and Nunavut Association of Professional Engineers and Geoscientists (No. L4508).
5. I have practiced my profession continuously since 1984 with experience in the development, design, operation, and commissioning of mineral processing plants, focusing on gold projects, both domestic and internationally. A summary of the more recent portion of my professional career is as follows
   - TR Raponi Consulting Ltd, Independent Consultant 2016-2021.
   - Centerra Gold Inc., Director of Metallurgy 2005-2016.
   - SNC-Lavalin Inc., Senior Metallurgist, 1995-2005.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I have not made a site visit to the *Eskay Creek Project*.
8. I am responsible for sections 1.1, 1.2, 1.3, 1.6, 1.15, 1.16.1,.16.2, 1.16.41.16.2, 1.16.4, 1.16.5, 1.18, 1.19, 1.19.1, 1.19.3, 1.19.5, 1.19.6, 1.20, 1.20.1, 1.20.3, 1.20.4, 1.20.5, 1.21, 1.22.3, 1.23, 1.23.4, 1.23.7, 1.24, 1.25, 2.1, 2.2, 2.3, 2.4, 2.5, 2.6, 2.7, 2.8, 2.9, 3, 6.2, 17, 18.1, 18.2, 18.7-18.10, 19, 21.1, 21.2, 21.2.1, 21.2.7-21.2.11, 21.3, 21.3.1-21.3.8, 22, 24, 25.1, 25.9, 25.12–25.17, 26, and 27 of the Technical Report.
9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.
10. I have had no previous involvement with the Eskay Creek Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 1 September 2021

"Signed and Sealed"

Tommaso Roberto Raponi, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Roland Tosney, P. Eng.

I, Roland Tosney, P. Eng., certify that:

1. I am a Professional Engineer, currently contracted as a Senior Mining Geotechnical Engineer, with AGP Mining Consultants Inc. (Canada), with an office at 132 Commerce Park Drive, Unit K #246, Barrie, ON, L4N 0Z7.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").
3. I graduated from the University of Saskatchewan with a Bachelor of Engineering degree in Geological Engineering in 1998, and a Master of Science Degree in Mining Geotechnical Engineering in 2001.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 29393.
5. I have practiced my profession continuously since 1998 and have been involved with operations and consulting at open-pit and underground mines in Canada and the United States, South America, and Africa. I have acquired expertise in mining geotechnical assessments, evaluations and safety and operational improvements.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I visited the *Eskay Creek Project* on July 18 to 22, 2020 for a visit duration of *five days,* and October 27, 2020, for a visit duration of *one day*.
8. I am responsible for section 1.1, 1.2, 1.14.1, 1.14.2, 1.24, 1.25, 2.2, 2.3, 2.4, 2.6, 2.8, 2.9, 3, 16.3, 16.3.1, 16.3.2, 26, and 27 of the Technical Report.
9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.
10. I have had no previous involvement with the Eskay Creek Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 1 September 2021

"Signed and Sealed"

Roland Tosney, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Rolf Schmitt, P. Geo.

I, Rolf Schmitt, P. Geo., certify that:

1. I am a Professional Geologist, currently employed as Technical Director, ERM, with an office at 1111 West Hastings St, Vancouver, BC V6E 2J3.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study,"* (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").
3. I graduated from the University of British Columbia with a Bachelor of Geology (Hons) in 1977, University of British Columbia with a Master of Science (Regional Resource Planning) in 1985, and University of Ottawa with a Master of Science Geology (1993), with specialization in exploration geochemistry.
4. I am a Professional Geologist, registered with Engineers and Geosciences of British Columbia, member number 121446.
5. I have practiced my profession continuously since 1977 and have been involved in: mineral exploration for porphyry copper-gold and VMS deposits in British Columbia, exploration geochemical surveys across Canada, regional land use policy and planning and mining regulatory development in British Columbia, in environmental assessment and permitting of mines in British Columbia, and ESG due diligence of base and precious metal projects in numerous countries throughout North and South America.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I conducted an aerial overflight of the *Eskay Creek Project* on July 19, 2019.
8. I am responsible for sections 1.1, 1.2, 1.4, 1.17, 1.17.1, 1.17.2, 1.17.3, 1.22.5, 1.23.6, 1.24, 1.25, 2.2, 2.3, 2.4, 2.6, 2.8, 2.9, 3, 4.4, 4.5, 4.7, 4.8, 4.9, 5, 20, 23, 25.16.2.6, 26, and 27 of the technical report.
9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.
10. I have had no previous involvement with the Eskay Creek Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 1 September 2021

"Signed and Sealed"

Rolf Schmitt, M. Sc., P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Scott Elfen, P.E.

I, Scott Elfen, P.E., certify that:

1. I am a Professional Engineer, currently employed as Global Lead Geotechnical Service, with Ausenco Engineering Canada Inc. (Canada), with an office at 855 Homer St, Vancouver, BC V6B 2W2.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").
3. I graduated from the University of California Davis with a Bachelor of Civil Engineering degree, with specialization in geotechnical and Hydrology in 1991.
4. I am a Registered Civil Engineer in the State of California (No. C56527) by exam since 1996 and am also a member of the American Society of Civil Engineers (ASCE), Society for Mining, Metallurgy & Exploration (SME) that are all in good standing.
5. I have practiced my profession continuously since 1990 and have been involved in: mineral processing and metallurgical testing, metallurgical process plant design and engineering, and metallurgical project evaluations for precious metal, including gold and silver, and base metal projects in Canada and numerous other countries.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I visited the *Eskay Creek Project* on June 19, 2019, October 27 and 28, 2020 and July 14 and 15, 2021.
8. I am responsible for sections 1.1, 1.2, 1.16, 1.16.3, 1.19.4, 1.22.4, 1.23.5, 1.24, 1.25, 2.2, 2.3, 2.4, 2.6, 2.8, 2.9, 3, 16.4, 16.4.1, 16.4.2, 16.4.3, 18.5-18.6, 25.16.2.7, 26, and 27 of the Technical Report.
9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.
10. I have had no previous involvement with the Eskay Creek Project.
11. I have read the NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.
12. As of the Effective Date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 1 September 2021

"Signed and Sealed"

Scott Elfen, P.E.

CERTIFICATE OF QUALIFIED PERSON
Sheila Ulansky, P. Geo.

I, Sheila Ulansky, P. Geo., certify that:

1. I am a Professional Geologist, currently employed as Senior Resource Consultant, with SRK Consulting, with an office at Suite 2200 - 1066 W Hastings St, Vancouver, BC V6E 3X2.

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").

3. I am a graduate of the University of Victoria, British Columbia in 2007 where I obtained a Bachelor of Science degree in Geology. In 2019 I obtained a Master of Science degree in Geology from Laurentian University, Ontario.

4. I am a Professional Geologist, registered with Engineers and Geosciences of British Columbia, member number 36085.

5. I have practiced my profession continuously since 2007, initially in exploration geology on a variety of deposit types. Since 2012, I have worked full time as a Resource Geologist with emphasis on QA/QC, exploratory data analysis, variography, 3D geological modelling, and resource estimation. I have worked on a number of gold, silver, and base metal deposit types, including Volcanogenic massive sulphide ore deposits, narrow vein in Orogenic systems, Carlin-style mineralization, and epithermal gold mineral systems; experience which is relevant to the Eskay Creek scope of work.

6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.

7. I visited the *Eskay Creek Project* on two occasions: between June 27 and 28, 2018, and July 27 and 30, 2020.

8. I am responsible for sections 1.1, 1.2, 1.5, 1.7, 1.8, 1.10, 1.11, 1.22.1, 1.23.1, 1.23.2, 1.24, 1.25, 2.2, 2.3, 2.4, 2.6, 2.8, 2.9, 3, 4.2, 6.1, 7, 8, 9, 10, 11, 12, 14, 25.16.2.1- 25.16.2.3, 26, and 27 of the Technical Report.

9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.

10. I have had no previous involvement with the Eskay Creek Project.

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

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

Dated: 1 September 2021

"Signed and Sealed"

Sheila Ulansky, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

Willie Hamilton, P. Eng.

I, Willie Hamilton, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Principal Mine Engineer, with AGP Mining Consultants Inc. (Canada), with an office at 132 Commerce Park Drive, Unit K #246, Barrie, ON, L4N 0Z7.

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Prefeasibility Study"*, (the "**Technical Report**"), that has an effective date of 22 July 2021, (the "**Effective Date**").

3. I graduated from the University of Alberta with a Bachelor of Science degree in Mining Engineering in 1988 and a Master of Science in Mining Engineering in 1990.

4. I am a Professional Engineer, registered with the Association of Professional Engineers and Geoscientists of Alberta, member number 47481.

5. I have practiced my profession continuously since 1990 and have been involved with operations and consulting at open-pit and underground, hard and soft-rock mines in Canada and the United States. With expertise in numerous mine planning, scheduling, and pit optimization software, as well as significant project evaluation work for all sizes of studies.

6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.

7. I visited the *Eskay Creek Project* on August 21-22, 2019, for a visit duration of two days.

8. I am responsible for sections 1.1, 1.2, 1.12, 1.13, 1.14, 1.14.3, 1.16.1, 1.19.2, 1.20.2, 1.23.1.2, 1.23.2.3, 1.24, 2.2, 2.3, 2.4, 2.6, 2.8, 2.9, 3, 15, 16.1, 16.2, 16.5-16.15, 18.3, 18.4, 21.2.2-21.2.6, 25.7, 25.8.3, 25.16.1.3, 25.16.2.4, 26, and 27 of the Technical Report.

9. I am independent of Skeena Resources Ltd. ("Skeena") as independence is described by Section 1.5 of the NI 43–101.

10. I have been involved with the Eskay Creek property since 2019, during preparation of both the Preliminary Economic Assessment and Prefeasibility reports.

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

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

Dated: 1 September 2021

"Signed and Sealed"

Willie Hamilton, P. Eng.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Skeena Resources Ltd. (Skeena) by Ausenco Engineering Canada Inc. (Ausenco), SRK Consulting Inc., AGP Mining Consultants, and ERM, collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Skeena, subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco), Hemmera Envirochem Inc., an Ausenco company (Ausenco), SRK Consulting (Canada) Inc. (SRK), and AGP Mining Consultants Inc. (AGP), prepared a technical report (the Report) on the results of a pre-feasibility study (2021 PFS) for Skeena Resources Limited (Skeena) on the volcanogenic massive sulphide (VMS) Eskay Creek Project (the Project) located in British Columbia.

Skeena wholly owns the Eskay Creek gold–silver project.

The Project hosts the previously-mined Eskay Creek deposit, which was in operation as an underground mine from 1995–2008.

1.2 Terms of Reference

The Report supports disclosures by Skeena in a news release dated 22 July, 2021 entitled, "Skeena Completes PFS for Eskay Creek: After-Tax NPV(5%) of C$1.4B, 56% IRR and 1.4 Year Payback".

All measurement units used in this Report are metric unless otherwise noted. Currency is expressed in Canadian (C) dollars (C$). The Report uses Canadian English. United States dollars, where referenced, are termed US$.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; the 2019 CIM Best Practice Guidelines).

As the ownership of, and ownership interests in, historical mining operations has changed hands numerous times during the production history, the Report uses the term "previous operator" to refer to work done from 1988 to 2017. The term "legacy" is used for data generated by the previous operator.

1.3 Project Setting

The Eskay Creek Project is located in the Golden Triangle region of British Columbia, Canada, 83 km northwest of Stewart. Support services for mining and other resource sector industries in the region are provided primarily by the communities of Smithers (pop. 5,400) and Terrace (pop. 11,500). Both communities are accessible by commercial airlines with daily flights to and from Vancouver.

Access to the Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction. The Eskay Mine Road is a 54.5 km private industrial road that is operated by Coast Mountain Hydro Corp. (0 km to 43.5 km) and Skeena (43.5 km to 54.5 km). There are two nearby gravel air strips: Bronson Strip which is about 40 km west of the mine site and Bob Quinn, roughly 37 km northeast of the Project.

The mean annual total precipitation at the former mine site is estimated to be 2,500 ± 500 mm. About 55–71% of precipitation falls as snow. The average temperature range is from -10.4°C in January to +15°C in July. Exploration activities can be curtailed by winter conditions. The previous mining operation was conducted on a year-round basis, and it is expected that any future open pit operations will also be year-round.

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (amsl), located on the eastern flank of the Boundary Ranges. The plateau is characterized by northeast-trending ridges with gently sloping meadows occupying valleys between the ridges. Relief over the plateau area ranges from 500 m in the existing Tom MacKay tailings storage facility (TMSF) area to over 1,000 m in the Unuk River and Ketchum Creek valleys. The plateau is drained by tributaries of the Stikine–Iskut and Unuk Rivers. The former Eskay Creek mine site is at approximately 800 m elevation. Mountain slopes are heavily forested. There are no known federal, provincial, or regional parks, wilderness or conservancy areas, ecological reserves, or recreational areas near the Project.

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

On December 18, 2017, Skeena and Barrick entered into an Option Agreement on the Eskay Creek Project. This agreement affects all mineral claims and mineral leases that comprise the Eskay Creek Project, except for the single mineral claim registered to Skeena Resources Ltd. On October 5, 2020, Skeena and Barrick agreed to amend the terms of the original option agreement on the Eskay Creek Project. Skeena acquired 100% ownership of Eskay Creek in October 2020 in consideration for:

- The issuance to Barrick of 22.5 million units, consisting of one common share of Skeena and a non-transferable half warrant;
- The grant of a 1% net smelter return (NSR) royalty on the entire Eskay Creek land package. Half of that royalty may be purchased from Barrick during the 24-month period after closing, at a cost of C$17.5 million;
- A contingent payment, payable if Skeena sells more than a 50% interest in Eskay Creek during the 24-month period after closing, of C$15 million.

The Eskay Creek Project covers 5,745.49 ha, consisting of 47 mineral claims (3,915.23 ha), and eight mineral leases (1,830.26 ha). Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under BC regulations. All statutory annual reporting obligations have been met.

Royalties are payable on a number of the claims including a 1% NSR payable to Euro-Nevada Mining Corporation Limited (now Franco-Nevada Corp.); a 2% NSR payable to ARC Resource Group Ltd. (Option Agreement dated 4 November 1988 between ARC Resource Group Ltd. and Canarc Resources Corp.), a 2% NSR payable to ARC Resource Group Ltd. (Royalty Deed dated 1 August 1990 between Adrian Resources Ltd. and ARC Resource Group Ltd.), and a 1% NSR payable to David A. Javorsky. There is also a 1% royalty payable to Barrick on all of the claims, which is in addition to the existing royalties.

Skeena holds an interest in two surface leases and the Eskay Road access. Skeena will need to acquire surface rights in support of any future mining operations. A permit amendment will be required for one of the surface licences to extend the boundary to include the surface area associated with the south end of TMSF. Two water rights are currently held. Skeena anticipates needing to apply for additional Water Licences under the BC Water Sustainability Act for the proposed Project.

Skeena's current environmental liabilities are related to activities undertaken by Skeena, and activities arising from permitting. The key liabilities would be remediation of drill pads and drill access roads. Skeena has posted an environmental bond with the relevant BC authorities in relation to the work programs that have been conducted.

1.5 Geology and Mineralization

The Eskay Creek deposit is generally classified as an example of a high-grade, precious metals-rich epithermal volcanogenic massive sulphide (VMS) deposit; however, it has also been suggested to be an example of a subaqueous hot spring gold–silver deposit.

The Eskay Creek Project is located along the western margin of the Stikine Terrane, within the Intermontane Tectonic Belt of the Northern Cordillera. It is hosted within the Jurassic rocks of the Stikinia Assemblage at the stratigraphic transition from volcanic rocks of the uppermost Hazelton Group to the marine sediments of the Bowser Lake Group.

The Project area is underlain by volcanic and sedimentary rocks of the regionally extensive Lower to Middle Jurassic Hazelton Group. The Hazelton Group can be further subdivided into the Jack, Betty Creek, Spatsizi, Iskut River, Mt. Dilworth and Quock Formations (arranged from oldest to youngest). The stratigraphy in the immediate area of the property consists of an upright succession of andesite, marine sediments, intermediate to felsic volcaniclastic rocks, rhyolite, contact mudstone (host to the main Eskay Creek deposits), and basaltic/andesitic sills and flows. This sequence is overlain by mudstones and conglomerates of the Bowser Lake Group. These rocks are folded into a gently, northeast-plunging fold, the Eskay Anticline, and are cut by north-, northwest- and northeast-trending fault structures.

Regional metamorphic grade in the area is lower greenschist facies. Alteration in the footwall volcanic units is characterized by a combination of pervasive quartz–sericite–pyrite, potassium feldspar, chlorite and silica. Intense alteration zones are locally associated with sulphide veins that contain pyrite, sphalerite, galena, and chalcopyrite. An intense, tabular-shaped blanket of chlorite–sericite alteration, up to 20 m thick, occurs in the Eskay Rhyolite member, immediately below the contact with the main stratiform sulphide mineralization.

Several styles of stratiform and discordant mineralization are present at the Eskay Creek Project, defined over an area approximately 1,400 m long and as much as 300 m wide. Distinct zones have been defined by variations in location, mineralogy, texture, and precious metal grades.

Stratiform-style mineralization is hosted in black carbonaceous mudstone and sericitic tuffaceous mudstone of the Contact Mudstone (Iskut River Formation), located between the footwall Eskay Rhyolite member and the hanging wall Willow Ridge andesite unit. The stratiform hosted zones include the 21B Zone, the NEX Zone, the 21A Zone (characterized by arsenic–antimony–mercury sulphides), the 21C Zone, the 21Be Zone and the 21E Zone. Stratigraphically above the Contact Mudstone, and usually above the first basaltic sill, the mudstones also host a localized body of base metal-rich, relatively precious metal-poor, massive sulphides referred to as the Hanging Wall or HW Zone.

Stockwork and discordant style mineralization at Eskay Creek is hosted in the rhyolite footwall within the PMP Zone, the 109 Zone, the 21A Zone the 21B Zone, the 21C Zone, the 21E Zone, the NEX Zone and 22 Zone. The PMP Zone is characterized by pyrite, sphalerite, galena, and chalcopyrite-rich veins and veinlets hosted in strongly sericitized and chloritized rhyolite. The 109 Zone consists of gold-rich quartz veins with sphalerite, galena, pyrite, and chalcopyrite associated with abundant carbonaceous material hosted predominantly in siliceous rhyolite. The 21A, 21B, 21C, NEX and 21E Zones consist of very fine-grained cryptic pyrite with rare sphalerite and galena in sericitized rhyolite. The 22 Zone consists of cross-cutting arsenopyrite, stibnite and tetrahedrite veins hosted in massive to pyroclastic facies rhyolite.

There is significant remaining exploration potential in the Eskay Creek deposit and environs. Exploration targets include syn-volcanic feeder structures at depth and along strike; mineralization hosted within the largely unexplored Lower Mudstone horizon; and the in the vicinity of the 22 Zone, which remains open along strike and at depth. Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

1.6 History

The Project area has a long exploration history, dating back to initial prospecting activities in 1932. Companies with Project interests prior to Skeena's involvement include Premier Gold Mining Co. Ltd., MacKay Gold Mines Ltd., Canadian Exploration Ltd., American Standard Mines Ltd., Pioneer Gold Mines of B.C. Ltd., New York-Alaska Gold Dredging Corp., Western Resources Ltd., Stikine Silver Ltd., Canex Aerial Exploration Ltd., Mount Washington Copper Co., Newmont Mining Corp., Kalco Valley Mines Ltd., Texas gulf Canada Ltd., May-Ralph Resources Ltd., Ryan Exploration Ltd. (U.S. Borax), Kerrisdale Resources Ltd., Consolidated Stikine Silver Ltd., International Corona Corp., Homestake Canada Inc., and Barrick Gold Inc. Work conducted during this period included prospecting, geological mapping and reconnaissance, rock, stream, sediment, and soil geochemical sampling, trenching, surface geophysical surveys (electromagnetic (EM), very low frequency (VLF), ground magnetic/VLF-EM, induced polarization (IP), seismic refraction, University of Toronto electro-magnetic system (UTEM)), borehole geophysics (frequency domain EM (FEM)) core drilling, exploration adit and underground development, petrography, and mining studies.

Underground mining operations were conducted from 1994 to 2008. From 1994–1997, ore was direct-shipped after blending and primary crushing. From 1997 to closure in 2008, ore was milled on site to produce a shipping concentrate.

Skeena has completed core drilling, an airborne light detection and ranging (LiDAR) and photo acquisition survey, Mineral Resource estimation, metallurgical testwork, environmental testwork and supporting studies, and preliminary technical studies.

1.7 Drilling and Sampling

Data collected prior to Skeena's Project interest is referred to as legacy data. Legacy drilling consists of 1,522 surface core drill holes totalling 342,119 m, drilled from 1932 until 2004. Since 2018, Skeena has drilled 751 surface drill holes totalling 104,740 m.

The underground areas are drilled at an average spacing of 10 m using BGM (~40 mm) core diameters. In highly complex areas where mining was active, drill spacing was locally reduced to 5 m. Underground drill holes are generally less than 100 m in length.

Limited information is available for procedures used during the exploration programs carried out before 2004. The drill core was logged using DLOG computer programs for data entry as well as for drill log printing. Information collected included lithology, mineralization, textural descriptions, rock colour, structure, core recovery, and rock quality designation (RQD). Skeena currently does not have access to the legacy RQD and recovery data. Underground collar location surveys were performed by the mine surveyors. These provided accurate collar locations for the holes, and a check on the initial azimuth and dip was recorded for each drill hole. Prior to 2004, most of the underground drill holes in the database were surveyed downhole using a Sperry Sun Single Shot instrument, with readings taken every 60 m, or by acid tubes, with readings every 30 m. In early 2004, downhole surveying used an Icefield Tools M13 instrument. This provided azimuths and dips for each hole every 3 m down the hole. Readings were reviewed by staff and inaccurate entries were removed from the database. All collar and survey information were tabulated in master files within the DLOG computer program. Completed logs were printed and the information was exported into ACAD and Vulcan software to facilitate plotting drill hole location maps and cross-sections.

During the Skeena drill program, core was geologically logged for lithology, alteration, veining, mineralization, and structural features. Geotechnical data such as recovery, RQD, longest stick, and magnetic susceptibility were recorded. Skeena recorded geological and geotechnical information into a GeoSpark database. Core was photographed wet. Surface drill hole collars were initially located using hand-held global positioning system (GPS) units and surveyed at the end of the drill program using a Trimble differential GPS (DGPS). Down hole orientation surveys for surface drill holes were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool.

Drill hole spacing throughout the deposit varies from 5 m, where underground production drilling encountered complex areas, to 25 m at the surface. The average drill hole spacing is approximately 10–15 m throughout the deposit. For surface drill holes, mineralization true width approximates 70–100% of drilled width; for underground drill holes positioned on single platforms and drilled in radiating fans, true drilling widths are more variable.

Historically, sampling at Eskay Creek was selective and primarily based on visual estimations of sulphide percent. All sample intervals sent to the laboratory were tested for gold and silver; however, lead, copper, zinc, mercury, antimony, and arsenic were inconsistently sampled from one drilling campaign to the next. For underground drilling, lead, copper, zinc, mercury, antimony, and arsenic were assayed when samples exceeded 8 g/t gold equivalent (AuEq, where AuEq equaled Au + (Ag/68)). Legacy sampling intervals were variable. Prior to 2003, sample intervals varied from about 0.25 m up to 1.5 m though the optimum sample interval was 1.0 m. Sample intervals were always contained within one geological unit and did not straddle contacts. During 2004, sample intervals were typically on 1 m intervals, but smaller increments were applied where necessary to honour geological contacts.

During Skeena’s drill programs, 1 m assay intervals were established when visible mineralization was first observed, and then uniform intervals were continued down the drill length until there is no evidence of mineralization. Assay intervals honoured geological contacts to a minimum of 0.5 m and a maximum of 1.5 m.

Specific gravity (SG) measurements collected during legacy programs were collected from drill core in 1996 (250 measurements from 20 drill holes) and 1997 (84 measurements from seven drill holes), using the water displacement method. SG models were subsequently created using a formula that was experimentally derived based on comparisons between actual measurements and analyses. The following formula was used:

- SG = (Pb + Zn + Cu) x 0.03491 + 2.67;

where all metals are reported in percent.

A default SG value of 2.67 was applied to samples for which base metals were not reported. This is the average value of unmineralized rhyolite and mudstone host rocks combined. The measured SG values from the early drill programs were primarily from relatively low base metal, 21B-style mineralization. The formula is therefore likely biased on the low side for rocks with higher base metal content. During the Skeena programs, SG samples were collected one in every 20 m down the hole and measured using the water displacement method.

Laboratories used for sample preparation and analysis during legacy programs, where known, include: Independent Plasma Laboratories (IPL; independent, accreditations not known) and the Eskay Mine laboratory (not independent, not accredited). Skeena used the ALS sample preparation facility in Kamloops (ALS Kamloops), which is independent and accredited. Analysis was completed at the ALS facility in Vancouver (ALS Vancouver), which holds ISO17025 accreditation for selected analytical methods. Both laboratories are independent of Skeena. SGS Canada, located in Burnaby, BC (SGS), was used to independently test pulp duplicates and a select number of standards. SGS holds ISO 17025 accreditations for selected analytical techniques. SGS is independent of Skeena.

Legacy sample preparation and analytical methods included:

- IPL: crushed to -10 mesh, riffle split and 250 g pulverized to -15 mesh. Gold was assayed by fire assay (30 g) with an atomic absorption (AA) finish. All gold values >1.00 g/t were re-assayed by fire assay (30 g) and finished gravimetrically. Silver was assayed by fire assay (30 g) with an AA finish. Analysis for lead, zinc, copper, arsenic, and antimony was done by an ore grade assay method using AA. Mercury analysis consisted of an aqua regia digestion and inductively-coupled plasma (ICP) finish;

- Eskay Mine laboratory: jaw-crushed to -⅛ inch, riffle split and pulverisation of 250–300 g. Gold was assayed by fire assay (10 g) with an AA finish. For analysis for zinc, antimony, copper, and lead, a 0.20g sample was digested in a heated solution of tartaric, nitric, perchloric and hydrochloric acids, and finished by AA. For mercury and arsenic, a 1.00 g sample was digested in a heated solution of nitric, perchloric and hydrochloric acids and finished by AA.

During the Skeena programs, all samples were initially sent and prepared at ALS Kamloops after which the pulp samples were split and shipped for analysis to ALS Vancouver. Sample preparation involved crushing to better than 70% passing 2 mm 10 mesh screen and pulverizing to better than 85% passing a 75 µm 200 mesh screen.

Gold assays were performed on 50 g samples by fire assay and atomic absorption (ALS code: Au-AA26) with a lower and upper detection limit of 0.01 g/t and 100 g/t, respectively. For assays above the upper detection limit then samples were analysed by fire assay with a gravimetric finish (ALS code: Au-GRA22) with lower and upper detection limits of 0.05 g/t and 10,000 g/t Au, respectively. Silver assays were performed on 50 g samples by fire assay and gravimetric finish (ALS code: Ag-GRA22) with lower and upper detection limits of 5 g/t and 10,000 g/t, respectively. For assays above the upper detection limit, a concentrate and bullion grade fire assay and gravimetric finish were performed (ALS code: Ag-CON01) with lower and upper detection limits of 0.7 g/t Ag and 995,000 g/t Ag, respectively.

Multi-element assays were performed using a combination of digest and finish methods: a 0.25 g sample using a four-acid digest followed by an ICP atomic emission spectroscopy (AES) finish (ALS code: ME-ICP61), and a 0.1 g sample using lithium borate fusion followed by an ICP-MS finish (ALS code: ME-MS81). This combination in assay methods for the multi-elements ensured that the range of concentrations for all elements of interest, particularly for antimony, were covered. In the Skeena database, the ICP-AES finish method took precedence. A limited number of samples exceeded the upper limits for silver, arsenic, copper, lead and zinc. For these samples, the laboratory was instructed to apply overlimit methods on a 0.4 g sample (ALS code: OG62) using a four-acid digest and ICP or AAS finish. Sulphur overlimits were re-analysed using the total sulphur Leco furnace method using a 0.1 g sample (ALS code: S-IR08) with a lower detection limit of 0.01% and upper detection limit of 50%. Mercury was separately analysed using low temperature aqua regia digestion followed by an ICP-AES finish (ALS code: Hg-ICP42) with a lower detection limit of 1 ppm and an upper detection limit of 100,000 ppm.

The Eskay Creek mine initiated quality assurance and quality control (QA/QC) measures into their sample stream in 1997. With progressive years the QA/QC protocol became more comprehensive and detailed. Prior to 2002, there was no formal QA/QC program in place; however, the Eskay Creek mine laboratory and IPL were regularly monitored using pulp duplicates. In 2003, the Eskay mine laboratory started to implement QA/QC procedures into the sampling process. Control blanks and standard reference materials (SRMs) were added to the sample stream. Acme inserted their own in-house SRMs, blanks and pulp repeats into the sample stream. Acme also routinely used preparation, pulp and reject duplicates. An official QA/QC program was undertaken in 2004 whereby the Eskay Creek exploration team added SRMs, blanks and field duplicates to the sample stream and submitted them to Acme for checking. Sample repeatability at Eskay Creek was closely monitored during the 2004 drilling campaign by the regular insertion of field duplicates into the sample stream. Field duplicates at the Eskay mine laboratory performed well with the duplicate sample set. An audit was conducted on the 2004 QA/QC results and procedures by Dr. Barry Smee, of Smee & Associates Consulting Ltd. The findings from the analysis identified a low bias in relation to Acme's internal SRMs for both aqua regia and fire assay methods. Acme corrected the inconsistencies with batch repeats. The sampling precision by means of using duplicate preparation and pulp samples was found to be within acceptable limits.

Skeena implemented a formal QA/QC program from the inception of their 2018 Phase 1 drilling program, consisting of blanks, duplicates and SRMs. SRMs and blanks were monitored when batches of assay data were first received. If analyses were outside of the acceptable range after checking for data entry errors, then repeat assay were requested. Where two or more consecutive SRMs were both biased high or low (more than 105% of the expected value or less than 95% of the expected value) repeat assays were requested. The laboratory was instructed to retrieve five pulp samples before and after the QC failure. Duplicate data were also monitored, with Skeena reporting any concerns to the laboratory manager.

1.8 Data Verification

SRK conducted an independent review of the historical database as well as the current database used for the 2018, 2019, 2020 Phase 1 and Phase 2 drilling programs. In addition, SRK reviewed the historical and current QA/QC programs and independently analysed the results from these programs. After the review, SRK concluded that the database was sufficiently reliable for resource estimation. The results of the QA/QC analysis indicate that the historical data are unbiased. A large number of assays in the database were validated against the original digital assay certificates. These assays ranged from the years 1999 to 2004, and less than 1% errors were found. In addition, the data analysed for the 2018, 2019, 2020 Phase 1 and Phase 2 drilling programs were collected and analysed in a systematic and unbiased manner. The data verification of this data did not identify any material issues and the QP is satisfied that the assay data are of suitable quality to be used as the basis for the resource estimate.

The QP conducted two site visits, during which time she reviewed surface and underground drill core to confirm the presence and nature of mineralization and appropriateness of the interpreted geological framework, observed abundant mineralization in drill core, verifying the presence, and nature of gold and silver mineralization at the Eskay Creek Project, and verified Skeena's drilling, sample preparation, handling, security, and chain of custody procedures, surface drill hole locations, and core logs.

1.9 Metallurgical Testwork

1.9.1 Legacy Processing and Testwork

The original operating plan was to construct the mining infrastructure at the mine site and transport ore to a processing facility located close to Placer Dome's Equity Silver mine, near Houston, B.C. In 1996, a testwork program was initiated at Process Research Associates with follow up locked-cycle testing at International Metallurgical and Environmental Inc. to evaluate the potential of a gravity/flotation process for upgrading ore from the NEX and 109 Zones into marketable concentrates. The work indicated that the mineralization could be economically upgraded to a saleable concentrate. In 1997, Prime completed the engineering and construction of a 150 t/d mill to concentrate the gold and silver values for the NEX and 109 Zones. Over the next several years, the mill was steadily upgraded and expanded to its final production capacity of 350 t/d. Since 2008, the mine area has been under a state of reclamation, care, and maintenance.

As part of a preliminary economic assessment conducted in 2019 (the 2019 PEA), testwork was completed by Blue Coast Research (Blue Coast) in Parksville BC, including comminution, whole ore leaching, gravity and flotation recovery methods. The process plant flowsheet assumed for the 2019 PEA included flotation recovery of a precious metal concentrate. Several issues were identified during the 2019 PEA testwork program associated with high or variable content of non-sulphide gangue (NSG) minerals such as muscovite, illite, chlorite, and silica. This resulted in extended flotation times due to slow kinetics as well as poor filtration properties of some of the final concentrate samples.

1.9.2 Current Testwork

In 2020, a comprehensive testwork program was completed by Base Metallurgical Laboratories Ltd. of Kamloops, B.C. (Base Met) focused on issues identified in the 2019 PEA and resulted in a modified process flowsheet. The Base Met program was completed on remaining 2019 PEA test sample material as well as several new drill core samples from the 2018–2020 drill campaigns. Tests included mineralogical analysis, open circuit rougher and rougher/cleaner flotation tests, locked cycle float tests, diagnostic leach and extended gravity-recoverable gold, gravity recovery followed by cleaner flotation, comminution (Bond ball mill work index, impact breakage, abrasion index, IsaMill signature plots) and settling, pressure/vacuum filtration.

For mine planning purposes, a number of recovery models were developed from the 2021 PFS testwork results. With the wider range of samples tested in the 2021 PFS program, different NSG mineral compositions were found to affect the final concentrate recovery vs. grade curves.

The recovery equations developed during the 2021 PFS are acceptable for use in the MRMR estimates and LOM plan used in financial modelling. These equations were applied to the LOM plan to generate estimates of impurities in a 45 g/t gold final concentrate. With higher-grade material processed in the first three years, arsenic, antimony, and mercury levels are expected to be elevated in the final concentrate, but not impact its saleability. After Year 3, these levels fall to below 1% As, 2% Sb and 1,000 g/t Hg. Sulphur levels are expected to be between 15% and 24% at this gold concentrate grade.

1.10 Mineral Resource Estimation

The Mineral Resource estimate is primarily based upon legacy drilling completed by the previous operator; however, additional holes drilled by Skeena in 2018 have been included. The database used in estimation contains 7,583 historical holes and 751 completed holes drilled by Skeena from 2018 to January 2021.

During 2020, the litho-structural model was updated to include six additional lithological units that were previously merged within the nearest stratigraphic package, namely, (1) the mudstone in the overlying Hanging Wall Andesite (Hanging Wall Mudstone), (2) two footwall sediment units (Lower Mudstone and Even Lower Mudstone), (3) extrusive units below the Rhyolite (Dacite and Footwall Andesite) and (4) the Bowser Group sediments. The structural model that was created in 2018 was also used. In total, 90 solids were created for the 2021 estimate including 84 mineralization solids, five low-grade envelope solids, and one solid used to restrict the influence of high-grade, mined out material. The mineralization domains were designed by lithology type, structural trends, and AuEQ assay intervals with a nominal cut-off of 0.5 g/t AuEQ or greater (where AuEQ = Au + Ag/74). Occasionally, lower-grade intersections were included to maintain continuity.

Three modelling methods were used:

1. Radial Basis Function (RFB) Indicator interpolants for the Contact Mudstones. The RBF is an estimator that models known data positions and can provide an estimate for any unknown points. Drill holes were composited to 1 m, with left over samples at the end of the holes appended to the previous sample. A 50% probability was applied, and a structural trend was used as the search orientation.

2. Interval selection for all other lithologies. A nominal cut-off grade of 0.5 g/t AuEQ was used to select assays intervals directly from the assay database. Domains were created using either the vein or intrusion tool.

3. Manual wireframing created in Vulcan. Two small solids in the Water Tower Zone were manually wireframed in Vulcan software.

Two block models were created:

- An open pit model using 9 x 9 x 49 m parent block sizes, with sub-block sizes of 3 x 3 x 2 m;
- An underground model using 3 x 3 x 2 m parent block sizes, with 1 x 1 x 1 m sub-block sizes.

Assays were composited from assays honouring the relevant mineralization domain boundaries to 2 m lengths for the open pit model, and 1 m lengths for the underground model.

Grades within each domain were capped within hard-domain boundaries. Capping values were selected on a zone-by-zone basis using the results from log probability plots, histograms, CV values, degradation plots, and percent metal loss analyses. Gold capping values ranged from 2–650 g/t Au and silver capping values ranged from no capping applied to 25,000 g/t Ag.

The density used for tonnage calculation for the 2021 estimate is based on the lithological model, with the mean value of measurements typically selected as the density for each lithology considered during modelling.

Variograms were used to assess for grade continuity, spatial variability in the estimation domains, sample search distances, and kriging parameters.

For the open pit model, grades were estimated into all 12 mineralization domains, and the five low-grade envelope domains. Five estimation domains below the bottom of the optimized resource pit were reported as resources potentially amenable to underground mining methods (22, HW, NEX, WTZ and the LP). Each of the models were optimized based on the defining mining scenario.

Ordinary kriging (OK) was used to estimate gold and silver in all domains within the open pit model. Gold and silver grades within the mineralization domains were estimated in three successive passes with increasing search radii based on variogram ranges. A hard boundary was applied within a 3 m restriction domain to limit the spread of high-grade values from mined-out intervals into the remaining resources area. Validation included visual inspection in plan and sectional views, comparison of OK estimates with inverse distance weighting to the second power (ID2) and nearest-neighbour (NN) methods, and swath plots. No major biases were noted.

OK was used to estimate gold and silver in all five domains within the underground model. Gold and silver grades within the mineralization domains were estimated in three successive passes with increasing search radii based on variogram ranges. A 1 m geotechnical solid around the underground workings was used as the depletion zone for reporting remaining resources. Validation included visual inspection in plan and sectional views, comparison of OK estimates with ID2 and NN methods, and swath plots. No major biases were noted.

For mineralization in domains exhibiting good geological continuity using adequate drill hole spacing, SRK considers that blocks estimated during the first estimation pass using a minimum of four holes, an average distance of less than 15 m and a kriging variance (KV) of less than 0.3, to be classified as the Measured category. KV provides a relative measure of accuracy of the local kriged estimate with respect to data coverage. Mineralization in domains exhibiting good geological continuity estimated during Pass 2 with a minimum of four drill holes were classified as Indicated. For Measured and Indicated blocks, the level of confidence is adequate for evaluating the economic viability of the deposit, as well as suitable for assessing technical and economic parameters to support mine planning. Blocks estimated during the Pass 3 pass, using search distances of 2.5 times the variogram range, and a KV of <0.8 were classified in the Inferred category. For those blocks, the level of confidence is inadequate for evaluating the economic viability of the deposit, as well as unsuitable for assessing technical and economic parameters to support mine planning. No Measured or Indicated Mineral Resources were classified in the low-grade envelope. Blocks in the low-grade envelope were classified as Inferred only if a minimum of three drill holes were used.

The epithermal suite of elements (antimony, mercury, and arsenic), base metals (lead, copper, and zinc) and metallurgical elements (iron and sulphur) were estimated into the open pit block model to provide results for the metallurgical study. A high degree of variability of the epithermal elements exists between the different zones and rocktypes, and elevated concentrations occur in localized zones/pods. The Contact Mudstone lithology within the 21A and 21B Zones have elevated levels of arsenic, mercury, and antimony. The 21A Zone is geologically and geochemically equivalent to the 21B Zone, an area that accounted for the bulk of mineralization historically mined at Eskay Creek. Smelter penalties for the elevated concentrations of arsenic, mercury, and antimony in the 21B Zone were often prevented via blending with material from other zones while maintaining a profitable head grade

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by open pit methods, SRK used a pit optimizer and reasonable mining assumptions to evaluate the proportion of the block model (Measured, Indicated, and Inferred blocks) that could be "reasonably expected" to be mined from the open pit. The optimization parameters were selected based on experience, and benchmarking against similar projects. The block model quantities and grade estimates were also reviewed to determine the portions of the Eskay Creek Project having "reasonable prospects for eventual economic extraction" using a long-hole underground mining scenario.

The cut-off grade for the open pit model was determined to be 0.66 g/t AuEQ; however, a pit constrained cut-off of 0.7 g/t AuEQ was selected for the estimate reporting. The long-hole mining and drift-and-fill underground mining method cut-off grades were calculated to be 2.4 g/t AuEQ and 2.8 g/t AuEQ, respectively. In the underground scenario, the steeply-dipping Water Tower Zone was determined to be potentially amenable to the long-hole mining method, while the NEX, HW, 22 and LP Zones were more potentially amenable to the drift-and-fill mining method.

1.11 Mineral Resource Statement

The Mineral Resources considered potentially amenable to underground mining are reported exclusive of the estimated Mineral Resources potentially amenable to open pit mining. Mineralization was depleted in the open pit model by removing all material within all historical workings, where the historical workings shells had been expanded by an additional 0.2 m in all directions. Mineralization within the underground model was depleted by removing all material within all historical workings, where the historical working shells has been expanded by an additional 1.0 m in all directions.

Mineral Resources are reported using the 2014 CIM Definition Standards in Table 1-1 and Table 1-2. Ms. S. Ulansky, Senior Resource Geologist, P. Geo (EGBC#36085), an employee of SRK. (Canada) Inc. is the Qualified Person for the estimates.

Table 1-1: Open Pit Constrained Mineral Resource Statement Reported at 0.7 g/t AuEQ Cut-Off Grade by Domain

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
Measured							
21A	1,863	4.9	3.9	71.8	291	233	4,303
21C	4,497	3.6	2.9	51.4	524	423	7,425
21B	1,997	10.9	7.4	257.5	697	474	16,533
21Be	1,640	8.8	5.8	220.5	462	305	11,630
21E	743	3.2	2.2	75.0	77	52	1,793
HW	919	5.8	3.6	163.9	172	107	4,840
NEX	4,540	5.5	3.8	125.2	804	557	18,271
WT	67	3.4	3.0	31.2	7	6	67
PMP	239	5.6	4.3	95.1	43	33	731
109	754	5.5	5.3	12.4	132	128	300
LP	52	1.2	1.1	9.2	2	2	15
Total Measured	17,312	5.8	4.2	118	3,213	2,322	65,908
Indicated							

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	3,764	3.4	2.7	46.1	406	330	5,583
21C	1,648	2.6	2.1	38.4	139	112	2,036
21B	3,100	3.9	2.9	75.3	390	289	7,501
21Be	848	5.1	3.9	92.4	140	105	2,522
21E	642	2.7	1.8	60.8	55	38	1,235
HW	1,470	3.9	2.5	104.5	185	118	4,938
NEX	3,171	2.4	1.8	40.3	244	188	4,104
WT	290	2.5	2.2	23.0	23	20	214
PMP	198	3.2	2.6	47.9	21	16	305
109	301	2.2	2.0	12.1	21	19	117
LP	1,465	1.1	0.9	9.6	51	45	545
Total Indicated	20,342	2.9	2.2	52.5	1,903	1,439	34,362
Measured + Indicated							
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	5,627	3.8	3.1	54.6	696	563	9,887
21C	6,145	3.4	2.7	47.9	663	535	9,461
21B	5,096	6.6	4.7	146.7	1,087	762	24,033
21Be	2,489	7.5	5.1	176.8	602	411	14,152
21E	1,385	2.9	2.0	68.4	131	90	3,047
HW	2,388	4.7	2.9	127.3	357	225	9,778
NEX	7,711	4.2	3.0	90.3	1,048	746	22,375
WT	358	2.7	2.3	24.5	31	27	282
PMP	437	4.5	3.5	73.7	64	50	1,036
109	1,055	4.5	4.3	12.3	153	148	416
LP	1,517	1.1	0.9	9.6	53	46	470
Total M + I	37,654	4.2	3.1	82.8	5,116	3,761	100,270
Inferred							
ENV	2,836	1.1	0.8	17.1	98	77	1,562
22	316	1.4	1.0	26.2	14	10	266
21A	938	1.1	0.8	24.5	34	24	739
21C	50	3.0	2.3	53.0	5	4	86
21B	564	2.0	1.6	26.0	36	30	471
21Be	22	3.3	2.7	41.0	2	2	29
21E	6	2.5	1.9	42.9	0.5	0.3	9
HW	324	3.3	2.0	92.0	34	21	958
NEX	30	2.5	2.1	25.7	2	2	25
WT	0.06	1.2	1.1	8.6	0.03	0.02	0.02
PMP	7	3.2	2.2	74.4	0.7	0.5	17
109	0.1	1.6	1.6	3.7	0.06	0.06	0.0
LP	145	1.0	2.3	9.0	5	4	40
Total Inferred	5,239	1.4	1.0	25.0	231	174	4,203

Table 1-2: Underground Mineral Resource Statement Reported at a 2.4 g/t AuEQ Cut-Off Grade for Long-Hole Mining and 2.8 g/t AuEQ Cut-Off Grade for Drift-and Fill-Mining

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
Measured							
WT	102	6.0	5.9	13.3	20	19	44
HW	19	5.7	4.5	95.3	3	3	57
NEX	222	6.2	5.0	90.3	44	36	645
LP	2	6.7	6.4	18.7	0.5	0.4	1
Total Measured	345	6.1	5.2	67.3	68	58	747
Indicated							
WT	215	5.4	5.3	10.4	38	37	72
22	61	6.5	4.9	117.2	13	10	230
HW	20	5.9	4.7	94.0	4	3	62
NEX	87	5.7	5.0	54.4	16	14	152
LP	123	4.3	4.1	17.0	17	16	67
Total Indicated	506	5.3	4.9	35.8	87	79	583
Measured + Indicated							
22	61	6.5	4.9	117.2	13	10	230
WT	317	5.6	5.5	11.3	58	56	116
HW	39	5.9	4.6	94.6	7	6	119
NEX	309	6.1	5.0	80.1	60	50	797
LP	125	4.3	4.1	17.0	17	16	68
Total M + I	851	5.7	5.0	48.6	155	137	1,330
Inferred							
WT	79	4.6	4.5	7.2	12	11	18
22	221	5.5	4.1	99.4	39	29	706
HW	1	5.3	4.2	83.1	103	81	2
LP	129	4.0	3.8	14.6	17	16	61
Total Inferred	429	4.9	4.1	57.0	67	57	787

Notes to accompany the Mineral Resource estimate statement:

- Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The Qualified Person for the estimate is Ms. S Ulansky, PGeo of SRK Consulting (Canada) who reviewed and validated the Mineral Resource estimate.
- The effective date of the Mineral Resource estimate is April 7, 2021.
- The number of metric tonnes and ounces were rounded to the nearest thousand. Any discrepancies in the totals are due to rounding.
- Open pit-constrained Mineral Resources are reported in relation to a conceptual pit shell.
- Reported underground Mineral Resources are exclusive of the Mineral Resources reported within the conceptual pit shell and reported using stope optimized shapes based on long-hole and drift-and-fill mining methods.
- Block tonnage was estimated from average specific gravity measurements using lithology groupings.
- All composites were capped where appropriate.
- Mineral Resources potentially amenable to open pit mining methods are reported at a cut-off grade of 0.7 g/t AuEQ and Mineral Resources potentially amenable to underground mining methods are reported at a cut-off grade of 2.4 g/t AuEQ for long-hole methods and 2.8 g/t AuEQ for drift-and-fill methods.

- Cut-off grades are based on a price of US$1,700/oz Au US$23/oz Ag, and gold recoveries of 90%, silver recoveries of 80% and without considering revenues from other metals. AuEQ = Au (g/t) + (Ag (g/t)/74).
- Open pit key assumptions for reasonable prospects of eventual economic extraction are as follows:
    - An overall pit wall angle of 45º;
    - A reference mining cost of US$3.00/t mined;
    - A processing cost of US$15.50/t processed;
    - General and administrative costs of US$6.00/t processed;
    - Mining dilution of 5%;
    - Mining recovery of 95%;
    - Transportation and refining costs of US$25/oz AuEQ;
- Underground key assumptions for reasonable prospects for eventual economic extraction are as follows:
    - A reference mining cost of US$80/t mined;
    - A processing cost of US$25/t milled;
    - General and administrative costs of US$12/t milled;
    - All in costs of US$117/t milled;
    - Transportation and refining costs of US$25/oz AuEQ;
- Estimates use metric units (metres, tonnes and g/t). Metals are reported in troy ounces (metric tonne * grade/31.10348).
- The 2014 CIM Definition Standards were used for the reporting of Mineral Resources.
- Neither Skeena nor SRK is aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimates.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to the density values applied to the mineralized zones; changes to geological shape and continuity assumptions; potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures; changes to the input values used to generate the AuEQ cut-off grade; changes to metallurgical recovery assumptions; changes in assumptions of marketability of final product; changes to the conceptual input assumptions for assumed open pit operations, changes to the input assumptions for assumed underground operations; variations in geotechnical, hydrogeological and mining assumptions; changes to environmental, permitting and social license assumptions.

1.12 Mineral Reserve Estimates

The Mineral Reserve estimates for the Eskay Creek Project are based on the conversion of the Measured and Indicated Mineral Resources within the current mine plan. Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted directly to Probable Mineral Reserves. Inferred Mineral Resources were treated as waste. The estimates assume conventional open pit mining and equipment.

Inputs to the estimates include:

- Open pit slope recommendations, which were based on geotechnical data from drilling, logging, mapping, sampling, and laboratory testing;

- Pit shells designed using the Lerchs–Grossmann (L–G) algorithm in MinePlan software. Ultimate pits were generated using a revenue factor of 0.9 or metal price of $1,328 /oz. These were used as the basis for the design;
- Two pits were designed, a large north pit with five phases, and a small single-phase south pit;
- A C$30.56/t NSR cut-off represented the marginal cut-off grade to flag initial feed and waste blocks;
- Contact dilution was modelled into the in-situ resource blocks using an assumed 1.25 m contact dilution distance between each block. The average grade of the dilution material was 0.16 g/t Au and 3.65 g/t Ag.

1.13 Mineral Reserve Statement

The total reserves for the Eskay Creek Project are shown in metric units in Table 1-3. Some variation may exist due to rounding.

Table 1-3: Proven and Probable Reserves (Metric Units)

Reserve Class	Tonnes		Grade			Contained Ounces	
	(Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)
Proven	13.5	4.25	124	5.81	1.85	53.7	2.53
Probable	12.9	2.46	64	3.26	1.02	26.5	1.35
Total	26.4	3.37	94	4.57	2.87	80.2	3.88

Note: This mineral reserve estimate has an effective date of June 30, 2021 and is based on the mineral resource estimate that has an effective date of April 7, 2021 for Skeena Resources by SRK Consulting. The Mineral Reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final design pit based on a US$1,475/oz gold price and US$20.00/oz silver price. An NSR cut-off of C$30.56/t was used to define the marginal cut-off material. The life-of-mine mining cost averaged C$3.14/t mined, preliminary processing costs are C$24.50/t ore and G&A was C$6.06/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. Gold concentrate grades varied from 20 to 45 g/t gold. Silver recovery was assumed to be 93% of the gold recovery.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves.

1.14 Mining Methods

1.14.1 Geotechnical Considerations

Following completion of the 2020 drilling program, AGP conducted a compilation, review, and assessment of available geotechnical data and information to determine initial estimates of suitable pit slope angles for PFS-level mine planning tasks. AGP's assessment is based primarily on geotechnical data from drilling, logging, mapping, sampling, and laboratory testing in consideration of economic pit shells, geologic models, and relevant background reports.

North Pit slopes are expected to consist primarily of andesites within the upper pit walls with rhyolite being more prevalent at lower pit elevations. Contact and hanging wall mudstone units intersect and are expected to impact pit slopes on the west and north walls of the pit. Overall, the geotechnical data indicates generally 'fair' rock mass conditions within the mining zone with 'poorer' quality rock masses and local bench-scale slope instability likely to be encountered in zones within and proximal to mudstone units, and adjacent to fault zones.

AGP divided the pit into slope design sectors based on slope height and dominant geology and geotechnical characteristics. The inter-ramp slope recommendations ranged between 34–46º.

Slope heights ranging from 100–300 m with inter-ramp and global slope angles varying from 30–45° were analysed under fully to partially saturated conditions for stability, and are expected to exhibit generally 'stable' conditions for a variety of scenarios, with typical 'minimum' factors of safety ranging from ~1.2 to >> 2.0 for inter-ramp and global slopes.

The proposed North pit will intersect and mine into the historical underground workings at approximately mid-slope height on the mid to north side of the pit. This will result in increased risks for safely mining in this area and prescriptive plans will need to be developed to adequately mitigate these risks to acceptable levels.

1.14.2 Hydrological Considerations

The regional groundwater regime is most likely controlled by the regional groundwater flow system, and from seasonal snow melt. The regional faults likely provide high permeability recharge pathways and groundwater storage areas; however, the rock units themselves are highly fractured and even away from major faults constitute fractured aquifers. Faulted andesite most likely provides the highest permeability and highest storage capacity of all the rock units. Historically, three high-permeability zones with large areal extents, and six hydrostratigraphic units were identified.

The planned ultimate pit bottom will be at 714 masl, and therefore only about 50 m of flooded working is likely to require dewatering. The andesite and mudstone lithologies will likely dewater easily compared to the rhyolite, which reportedly has high fines content and drains poorly (significantly lower hydraulic conductivity than the andesite). The rhyolite will generally occupy lower elevations in the final pit extent; however, rhyolite would be present on the south and east pit highwall and may be susceptible to failure if pore-water pressure builds up on fault planes.

Pit stability can be managed by progressive dewatering of the ground behind the pit slope with vertical or horizontal boreholes. The hanging wall (andesite and mudstone) rocks are rated as highly conductive (K = 3E-06 m/s) compared to the footwall (rhyolite) rock (K = 4E-08 m/s). The mudstones may require special attention as matrix pore pressures could remain elevated despite successful dewatering.

1.14.3 Mine Plan

The Project is located predominantly to the south of Tom Mackay Creek with a small portion extending to the north. Infrastructure will be located on the south side of Tom Mackay Creek, with the pit extending to the north beyond Tom Mackay Creek. Underground mining has previously been conducted in the northern portion of the Project at depth. The potential for underground development beneath the open pit was examined in preliminary evaluations during the 2021 PFS but was not included as part of the PFS. There is still potential for the inclusion of underground mining in future mining studies.

Each pit phase was designed to accommodate the proposed mining fleet. Mining will occur on 8 m benches with catch benches spaced either 8 or 16 m vertically depending on lithology type. The haul roads will be 30.2 m in width with a road grade of 10%.

The mine schedule plans to deliver 26.4 Mt of mill feed grading 3.37 g/t Au and 94.4 g/t Ag over a mine life of 10 years. Waste tonnage totalling 212 Mt will be placed into either non-acid generating (NAG) or potentially acid-generating (PAG) waste destinations. The overall strip ratio is estimated at 8.0:1. The mine schedule assumed a maximum of 2.9 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. A maximum descent rate of eight benches per year per phase was applied.

The proposed mine life includes 30 months of pre-stripping and 10 years of mining. Mill feed will be stockpiled during the pre-production years, with three grade stockpiles envisaged. A technical sample will be mined in Year -3 so that process performance of the mill can be evaluated on a bulk sample.

The mine equipment fleet is anticipated to be leased to lower capital requirements. Pre-production mining will be completed with 11.5 m^3 loaders and 91 t rigid body trucks. This smaller fleet is better suited to the lower production tonnage requirements and narrower working conditions. With full production starting in Year 1, the primary loading units will be 22 m^3 hydraulic shovels. Additional loading will be completed by a 23 m^3 loader. The smaller loaders will shift to working at the primary crusher and site maintenance roles (snow removal, etc.). It is expected that one of the 11.5 m^3 loaders will be at the primary crusher full time. The main production haulage trucks will be conventional 144 t rigid body trucks from Year 1 onwards. The support equipment fleet will be responsible for the usual road, pit, and waste rock storage facility (WRSF) maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management.

Grade control will be completed with a separate fleet of RC drill rigs, with a 10 m x 5 m pattern in ore and 20 m x 10 m pattern in waste. Blasthole sampling will also be part of the initial grade control program to determine the best sampling method for operations.

There will be three WRSFs that will store the NAG waste. The largest will be located to the immediate west of the north and south pits, and two smaller WRSFs will be constructed to the west and northeast of the North pit respectively. A portion of the NAG waste will also be disposed of in the North pit as backfill. The WRSF design used a swell factor of 1.30. For the WDW facility, the lift height will be 20 m. Assuming a 37° face slope, the overall slope will be 26.5° with 13.6 m berm widths. A 37° face slope was also used for the in-pit backfill WRSFs. PAG waste will be sent to the TMSF to be submersed below water.

1.15 Recovery Methods

The plant will process ore at a nominal rate of 2.9 Mt/a for Years 1 to 4 and 2.7 Mt/a for the remaining years with an average head grade of 3.2 g/t Au and 94 g/t Ag. The ore becomes harder and more competent after the first four years of operation. The plant is designed to operate two shifts per day, 365 d/a with an overall plant availability of 92%. The process plant feed will be supplied from the Eskay Creek open pit mine and the process plant will produce gold concentrate to be sold to refineries.

The process plant will consist of the following areas:

- Single stage crushing circuit (jaw), fed from the open pit mine;
- Coarse ore stockpile with reclaim system, fed from an overland conveyor;
- Primary grinding including a SAG mill, pebble crusher (installed at Year 4), and ball mill in closed circuit with hydrocyclones;
- Rougher flotation with conventional concentrate regrind and two stages of cleaning;
- Slimes classification via two stages of hydrocycloning, fed from the rougher flotation tails;
- Secondary grinding and scavenger flotation, fed from the slimes circuit underflow;

- Fines flotation and two stages of cleaning, fed from the slimes circuit overflow;
- Concentrate thickening, storage and filtration;
- Concentrate load-out by way of front-end loader filling concentrate transportation;
- Final tailings pumping to the TMSF.

1.16 Project Infrastructure

1.16.1 Facilities

Infrastructure to support the Project will consist of site civil work, site facilities/building, a water system, and site electrical. Site facilities will include both mine and process facilities:

- Mine: administration offices, truck shop and warehouse, tire repair shop, mine workshop, mine dry, fuel storage and distribution, mobile equipment, temporary camp for accommodating construction crew, permanent camp facility and miscellaneous facilities;
- Process: process plant, crusher facility, process plant workshop and assay laboratory;
- Services: security, information technology, potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

  Waste material storage: Tom MacKay Storage Facility (Tailings and PAG Waste Rock) and Waste Rock Storage Facility

1.16.2 Logistics

Multiple options for the export of concentrate were studied, with two options through Stewart identified as preferred.

- Option 1: uses bulk haul of concentrate with heavy-haul side-dump truck and trailer units to the Stewart Bulk Terminal (SBT) facility. Product would be stored in a separate bulk storage building before loading bulk carrier vessels with a conventional bulk shiploader;
- Option 2: uses specialized bulk container units that would be trucked to Stewart World Port (SWP). Loaded containers would be swapped with empty ones and used as temporary storage until a bulk vessel arrives. Containers would be lifted by the ship's cranes using a specialty spreader unit that would discharge the concentrate directly into the vessel's hold.

Both transportation options have similar overall logistics costs for the movement of concentrate from the mine into a ship. The bulk carrier vessel would be the same in each case and would transport the concentrate to a terminal facility nearest the preferred smelter location in southern China. While both options were considered during the PFS, the selected option as well as the option upon which the PFS values and costs were based on is the bulk transportation option, Option 1.

Construction materials and mine consumables would be moved through the SBT site, which has a general cargo dock.

1.16.3 Tom Mackay Tailings Storage Facility

The existing TMSF was selected as the preferred tailings storage option since it is permitted as a tailings storage facility (TSF). The TMSF will have sufficient capacity to contain 76.7 Mt of tailings and PAG waste rock and will be constructed in two phases over the LOM based on storage and operating criteria.

The tailings and PAG waste rock embankments at Eskay are designed in accordance with Canadian Dam Association (CDA) "Dam Safety Guidelines" (CDA 2007; 2013), which also provides guidelines in evaluating the classification of dams in terms of the consequence of failure. Based on the dam breach analysis and expected area of inundation downstream of the tailings and PAG waste rock storage facility, the consequence of a dam failure based on HSRC Guidance Document, Section 3.4 (BC Ministry of Energy and Mine 2016) and CDA (2013) Dam Safety Guidelines is "very high" for the TMSF. Therefore, the facility was design in accordance with those guidelines.

The TMSF is designed to be founded on bedrock with low permeability characteristics to limit seepage below the embankment. The overall design objective of the TMSF is to protect the regional groundwater and source waters resources during both operations and over the long term (after closure). TMSF development will be phased with downstream embankment construction methodology. NAG mine waste from the pit will be used as the primary construction material. The upstream side of the embankment will be lined with a geomembrane to minimize potential seepage through the dams. Between the geomembrane liner and the waste rock shell will be a filter zone and low permeability zone to aid in minimizing seepage through the embankments. A floating turbidity fence will be installed between the embankment and the waste rock storage area to reduce and/or eliminate the passage of fine-grained suspended solids that would otherwise be discharged downstream.

The operational plan of the TMSF is to deposit slurry tailings at the south end of the facility and PAG waste rock at the north end of the facility. PAG waste rock deposition will use a causeway approach, depositing waste across the facility from west to east. The causeways will be constructed 2 m above the water surface with a crest width of 65 m to provide sufficient operating area for haul trucks, dozers, and a dragline excavator. Once completed the next causeway will be constructed next to the completed causeway. During the construction of the next causeway, a dozer and dragline excavator will remove the upper 5 m and place the material to the south of the causeway to minimize sediment migration toward the north due to excavation operations. The final height of the causeway will be 3 m below the water surface.

Tailings will be slurried from the process plant to the TMSF by way of a pipeline, which would extend onto the TMSF to a floating barge. Due to the fine ore grind (P80 = 45 µm), the end of the pipeline will be positioned close to the bottom of facility (deposited tailings) to maximize settling and minimize entrainment of fine particles to the surface of the TMSF. The minimum water depth over the tailings would be 3 m during operations and 6 m at closure to prevent both wind and ice remobilization of the tailings. The TMSF has sufficient capacity to store tailings with three small embankments (an average less than 10 m) during the initial years of operations while maintaining 3 m (3–5 Mm^3) of water cover over the tailings bed and PAG waste rock. In year 4 of operations, a single raise of the three embankments (less than a total height of 50 m) will be required to be constructed, so as to store the balance of the LOM tailings and PAG waste rock while maintaining 3 m of water cover during operations and 6 m of water cover at post-closure.

1.16.4 Water Supply and Management

Pit water will be sent directly to a water treatment plant (WTP), then to D7 polishing ponds, and finally to Ketchum Creek during pre-production. The water treatment plant's maximum capacity has been designed to accommodate the pit water with additional treatment capacity. The WTP has a capacity of approximately 150 L/s, which supports pre-production operations. Once the tailings pipeline is installed and operations begin, pit water will report to the tailings mixing tank at the plant and sent with the tailings in the tailings transportation pipeline to the TMSF. As the open pit becomes larger, pit dewatering flow rates will increase. The pit dewater flow to the tailings mixing tank will range from 65.5 to 376.3 L/s during the mine life.

The WDW water management includes both contact and non-contact water management structures. The facility is located in a relatively small watershed. The non-contact water will pass underneath the facility in a rock drain that converts to 2 solid wall HDPE pipes that discharge water directly into Tom MacKay Creek. The surface contact water from the WRSF will be conveyed in both temporary and permanent diversion channel to contact water 5 Pond to remove sediment 10 microns and above prior to releasing water into Tom MacKay Creek. The contact water management system was designed for 1:200 year event and the non-contact water management system for 1:475 year event.

The industrial water requirements will come from the TMSF, which are estimated to be 113 L/s to be used in mineral processing. Fresh/fire water will be pumped from a local fresh water supply well into a fresh/fire water tank.

The planned camp will be supplied for all its water needs from a local well. It is estimated that the average consumption of water, based on the size of the camp, is 1 L/s. Any effluent coming from the camp will be treated and discharged into the TSF.

No diversion works are anticipated. There will be inflow of water into the TMSF from direct rainfall and snow and runoff from the surrounding catchment into the TMSF.

1.16.5 Power

Project power will be provided through a 20 km long 69 kV overhead transmission line. The source of power will be from the Volcano Creek 287 kV substation. The estimated power demand for the Project is 21 MW.

1.17 Environmental, Permitting, and Social Considerations

1.17.1 Environmental Considerations

Several environmental studies were completed at the Eskay Creek mine under various owners. Environmental monitoring was also completed during and after operations. In 2020, Skeena began additional geochemical, environmental, social, economic, heritage and health baseline studies to reflect current environmental and social conditions. These studies will help refine the Project design and support applications for provincial and federal regulatory approvals.

The main waste management issue for the Project is the prevention and control of metal leaching/acid rock drainage (ML/ARD) from the tailings and waste rock. NAG waste rock will be deposited in two locations: approximately 90% will be stored in the WD-01 facility that will be located to the south of the open pit. The remaining 10% of the total waste rock will be backfilled in the north pit. PAG waste rock will be deposited in the TMSF with a water cover. Tailings will be deposited sub-aqueously in the permitted TMSF with a water cover. In 2020, a geochemical study was initiated on new waste rock, ore, tailings and overburden sources for the Project together with the existing tailings in TMSF. The purpose of this study was to update and inform waste management decisions for the Project design. To manage the potential for ML/ARD, Skeena has incorporated design features and mitigation measures that are consistent with best practices for waste and water management.

Site water management will be a critical component of the Project design. Mine water can be divided into two categories depending on the potential for contamination:

- Non-contact water from upstream catchments that has not been in contact with mine workings and surface infrastructure will be kept from water which will come into contact with mine workings and surface infrastructure. Non-contact water will be diverted around the mine site as much as possible;

- Contact water will interact with potential sources of contamination including seepage from the WRSF, temporary stockpiles, process water, infrastructure surface runoff, and pit dewatering. Contact water will be collected and if required, treated to meet permit discharge limits prior to discharge. Process water will be discharged to the TMSF.

Strategies for water management include collecting surface water from disturbed areas (mine-contact) to manage surface water erosion; recycle mine-contact water whenever possible; treat mine-contact water as required; and monitor water quality to meet discharge standards prior to discharge.

1.17.2 Closure and Reclamation Planning

The mine closure strategy for the mine will be to have a stable, revegetated site with best mitigation of potential ML/ARD and water quality risks that is consistent with the Tahltan and Skeena's agreed Social and Environmental Design Principles and post-mining end land uses. A Closure and Reclamation Plan will be developed during the permitting process which is designed to achieve end land use objectives (e.g. wildlife habitat), in consideration of Indigenous interests. Closure planning will include consultation with Indigenous groups and stakeholders to determine post-mining land use objectives and supporting strategies, including addressing regulatory requirements. Achieving the desired outcomes will be an iterative process during the design and permitting process and incorporate social, environmental, engineering, technical and Tahltan criteria.

The proposed Project is anticipated to undergo a concurrent Environmental Assessment /Impact Assessment (EA/IA), called a substituted process, under federal and provincial regulations. Since the Eskay Creek Mine has two existing Certificates, one or both will be amended through a substituted EA/IA process. The Eskay Creek Mine went through two EA processes in its history. An application for a Mine Development Certificate (MDC) was approved in 1994 and the MDC was issued under previous environmental review legislation and is considered equivalent to an EA Certificate under present legislation. In 2000, an application for an EA Certificate was reviewed and a Project Approval Certificate was approved for disposal of mine tailings into Tom MacKay Lake and is also equivalent to a present-day EA Certificate.

The 1993 the MDC enabled the previous operator to obtain construction/operation permits under the Mines Act, to build the Eskay Creek mine, including underground mining, surface workings, and use of Albino Lake as a WRSF and offsite shipping of ore. In 1997, permits were amended to build a mill onsite and dispose of tailings with waste rock to Albino Lake. Once the Project Approval Certificate was issued in 2000 for the use of Tom MacKay Lake as a tailings disposal facility, construction and operation permits were obtained. The deposition of mine waste in Albino Lake and Tom Mackay Lake is listed under Schedule 2 - Tailings Impoundment Areas, of the federal Fisheries Act.

For the proposed Project, Skeena will undertake a substituted process to amend an existing EA Certificate or obtain a new EA Certificate. The process to follow for the EA/IA is being developed with the provincial and federal regulators, the Tahltan Nation and Skeena based upon the legislative steps, criteria, and procedures. In addition to obtaining the EAC, the Project will require permits and authorizations in accordance with provincial and federal legislation and regulations prior to construction, operation and ultimately mine closure. A new mine reclamation security bond will be established in conjunction with the approved mine plan and reclamation program under the Mines Act. No permits for project commercial development will be issued before an EA Certificate is obtained. Consequently, Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction.

No technical or policy issues are anticipated for obtaining the required Project permits and approvals, given its long mining history.

1.17.3 Social Considerations

Northwestern BC is a sparsely populated and relatively undeveloped region of the province. Many of the smaller communities have predominantly Indigenous populations that are isolated from one another as well as from the main regional centres of Smithers and Terrace. Land and resource uses within the region include trapping, guided hunting, commercial recreation and outdoor recreation including fishing, hunting, camping, hiking, snowmobiling, all-terrain vehicle (ATV) riding and skiing. In the vicinity of the Project, there are mineral, water and range tenures, guide outfitter, and traplines. There are seasonal Tahltan cabins along the Eskay Mine Road. Community and socio-economic impacts of the Project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers.

Provisions for consultation with Indigenous Nations and the public are a component of the provincial and federal legislation for both the EA/IA processes and permitting activities. Skeena is developing an Engagement Plan for the Project as required by the provincial and federal EA processes. This plan provides a summary of Skeena's engagement activities as well as serve as a guide for Skeena's engagement activities with identified Indigenous Nations and stakeholders throughout the EA/IA process. The Engagement Plan will be submitted with the Initial Project Description to begin the EA/IA process. Ongoing and future engagement and consultation measures by Skeena are driven by best practices as well as Skeena's internal company policies. These measures will at a minimum comply with federal and provincial regulations.

Skeena recognizes engagement and support of the Project from Indigenous Nations from initial project design until post-closure is critical for the success of the Project. Skeena is and will consult with local Indigenous Nations to gain that support, yet also recognizes this is part of the EA process at both the provincial and federal level. Engagement with local Indigenous Nations will continue throughout the Project design, construction, operations, closure, and post-closure. The Project is located within the traditional territory of the Tahltan Nation and the asserted territory of the Tsetsaut Skii Km Lax Ha. The historical environmental process and subsequent expansions included consultation with the Iskut Band, Tahltan Band, and the Tahltan Central Government. Project traffic will use Highways 37 and 37A which pass through the Nass Area and Nass Wildlife Area (as defined by the Nisga'a Final Agreement) and the traditional territory of the Gitanyow Nation.

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will form a Project-specific working group at the early stages of the EA/IA process, which will include representatives from many government groups. Skeena will consult with the working group on project-related developments during the EA/IA process. Skeena will consult with the public and relevant stakeholder groups, including tenure holders, businesses, economic development organizations, businesses and contractors (e.g., suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation groups), as appropriate.

1.18 Markets and Contracts

The proposed Eskay Creek operation will produce a gold concentrate on site, which will then be shipped out of province to processing facilities. There is currently no contract in place with any smelter or buyer for the concentrate. The concentrate as proposed is a complex gold concentrate with relatively low gold content and elevated levels of arsenic, mercury, and antimony. Deleterious element assays are notably elevated in the first few years of mine life (arsenic in Years 1 and 2 and mercury in Years 1 to 3) before dropping to values which fall within typical industry expectations.

Given the complexity of the Eskay Creek concentrate, combined with the historical production of relatively difficult-to-market concentrates from the mine during its previous operational period, two independent, preliminary market studies were completed to support the NSR used in the 2021 PFS. Concentrate quality parameters are based on the results of ICP analysis of gold–silver concentrates produced during the variability flotation testwork at BaseMet.

Skeena engaged several internationally known consultants, expert in the marketing of complex gold concentrates. Each consultant conducted their own studies and provided opinions on potential smelters, treatment charges and penalties and net gold and silver payable. The consultants' reports were made available to the QP, and several meetings were held with the consultants that the QP participated in. In the opinion of the QP, the reports are suitable for use in this study and the selected smelter terms accurately reflect the potential treatment charges, penalties and net smelter returns for the Eskay Creek concentrates.

Based on the predicted analysis, the Eskay Creek concentrates will be saleable. The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across several buyers since individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable limits. An alternative option is to sell the concentrate to traders who may be able to buy all concentrate and spread distribution across a range of end customers, potentially including a mix of gold and copper smelters. Expectations of NSR may be achieved and penalties for deleterious elements may be minimized. Concentrate grades for gold, silver, mercury, antimony, and arsenic are expected to vary throughout the life of mine which will impact the marketability and net revenue. Concentrate volumes are expected to decrease over the mine life as the feed grade decreases. This should result in an easier blending of the deleterious elements out of the concentrate over time.

The most likely market for the concentrate is China, where the material will be imported as a gold concentrate (exceeding the minimum gold content criterion) and will therefore not be subject to arsenic import limits that would be imposed on base metal concentrate imports. The Chinese market offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate. Chinese gold smelters can typically monetize antimony at the levels found in the Eskay Creek concentrates.

No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts. Any future contracts are expected to be within industry norms.

The base case for logistics is moving the concentrate by bulk bags to Prince Rupert, where they will be loaded into containers for export via container vessels to China. The projected overall transport cost is US$137/t.

1.19 Capital Cost Estimates

1.19.1 Summary

LOM Project capital costs total $627.7M (Table 1-4).

Table 1-4: Capital Cost Estimate Summary

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Mine			
Pre-stripping	88.2	0.0	88.2
Mining equipment	14.1	17.2	31.3
Mine infrastructure	4.0	18.1	22.1
Mine Infrastructure (waste rock storage facility, waste management pond & channels, initial dewatering, water treatment plant, Truck Shop)	13.6	4.7	18.3
Sub-total mine	*119.9*	*40.0*	*159.9*
Processing			
Ore handling	17.4		17.4
Processing plant	97.4	1.3	98.7
Tailings and reclaim water	8.1	6.1	14.2
Onsite infrastructure	68.1		68.1
Sub-total processing	*191.0*	*7.4*	*198.4*
Offsite Infrastructure			
Access road	4.3		4.3
Power supply	24.9		24.9
Sub-total offsite Infrastructure	*29.2*		*29.2*
Sub-total direct costs	*340.1*	*47.4*	*387.5*
Indirect Costs	68.0		68.0
Sub-total directs + indirect costs	*408.1*	*47.4*	*455.5*
Owner's costs	27.2		27.2
Total excluding contingency	*435.3*	*47.4*	*482.7*
Project contingency	52.6		52.6
Sub-total	*487.9*	*47.4*	*535.3*
Closure costs		92.4	92.4
Total	*487.9*	*139.8*	*627.7*

The costs can be broken down as follows:

- Initial capital costs: include the costs required to construct all the surface facilities, and open pit development to commence a 2.9 Mt/a operation. The initial capital cost is estimated to be $487.9 M;
- Sustaining capital costs: include all the costs required to sustain operations, with the most significant component being open pit mine development. Sustaining capital costs total $47.4 M over the LOM;
- Closure costs: include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $92.4 M.

The estimate is based on assumptions of an exchange rate of US$0.78:C$1.00, is expressed in Canadian dollars, has a base date of Q1, 2021, and has an accuracy range of -20% to +30%.

1.19.2 Mining Costs

The mining capital cost estimate is grouped into three main categories:

- Pre-production stripping costs: $88.2 M; covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and financing costs;
- Mining equipment capital: $31.3 M; reflect the use of financing of the major equipment and some support equipment. Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed. The remaining cost was included in operating costs;
- Miscellaneous mine capital. $22.1 M; includes various separate line items in the costing, such as office, dispatch, communication, dewatering equipment and software, road development, water diversion tunnel, and clearing/grubbing.

1.19.3 Process Costs

Direct process costs were estimated from a combination of budgetary quotations received from qualified vendors, estimates based on preliminary general arrangement drawings and previous designs and factoring from historical costs. Project indirect costs, including construction indirect costs, spare parts, were developed using first principles methods, based on Ausenco experience. Camp requirements were based on personnel number estimates.

The process initial capital cost is $191.0 M, and includes provision for: ore handling; grinding, milling and classification; separation and concentration; reagents and process utilities; tailings and reclaim water; site preparation; onsite roads; onsite power transmission; and other onsite infrastructure. Sustaining capital costs include provision for grinding, milling and classification, and tailings and reclaim water, and total $7.4 M.

1.19.4 Offsite Infrastructure Costs

Offsite cost allocations were made for the electrical substation at Volcano Creek, a 20 km-long overhead powerline, and widening of the access road.

1.19.5 Indirect Costs

Indirect costs were developed using the first-principles method. Engineering, procurement, and construction management (EPCM) was estimated at 16.6% of total direct costs (excluding mining costs), field indirect costs at 15% of total direct costs and spares and first fills at 2% of total direct costs. Costs totalled $68.5 M and were included as part of the initial capital estimate.

1.19.6 Owner's Costs

Owner's costs were estimated at 4% of total direct and indirect costs. These costs include an allowance for: general and administrative costs for the Owner's project team on and off-site; consultants and contractors; mobile equipment and fixed plant; and pre-production operations.

1.20 Operating Cost Estimates

1.20.1 Summary

The operating cost estimate provided in Table 1-5 is based on a combination of first-principle calculations, experience, reference projects and factors.

Table 1-5: Operating Cost Estimate

Operating Cost	Annual Cost ($M)	Annual Cost ($/t Processed)
Processing	37.46	14.18
Maintenance	7.97	3.02
G&A	16.46	6.23
Road and bridge maintenance	2.70	1.02
Mining	80.74	30.56
Total	*145.34*	*55.01*

1.20.2 Mining Operating Costs

Costs were estimated from base principles with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The fuel price provided for the project was $1.18/L delivered to the site. The mine fleet will be mostly diesel powered, except for the 22 m^3 loading shovel. The dewatering pumps will be electric powered and a price of $0.06 per kilowatt hour was used.

Labour costs for the various job classifications were obtained from salary surveys in British Columbia and other operations. A burden rate between 39% and 44% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis using a rotation of either two weeks on/two weeks off or four days x three days.

All the major mine equipment, and the majority of the support equipment, where it was considered reasonable, was assumed to be leased. If the equipment had a life greater than the lease term length, then the following years onward of the lease did not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

Using the leasing option adds $0.38/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it was $3.24/t mill feed.

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Eskay Creek 2021 PFS. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. Drilling in the open pit will use down-the-hole hammers drill rigs with 229 mm bits. The pattern size varies between mill feed and waste and is blasted in recognition of the equipment being used. An emulsion product will be used for blasting to provide water protection. The blasting cost is estimated using quotations from a local explosives vendor. Loading costs for both mill feed and waste are based on the use of hydraulic shovels and front-end loaders. Haulage profiles were determined for each pit phase for the primary crusher or the waste rock facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Support equipment hours and costs were determined on factors applied to various major pieces of equipment. Grade control will be completed with a separate fleet of RC drill rigs.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface to push the water to the settling ponds. Additional dewatering in the form of horizontal drill holes is included as part of the dewatering costs. These holes will be campaigned and are part of the sustaining mine capital.

1.20.3 Process Costs

Processing reagent and consumable costs were estimated from first principles. Annual maintenance spares and consumable costs were estimated at 3% of total installed costs for mechanical equipment, plate work, support steel and electrics. Labour costs include all processing and maintenance costs. Costs were estimated from a breakdown of staffing positions, excluding G&A manpower. An allowance of 15% of all other operating costs was made, to include fuel costs, laboratory chemicals and similar sundry items. Reagent consumptions are based on testwork results while consumable wear items are estimated based on hardness and abrasion data.

1.20.4 Power Costs

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh. Power consumption was derived from calculated power draw of the ball and SAG mills, plus an allowance for the remainder of the plant, based on typical flotation plants. The average on-line power draw is estimated at 19 MW. Annual energy consumption is estimated at 127,564 MWh, or about $7.65 M.

1.20.5 General & Administration Costs

The G&A operating costs were estimated based on benchmarked data from similar projects in BC Canada. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

1.21 Economic Analysis

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

- Mineral Resource and Mineral Reserve estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and ability to mine in areas previously exploited using underground mining methods as envisaged;

- Sustaining costs and proposed operating costs;
- Interpretations and assumptions as to joint venture and agreement terms;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of ore, grade or recovery rates;
- Geotechnical or hydrogeological considerations during mining being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates;
- Changes to tax rates.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations, and approval for development to be provided by Skeena's Board.

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess impact of variations in metal prices, head grades, operating costs and capital costs. The economic analysis has been run with no inflation (constant dollar basis).

The economic analysis was performed using the following assumptions:

- Construction period of three years;
- Mine life of 9.8 years;

- Base case gold price of US$1,550/oz and silver price of US$22/oz were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;

- United States to Canadian dollar exchange rate assumption of 0.78 (US$/C$)

- Cost estimates in constant Q2 2021 C$ with no inflation or escalation factors considered;

- Results are based on 100% ownership with 2% NSR;

- Capital costs funded with 100% equity (i.e. no financing costs assumed);

- All cash flows discounted to start of construction;

- All metal products are assumed sold in the same year they are produced;

- Project revenue is derived from the sale of gold concentrate into the international marketplace;

- No contractual arrangements for smelting or refining currently exist.

The Project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%);

- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

Total tax payments are estimated to be C$1,145 M over the LOM.

A 2% NSR royalty has been assumed for the Project, resulting in approximately C$109 M in royalty payments over life of mine.

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$2,174 M, the internal rate of return IRR is 68.3%, and payback is 1.3 years. On an after-tax basis, the NPV5% is C$1,399 M, the IRR is 55.5%, and the payback period is 1.4 years.

A summary of the Project economics is included in Table 1-6 and shown graphically in Figure 1-1.

Table 1-6: Summary, Projected LOM Cashflow Assumptions and Results

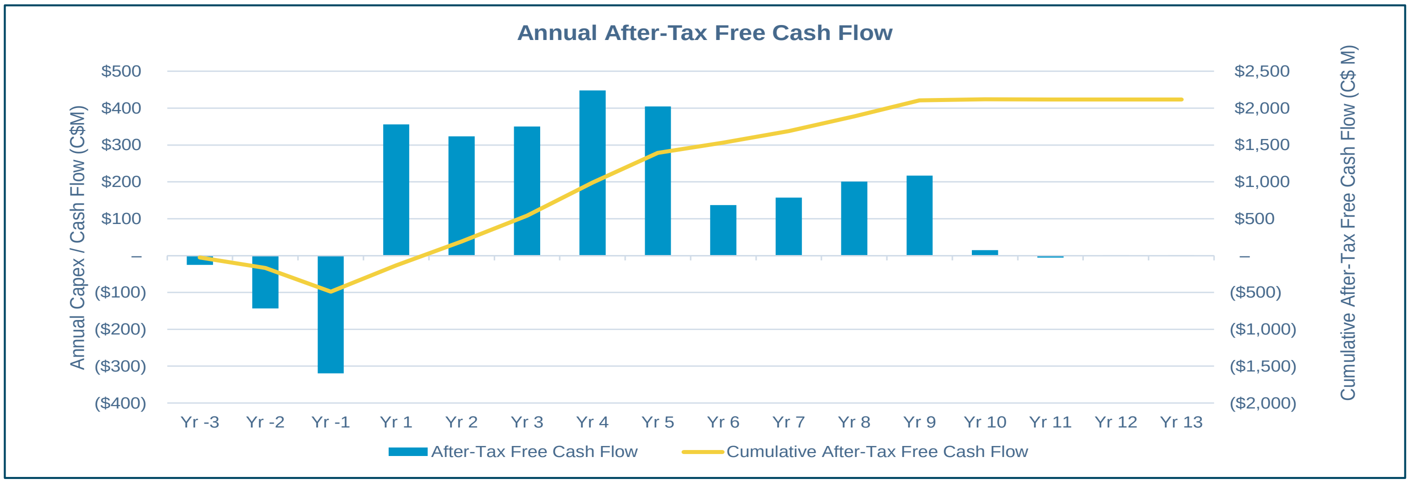
	Units	Values
General Assumptions		
Gold price	(US$)	1,550
Silver price	(US$)	22
Exchange rate	(US$/C$)	0.78
Fuel cost	(C$/litre)	1.18

	Units	Values
Power cost	(C$/kWh)	0.06
Discount rate	(%)	5
Net smelter royalty	(%)	2
Contained Metals		
Contained gold ounces	(koz)	2,866
Contained silver ounces	(koz)	80,197
Production		
Gold recovery	(%)	84.2
Silver recovery	(%)	87.3
LOM gold production	(koz)	2,448
LOM silver production	(koz)	70,902
LOM gold equiv. production	(koz)	3,455
LOM avg. annual gold production	(koz per annum)	249
LOM avg. annual silver production	(koz per annum)	7,222
LOM avg. annual gold equiv. production	(koz per annum)	352
Operating Costs Per Tonne		
Mining cost	(C$/t mined)	3.6
Mining cost	(C$/t milled)	30.6
Processing cost	(C$/t milled)	17.2
G&A cost	(C$/t milled)	6.2
Road and bridge maintenance cost	(C$/t milled)	1.0
Total operating costs	(C$/t milled)	55.0
NSR Parameters		
Gold payability	(%)	83.9
Silver payability	(%)	83.2
Transport to smelter	(C$/wmt)	146
Cash Costs and All-in Sustaining Costs		
LOM cash cost net of silver by-product	(US$/oz Au)	84
LOM cash cost co-product	(US$/oz AuEQ)	509
LOM AISC net of silver by-product	(US$/oz Au)	138
LOM AISC co-product	(US$/oz AuEQ)	548
Capital Expenditures		
Initial capex	(C$M)	488
Sustaining capex	(C$M)	47
Closure capex	(C$M)	92

	Units	Values
Economics		
Pre-tax NPV (5%)	(C$M)	2,174
Pre-tax IRR	(%)	68.3
Pre-tax payback period	(years)	1.3
Pre-Tax NPV / Initial Capex	(x)	4.5
After-tax NPV (5%)	(C$M)	1,399
After-tax IRR	(%)	56
After-tax payback period	(years)	1.4
After-Tax NPV / Initial Capex	(x)	2.9
Average annual after-tax free cash flow (Year 1–9)	(C$M)	265
LOM after-tax free cash flow	(C$M)	2,118

Notes: Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. All-in sustaining cost (AISC) includes cash costs plus corporate G&A, sustaining capital and closure costs. Gold equivalent (AuEQ) calculated using the formula: Au (g/t) + [Ag (g/t) / 70].

Figure 1-1: Projected LOM Cashflow



Note: Figure prepared by Ausenco, 2021.

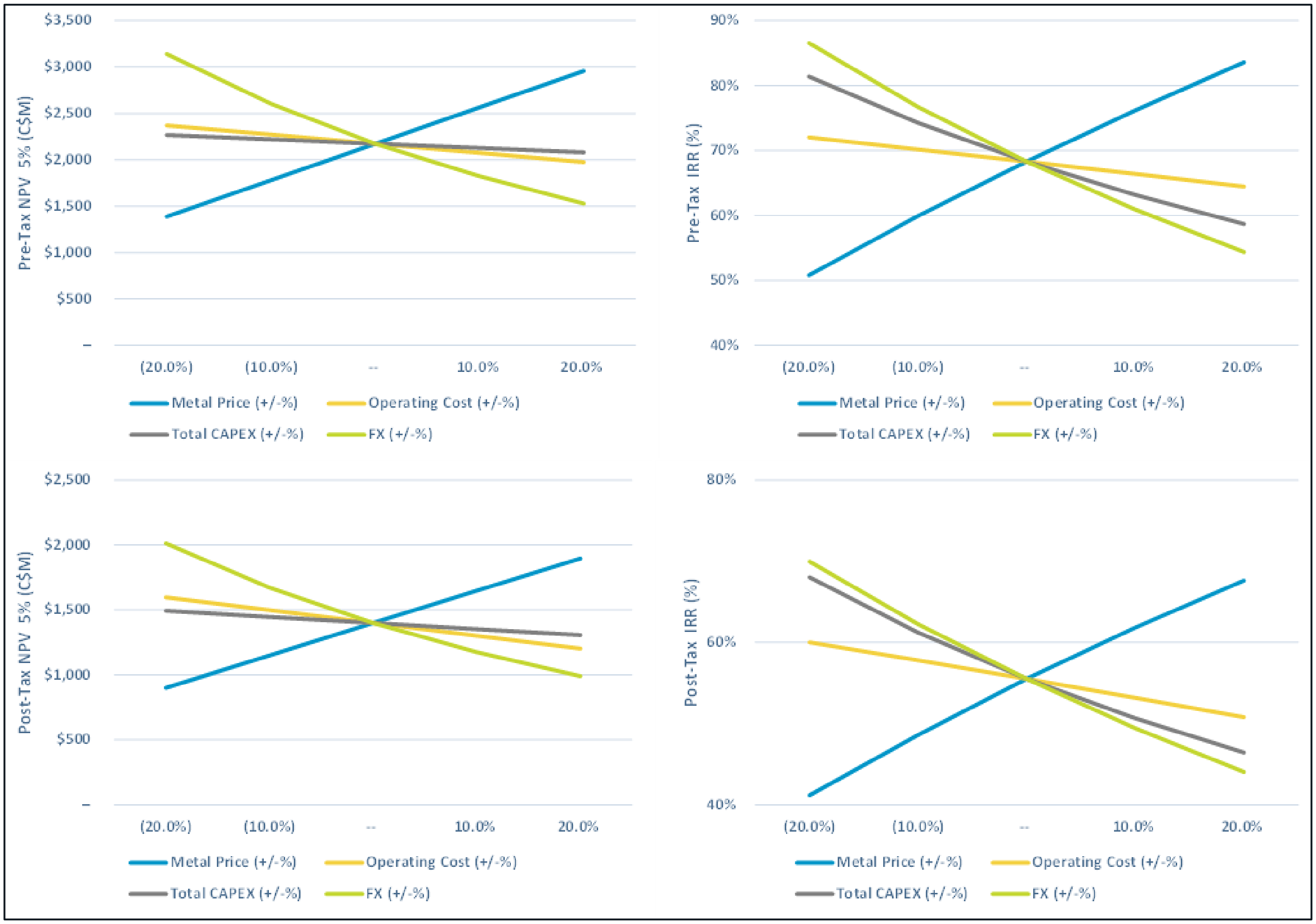
1.22 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, foreign exchange rate, capital costs, and operating costs. The project economics are less sensitive to head grades due to the impact of variable mineralogy, lower concentrate grades and penalty elements on concentrate net smelter returns. Table 1-7 summarizes the sensitivity analysis results. Figure 1-2 shows the pre-tax sensitivity analysis findings.

Table 1-7: Sensitivity Analysis Summary

Sensitivity Summary		Lower Case	Base Case	Higher Case
Gold Price (US$/oz)		1,400	1,550	1,700
Silver Price (US$/oz)		20	22	24
After-Tax NPV(5%) (C$M)		1,162	1,399	1,635
After-Tax IRR (%)		48.9	55.5	61.5
After-Tax Payback (years)		1.6	1.4	1.2
After-Tax NPV / Initial Capex		2.4	2.9	3.4
Average Annual After-tax Free Cash Flow (year 1-10) (C$M)		$231	265	300

Figure 1-2: NPV & IRR Sensitivity Results



Note: Figure prepared by Ausenco, 2021.

1.23 Risks and Opportunities

1.23.1 Risks

1.23.1.1 Geology and Resource Modelling

The suite of deleterious elements (arsenic, mercury, and antimony) that are associated with gold and silver mineralization require additional assays to fully understand the impacts on revenue and the environment.

1.23.1.2 Mining

Mining through voids during open pit operations is a generally manageable risk where such voids are known to exist. However, unidentified voids may exist, and present a risk to mine and production plans if alternate schedules have to be derived, or new safety measures implemented.

It is probable that unfavorably oriented geological structures are present locally within various slope pit sectors, particularly given the size and extents of the pit and the observed variability in discontinuity orientations. Seismic loading and multi-bench-scale to pit-scale structures have the potential to significantly affect overall pit slope stability. The current status and impact of these features are largely unknown. Additional geotechnical investigations are warranted.

The sampling program designed to segregate PAG and NAG waste rock must be adhered to during mining operations to minimize economic and water quality impacts.

The WRSF design assumes that no geotextile liner will be required. If, with further data, such a liner is required, this will affect the mining capital cost estimate.

More detailed geotechnical information is required to support the assumption in the 2021 PFS that mining will extend across Tom Mackay Creek. These data will be used to develop a more detailed water diversion tunnel design and strategy. Geotechnical information may require realignment of the tunnel to avoid potentially problematic material or need additional support requirements which may alter the cost attributed to the tunnel. There is a risk that this design could result in mining capital and operating cost increases.

Detailed operating procedures will need to be established to ensure the PAG rock exposure to air is minimized when placing PAG material into the TMSF.

The support equipment fleet will be responsible for the usual road, pit and WRSF maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management. This is considered an important, but manageable operating risk to meet production targets.

1.23.1.3 Process

The process design as assumed for the 2021 PFS has some risks that could be mitigated by additional testwork and studies. Areas where additional testwork is required include:

- Additional variability samples should be tested to ensure there is a reasonable 3D range of test results covering the mineralization that will be mined in the envisaged LOM open pit mine plan;
- Piloting data should be obtained to confirm that the DFR cells will perform as projected;

- Additional flotation data should be collected to confirm that the low pulp densities required for successful DFR operation can be achieved;

- Sufficient data on settling kinetics should be obtained to support DFR performance assumptions. This should include flocculant optimization studies.

The mill feed will require close management of deleterious elements and the effect on resulting mill performance.

Depending on results, the testwork could indicate that the selected DFR parameters are too optimistic for the mineralization to be treated, or that there can be improvements to the assumptions in the 2021 PFS resulting in lower operating costs and better recovery performance.

The smelter terms that can be obtained over the LOM, including payability and penalty assumptions, are likely to be more complex than currently represented in the 2021 PFS. A focused study that will evaluate projected concentrate grades (payable and penalty) and recovery forecasts is required to provide additional support for assumptions as to smelter terms that would be available to the project. This study could impact project economics negatively if the study indicates higher penalties or less favourable terms than assumed in the 2021 PFS. Conversely, there could be a positive impact if lower penalties and more favourable terms are indicated than assumed in the 2021 PFS.

Solid/liquid separation issues could increase process costs due to larger thickeners and filters and use of flocculant.

Higher mass pull to final concentrate might result without careful control of the grinding pulp chemistry (e.g., stainless-steel media).

1.23.1.4 Infrastructure

A portion of the access road passes through topography which is known to have an elevated geohazard (e.g., avalanche) risk. There is potential for geohazard events to temporarily halt movement along the access corridor. This could affect supply logistics and could result in temporary halts to mining and/or processing operations.

Until there is an agreement in place to connect the powerline at Volcano Creek, there is a risk that the power would have to come from onsite liquified natural gas power generation or a powerline from Bob Quinn which is farther away than Volcano Creek. This would affect the power capital and operating cost assumptions as envisaged in the 2021 PFS.

A WTP at the discharge of the TSF has not been included in the scope of the PFS. Further testing will be done in the next phase to confirm there is no requirement for water treatment. If required, this would affect the water-related capital and operating cost assumptions in the 2021 PFS.

A PAG waste rock deposition plan into the TMSF was developed for the 2021 PFS. A detailed operating procedure will need to be established in the next phase to ensure the PAG waste rock exposure time to air is minimized when placing PAG material into the TMSF to prevent acidification and metal leaching. A change in the deposition plan for the PAG waste rock could result in capital and operating cost increases.

Additional testing is required to understand settling times of tailings and fine particle material from the waste rock placed into the TMSF. Currently, the practice is to place the tailings at the south end of the facility to allow additional settling time. The waste rock deposition plan has assumed the fine-grained particles will settle more quickly. Currently, there is a turbidity fence to reduce the potential for turbid waters to discharge from the facility. If the settling tests indicate that additional measures are required to prevent suspended solids from migrating downstream, this could result in capital and operating cost increases. Rock and sand filters toward the north end could be installed if additional findings during more detailed studies warrant that.

Deposition of the PAG waste rock during winter operations needs further study to ensure there is a sufficient ice-free zone on the causeway to deposit the PAG waste rock. The dragline or an additional dragline could be used to keep an ice-free zone for deposition operations; however, this would result in capital and operating cost increases.

1.23.1.5 Environmental, Permitting, and Social

The permitting timeline assumptions may be affected by recent legislative changes at the provincial and federal levels. An extended permitting process would result in changes to the assumptions made in the economic analysis.

Agreements remain to be negotiated with Indigenous Peoples that may be affected by the Project. If such agreements include royalty or similar payments, this could result in changes to the assumptions made in the economic analysis.

The current permits for the Eskay Mine do not consider operations at the scale contemplated in the 2021 PFS. Additional work will be required to support permit updates and amendment applications, which will include environmental baseline data collection, environmental assessment and proposed mine plan and reclamation and closure plan.

The Project is within the territories of Indigenous groups. Agreements with such groups that may be affected by the envisaged project remain to be negotiated.

1.23.2 Opportunities

1.23.2.1 Exploration

Exploration activities may result in definition of additional mineralization that could support Mineral Resource estimates.

Upside potential for further mineralization discoveries exists within the Lower Mudstone and Even Lower Mudstone units in the Lower Package. These units are typically at depth below the current limits of the proposed open pit.

1.23.2.2 Resource Estimation

There is upside Project potential if mineralization currently classified as Inferred can be upgraded to higher confidence categories. There is also potential for mineralization that is currently outside the estimate boundaries, or discovery of previously unknown mineralization, to be included in estimation with support of drilling and testwork.

1.23.2.3 Mining

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value. Additional hardness testing is likely to be available in the next study stage to inform more detailed throughput management and potentially higher value.

There is potential for improved slope design, when additional geotechnical data such as waste rock strength and joint orientations, are available from drill testing. Steeper pit slopes would reduce the cost associated with waste stripping and provide an opportunity to improve economics.

Slightly higher bench heights could provide an opportunity to better match blasting performance with mine productivity. This will be dependent on the ability to separate ore near underground workings. Higher mine production rates could result in lower mine operating costs and also lower risk to the achieve the mine schedule.

As the metallurgical and marketing information is better understood, the use of stockpiles will likely be modified to allow for improved blending of mill feed material. Stockpile space is fairly limited near the crusher, so a location for lower value material would be useful to ensure high value stockpiles have adequate capacity. This could result in better process performance and improved project economics.

Ongoing test work results will be monitored to see if a portion of the PAG waste material can be effectively neutralized by blending with NAG waste. The ability to blend a portion of this material could result in less PAG material being sent to the TMSF and therefore lower waste haulage and deposition costs.

1.23.2.4 Process

Higher gold and silver recoveries may be obtained from lower head grade samples with optimized flotation conditions.

Pre-concentration by screening and/or bulk sorting might reject waste material and increase plant feed grade. Including gravity concentration may provide an opportunity to remove any free gold early in the process direct to final concentrate.

Investigations into the geometallurgy may lead to flowsheet optimization for the various geological and mineralogical zones for the comminution, flotation, and regrinding configurations.

Incorporation of gravity concentration in the grinding circuit may provide an opportunity to remove free gold prior to flotation and direct it to final concentrate. This may potentially increase gold recovery and reduce operating costs.

An improved Project schedule may be achievable due to shortened equipment leads times, fewer bulk materials, and resulting reduction in construction and installation of the DFR cells. A reduction in the Project schedule will reduce capital costs.

Albino Lake is a subaqueous repository for mine waste rock and tailings used by the previous operators. Initial drilling has indicated elevated gold values in this material. If testwork shows that the gold can be economically extracted, this material could be incorporated into the mine plan and potentially result in an improvement in Project economics.

1.23.2.5 Infrastructure

The TMSF has significant expansion capability (> 20Mm3 of waste materials) if additional mineralization that could support incorporation in the mine plan is discovered. The capital and operating costs would be significantly less than constructing a new storage facility.

1.23.2.6 Environmental, Permitting, and Social

Potential environmental and social opportunities within this Project include the following:

- Collaboration with Indigenous Peoples to develop the Project Closure and Reclamation Plan to meet long term Indigenous End Land Use objectives will gain support for the Project and reduce post-closure cost estimate uncertainty;

- Rationalization of regulatory timeframes in a project charter agreement with regulators and Indigenous peoples can support predictable Project permitting timelines in parallel with testing programs and site development;

Geochemical baseline studies to refine NAG/PAG classifications and material segregation may help optimize waste management costs, design, and complexity. Commenced early, geochemical studies improve regulator confidence in modelled outcomes of post-closure environmental management.

- Assessment of energy efficiencies and fleet/machinery composition may present opportunities to reduce emissions over mine life;

- Incorporation of Indigenous perspectives and values on how mining development occurs on the landscape and its effects on land and water will be pursued throughout the Project life resulting in a Project viewed as meeting sustainability goals by Indigenous communities.

1.23.2.7 Marketability

There is upside potential for the Project if the planned drill programs more comprehensively document deleterious elemental distributions such that the levels of these elements, in particular arsenic and mercury, can be minimized in the concentrate to below smelter penalty thresholds.

1.24 Interpretation and Conclusions

Based on the assumptions and parameters presented in this Report, the 2021 PFS shows positive economics. The 2021 PFS supports that more detailed studies are warranted.

1.25 Recommendations

The recommended work program is divided into two phases. The phases can be conducted concurrently, but some portions of the phase 1 work plan would be incorporated into the phase 2 recommendations.

The first recommendations phase totals about $11.49 M. Recommendations consist of drilling; determination of whether bulk ore-sorting could potentially be implemented at the pre-mining stage; a study to determine if a relationship between rock mass structure and head grade exists; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second phase is estimated at about $4.6 M, and will consist of project environmental, permitting, and social de-risking activities, including baseline and targeted environmental studies, negotiation of agreements with Indigenous groups, stakeholder engagement, an environmental assessment, application for operating permits, and an updated water balance to better understand makeup requirements, distribution of site flows, site water quality and water treatment requirements.

2 INTRODUCTION

2.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco), Hemmera Envirochem Inc., an Ausenco company (Hemmera), SRK Consulting (Canada) Inc. (SRK), and AGP Mining Consultants Inc. (AGP), prepared a technical report (the Report) on the results of a pre-feasibility study (2021 PFS) for Skeena Resources Limited (Skeena) on the volcanogenic massive sulphide (VMS) Eskay Creek Project (the Project) located in British Columbia (Figure 2-1).

Skeena owns 100% of the Eskay Creek gold-silver project, where it was purchased from Barrick Gold in October 2020.

The Project hosts the previously-mined Eskay Creek deposit, which was in operation as an underground mine from 1995–2008.

2.2 Terms of Reference

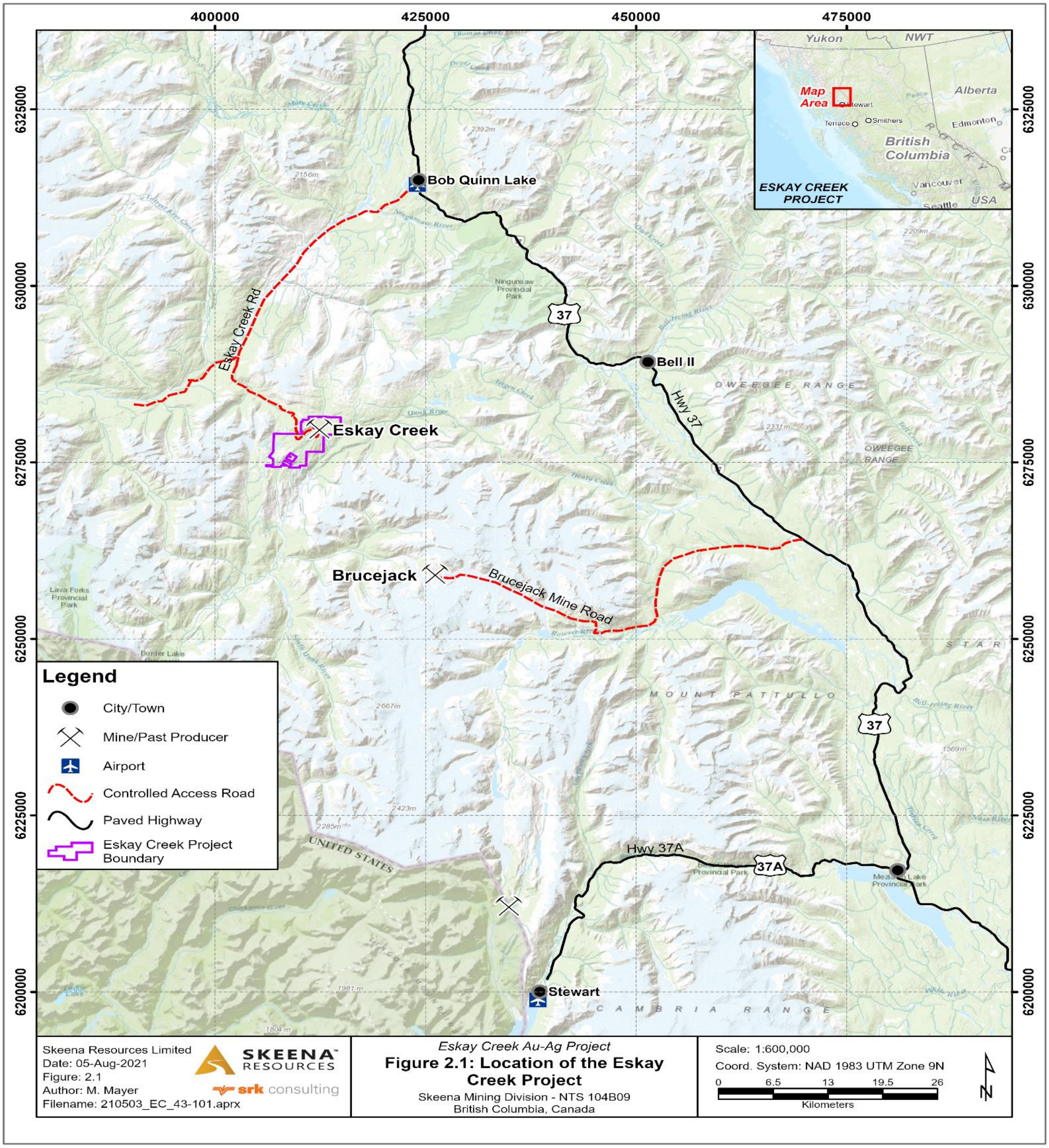
The Report supports disclosures by Skeena in a news release dated 22 July 2021 entitled “Skeena Completes PFS for Eskay Creek: After-Tax NPV(5%) of C$1.4B, 56% IRR and 1.4 Year Payback”.

All measurement units used in this Report are metric unless otherwise noted. Currency is expressed in Canadian (C) dollars (C$). The Report uses Canadian English.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; the 2019 CIM Best Practice Guidelines).

As the ownership of, and ownership interests in, the historical mining operations changed hands numerous times during the production history (refer to Section 6), the Report uses the term “previous operator” to refer to work done from 1988 to 2017. The term “legacy” is used for data generated by the previous operator. Skeena obtained its option interest in December 2018.

Figure 2-1: Project Location Plan



Note: Brucejack Mine is owned by third parties.

2.3 Qualified Persons

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

- Mr. Tommaso Robert Raponi, P.Eng., Principal Metallurgist, Minerals & Metals, Ausenco;
- Mr. Scott Elfen, P.E., Global Lead Geotechnical Service, Ausenco;
- Ms. Sheila Ulansky, P.Geo., Senior Resource Consultant, SRK;
- Mr. Rolf Schmitt, P. Geo., Technical Director - Permitting, ERM;
- Dr. Adrian Dance, P.Eng., Principal Metallurgist, SRK;
- Mr. Willie Hamilton, P.Eng., Senior Mining Engineer, AGP;
- Mr. Roland Tosney, P.Eng., Principal Mine Engineer, AGP.

2.4 Site Visits and Scope of Personal Inspection

Ms. Ulansky, representing SRK, visited the Eskay Creek property from 27–28 June 2018. During that visit she viewed the general topography, independently located and surveyed 50 surface drill hole collars, and inspected the existing mine infrastructure.

Mr. Hamilton, representing AGP, visited the Eskay Creek site from 21–22 August 2019. On 21 August, he travelled by vehicle and observed the existing site facilities, active exploration drilling sites, the Tom Mackay tailings storage facility (TMSF) and the Albino Lake storage facility (SF) site. On August 22, Mr. Hamilton toured the site via helicopter and also spent time reviewing drill core in the core shed.

Mr. Elfen, representing Ausenco, completed a site visit on June 19, 2019, October 27 and 28, 2020 and July 14 and 15, 2021. The objectives of the site visits were to review the site wide geotechnical programs to ensure the two programs were meeting the proposed objectives for the PFS and FS, siting of the TMSF and WRSF. In addition, reviewed the general topography and geotechnical surface conditions for the site wide infrastructure to ensure there were no significant geotechnical issues.

Mr. Tosney, representing AGP, completed a site visit on July 18 - 22, 2020 for a visit duration of five days, and October 27, 2020, for a visit duration of one day. The objectives of the site visits were to meet with Skeena geology and exploration staff, review open pit geotechnical drill plans and status, collect and compile site geological and geotechnical data, complete domain-scale geotechnical logging of select intervals of drill core, and conduct geotechnical mapping and rock mass characterization focused on verifying and supplementing existing information, including lithology, rock mass strength, and discontinuity characteristics.

Mr. Schmitt, representing ERM, on July 13, 2019, completed an aerial reconnaissance by helicopter, of the Tom Mackay tailings storage facility (TMSF), access road, and mine site area. The objective of the aerial reconnaissance was to gain an understanding of the Project location and environmental setting.

2.5 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of last information on mineral tenure, surface rights and agreements: 9 July, 2021;
- Date of supply of most recent information on ongoing drill program: 9 March, 2021;
- Mineral Resource estimate: 7 April, 2021;
- Date of PFS financial analysis: 22 July, 2021

The overall effective date of this Report is the effective date of the financial analysis which is 22 July 2021.

2.6 Information Sources and References

The Report is primarily based on a preliminary economic assessment prepared for Skeena in June–November 2019 and supporting memoranda and trade-off studies. This Report is also based in part on internal company reports, maps, published government reports, and public information.

Additional information was sought from Skeena employees in their areas of expertise as required.

2.7 Previous Technical Reports

Skeena has filed the following technical reports on the Project:

- Ulansky, S., and Carlson, G., 2021: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 7 April, 2021;
- Kalanchey, R., Elfen, S., Weston, S., Ulansky, S., Dance, A., Zurowski, G., and Hamilton, W., 2019: NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared by Ausenco Canada, Hemmera Envirochem Inc., SRK Consulting (Canada) Inc. and AGP Mining Consultants Inc. for Skeena, effective date 7 November, 2019;
- Ulansky, S., Uken, R., and Carlson, G., 2019: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 28 February 2019;
- Ulansky, S., Uken, R., and Carlson, G., 2018: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 6 July 2018.

2.8 Unit Abbreviations

Abbreviation	Unit
3D	Three-Dimensional
°C	degrees Celsius
C$	Canadian dollar
cm	centimetre
%	percent

%w/w	dry weight concentration of a solution
µ	micro
µm	micrometre
g	gram
g/cm^3	Grams per centimetre cubed
g/t	grams per tonne
ha	hectare
HP	horsepower
hr	hour
Kg	kilogram
Km	kilometre
Koz	thousand ounces
kt/d	thousand tonnes per day
kV	kilovolt
kWh	Kilowatt hour
L/s	litre per second
M	million
m	metre
m^2	square metre
m^3	cubic metre
masl	metres above sea level
mamsl	Metres above mean sea level
Mg/L	milligrams per liter
mm	millimetres
Mt	million tonnes
Mt/a	million tonnes per annum
mV/V	millivolts per volt
MW	Megawatt
MWh	Megawatt hour
oz	ounce
P$_{80}$	Passing grind size
ppm	parts per million
ppb	parts per billion
t	metric tonne
t/d	tonnes per day
t/m^2/hr	tonnes per metre squared per hour
US$	United States dollar

X	times

2.9 Name Abbreviations

Abbreviation	Name
AAS	Atomic Absorption Spectroscopy
AES	Atomic Emission Spectroscopy
Ag	Silver
AGP	AGP Mining Consultants Inc.
As	Arsenic
ATV	All terrain Vehicle
Au	Gold
AuEq	Gold Equivalent
Barrick	Barrick Gold Inc.
BaseMet	Base Metallurgical Laboratories Ltd.
BC	British Columbia
BCEAA	British Columbia Environmental Assessment Act
Blue Coast	Blue Coast Research
BOO	Build Own Operate
BOOT	Build Own Operate and Transfer
BWi	Bond ball mill work index
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CM	Construction Management
CMC	Sodium silicate and carboxymethyl cellulose
Corg	Organic Carbon
Cu	Copper
$CuSO_4$	Copper Sulphate
CV	Coefficient of Variation
CVR	Common Voltage Reference
D	Disturbance Factor
Dias	Dias Geophysical Limited
DCIP	DC Resistivity and Induced Polarization
DFR	Direct Flotation Reactor
DGPS	Differential Global Positioning System
DTH	Down the hole Hammer
DWi	Mill Comminution
EA	Economic Assessment

Abbreviation	Name
E-GRG	Extended gravity recoverable gold
EM	Electromagnetic
EP	Engagement Plan
EPCM	Engineering Procurement and Construction Management
Fe	Iron
FEL	Front-End Loader
FOS	Factor of Safety
GPS	Global Positioning System
GVW	Gross Vehicle Weights
Hg	Mercury
HMI	Human Machine Interface
HW	Hanging Wall
HWA	Hanging wall Andesites
IAA	Impact Assessment Act
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ID	Inverse distance
IP	Induced Polarization
IPD	Initial Project Description
IPL	Independent Plasma Laboratories
IRR	Internal Rate of Return
KV	Kriging Variance
LBMA	London Bullion Market Association
LDL	Low detection limit
LiDAR	Light detection and Ranging
LNG	Liquefied Natural Gas
LOM	Life of Mine
LRS	Liquid Resistance Starter
M+I	Measured and Indicated
MAP	Mean Annual Precipitation
Max.	maximum
McElhanney	McElhanney Consulting Services Ltd.
Min.	minimum
MRE	Mineral resource estimate
MSE	Mechanically Stabilized Earth

Abbreviation	Name
MTO	Mineral Titles Online
No.	number
NAG	Non-acid generating
NN	Nearest Neighbour
NPV	Net Present Value
NSG	Non-sulphide gangue
NSR	Net Smelter Return
OK	Ordinary kriging
OR	Ordinary Kriging
PAG	potentially acid generating
PAX	Potassium Amyl Xanthate
Pb	lead
PEA	Preliminary Economic Assessment
PFS	Prefeasibility Study
PID	Proportional-Index-Derivative
PLC	Programmable Logic Controllers
PM	Project Management
PMF	Probable Maximum Flood
POX	Pressure Oxidation
QA/QC	Quality Assurance
QC	Quality Control
QP	Qualified Person
R&M	Repair and maintenance
RBF	Radial basis function
RDKS	Regional District of Kitimat-Stikine
RMR	Rock Mass Rating
ROM	Run-of-Mine
RQD	Rock Quality Design
RWi	Bond rod mill work index
S	Sulphur
SAG	Semi-Autonomous Grinding
Sb	antimony
SBT	Stewart Bulk Terminal
SER	Slip Energy Recovery
SG	Specific Gravity
Skeena	Skeena Resources Limited

Abbreviation	Name
SPI	SAG Power Index
SRK	SRK Consulting Canada Inc.
SRM	Standard Reference Materials
SWP	Stewart World Port
TDS	Total Dissolved Solid
TMSF	Tom MacKay Tailings Storage Facility
TSF	Tailings Storage Facility
TSKLH	Tsetsaut Skii Km Lax Ha
VAT	Value Added Tax
VLF	Very Low Frequency
VMS	Volcanogenic massive sulphide
WDN	Waste Dump North
WDNE	Waste Dum Northeast
WDW	Waste Dump West
WRIM	Wound rotor drive motor
WRSF	Waste Rock Storage Facility
WTP	Water Treatment Plant
Zn	zinc

3 RELIANCE ON OTHER EXPERTS

3.1 General

Ms. S. Ulansky (QP) relied upon the expertise of Ms. Kathi Dilworth (of Skeena Resources) for Section 14 of this Technical Report. Ms. Dilworth completed the majority of the report relating to the Mineral Resources, including the construction of the report figures and tables. Ms. Ulansky relied on the opinions and statements of Skeena Resources personnel in relation to information concerning legal, environmental liabilities, political, or other issues and factors relevant to the technical report.

3.2 Markets

Mr. Raponi (QP) has not independently reviewed the marketing, smelter terms, or metal price forecast information. Mr. Raponi has fully relied upon, and disclaims responsibility for, information derived from experts retained by Skeena.

Skeena engaged several internationally known consultants, expert in the marketing of complex gold concentrates. Each consultant conducted their own studies and provided opinions on potential smelters, treatment charges and penalties and net gold and silver payable. The consultants' reports were made available to the QP, and several meetings were held with the consultants that the QP participated in. In the opinion of the QP, the reports are suitable for use in this study and the selected smelter terms accurately reflect the potential treatment charges, penalties and net smelter returns for the Eskay Creek concentrates.

This information is used in Section 19, and in support of the financial analysis in Section 22.

Metals marketing, global concentrate market terms and conditions, and metals forecasting are specialized businesses requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive database that is outside of the purview of a QP.

Mr. Raponi considers it reasonable to rely on such information as the consultants are specialist in commodities trading. Detailed information outlining all payables, penalties, deductions, and charges, was provided to arrive at estimated value of concentrate. It also provides information for sellers, buyers, and finance providers to understand the value of each material based on the current market terms.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Eskay Creek Project is located in the Golden Triangle region of British Columbia, Canada, 83 km northwest of Stewart, on the eastern flanks of the Coast Mountain ranges. The Project is situated at an elevation of 800 m above sea level at 56° 39’ 13.9968” N and 130° 25’ 44.0004” W.

4.2 Mineral Tenure

The status of all mining titles was checked using Mineral Titles Online (MTO), the British Columbia government’s online mineral titles administration system.

The Eskay Creek Project covers a total of 5,745.49 hectares and consists of the following (Figure 4-1):

- Forty-seven mineral claims totalling 3,915.23 hectares (Table 4-1);
- Eight mineral leases totalling 1,830.26 hectares (Table 4-2).

Forty-five mineral claims are 100% registered to Skeena Resources Limited, and two mineral claims are held 66.67% Skeena Resources Limited, and 33.33% are held by Canarc Resource Corp. Five mineral leases are 100% held by Skeena Resources Limited, and three mineral leases are held 66.67% Skeena Resources Limited and 33.33% are held by Canarc Resource Corp.

Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under the BC *Mineral Tenure Act* Regulation.

4.3 Property Agreements

On December 18, 2017, Skeena and Barrick entered into an Option Agreement on the Eskay Creek Project. This agreement affects all mineral claims and mineral leases that comprise the Eskay Creek Project, except for the single mineral claim registered to Skeena Resources Ltd.

On October 2, 2020, Skeena and Barrick agreed to amend the terms of the original option agreement on the Eskay Creek property. Skeena acquired 100% ownership of Eskay Creek in consideration for:

- The issuance to Barrick of 22.5 million units, comprising one common share of Skeena and a non-transferable half warrant;
- The grant of a 1% net smelter return (NSR) royalty on the entire Eskay Creek land package. Half of that royalty may be purchased from Barrick during the 24-month period after closing, at a cost of C$17.5 million;

- A contingent payment, payable if Skeena sells more than a 50% interest in Eskay Creek during the 24-month period after closing, of C$15 million.

Table 4-1: Mineral Claim Summary

Tenure Number	Claim Name	Description	Issue Date	Good to Date	Title Protection Expiry Date	Area (Hectares)	Owner Name	Number of Owners
252966	CAL #2	CLAIM	1989-08-05	2021/JAN/15	2021/DEC/31	500	Skeena	2
252967	CAL #3	CLAIM	1989-08-06	2021/JUN/22	2021/DEC/31	400	Skeena	2
252976	IKS 2	CLAIM	1989-08-02	2025/JUL/12		500	Skeena	1
300298	P-1	CLAIM	1991-06-11	2021/MAY/20		25	Skeena	1
300299	P-2	CLAIM	1991-06-11	2021/MAY/20		25	Skeena	1
300300	P-3	CLAIM	1991-06-11	2021/MAY/20		25	Skeena	1
300301	P-4	CLAIM	1991-06-11	2021/MAY/20		25	Skeena	1
329241	MACK 23	CLAIM	1994-07-21	2025/JUN/25		500	Skeena	1
329244	MACK 1	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329245	MACK 2	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329246	MACK 3	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329247	MACK 4	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329248	MACK 5	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329249	MACK 6	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329252	MACK 9	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329253	MACK 10	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329254	MACK 11	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329255	MACK 12	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329256	MACK 13	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329257	MACK 14	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329258	MACK 15	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329259	MACK 16	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329260	MACK 17	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329261	MACK 18	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329262	MACK 19	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329263	MACK 20	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329264	MACK 21	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329265	MACK 22	CLAIM	1994-07-21	2025/JUN/25		25	Skeena	1
329363	MACK 26 FR.	CLAIM	1994-08-03	2025/JUN/25		25	Skeena	1
352974	STAR 21	CLAIM	1996-12-07	2021/JUN/22	2021/DEC/31	250	Skeena	1
352975	STAR 22	CLAIM	1996-12-07	2025/JUN/25		150	Skeena	1
365539	KAY 1	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365541	KAY 3	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365542	KAY 4	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365543	KAY 5	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365544	KAY 6	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365545	KAY 7	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365546	KAY 8	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365547	KAY 9	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
365548	KAY 10	CLAIM	1998-09-12	2025/OCT/06		25	Skeena	1
512867	<Null>	CLAIM	2005-05-17	2021/JUN/25	2021/DEC/31	106.8	Skeena	1
512879	<Null>	CLAIM	2005-05-18	2021/APR/06	2021/DEC/31	35.58	Skeena	1
512881	<Null>	CLAIM	2005-05-18	2021/JUN/25	2021/DEC/31	17.8	Skeena	1

Tenure Number	Claim Name	Description	Issue Date	Good to Date	Title Protection Expiry Date	Area (Hectares)	Owner Name	Number of Owners
1037725	ESKAY CREEK MAC 25	CLAIM	2015-08-04	2021/OCT/04		338.3283	Skeena	1
1041101	ESKEY CREEK TREND	CLAIM	2016-01-09	2021/SEP/10		124.4705	Skeena	1
1041102	ESKEY CREEK 1983 FILE	CLAIM	2016-01-09	2021/JUL/10		88.9027	Skeena	1
1056639	MELISSA	CLAIM	2017-11-24	2020/OCT/06		53.35	Skeena	1

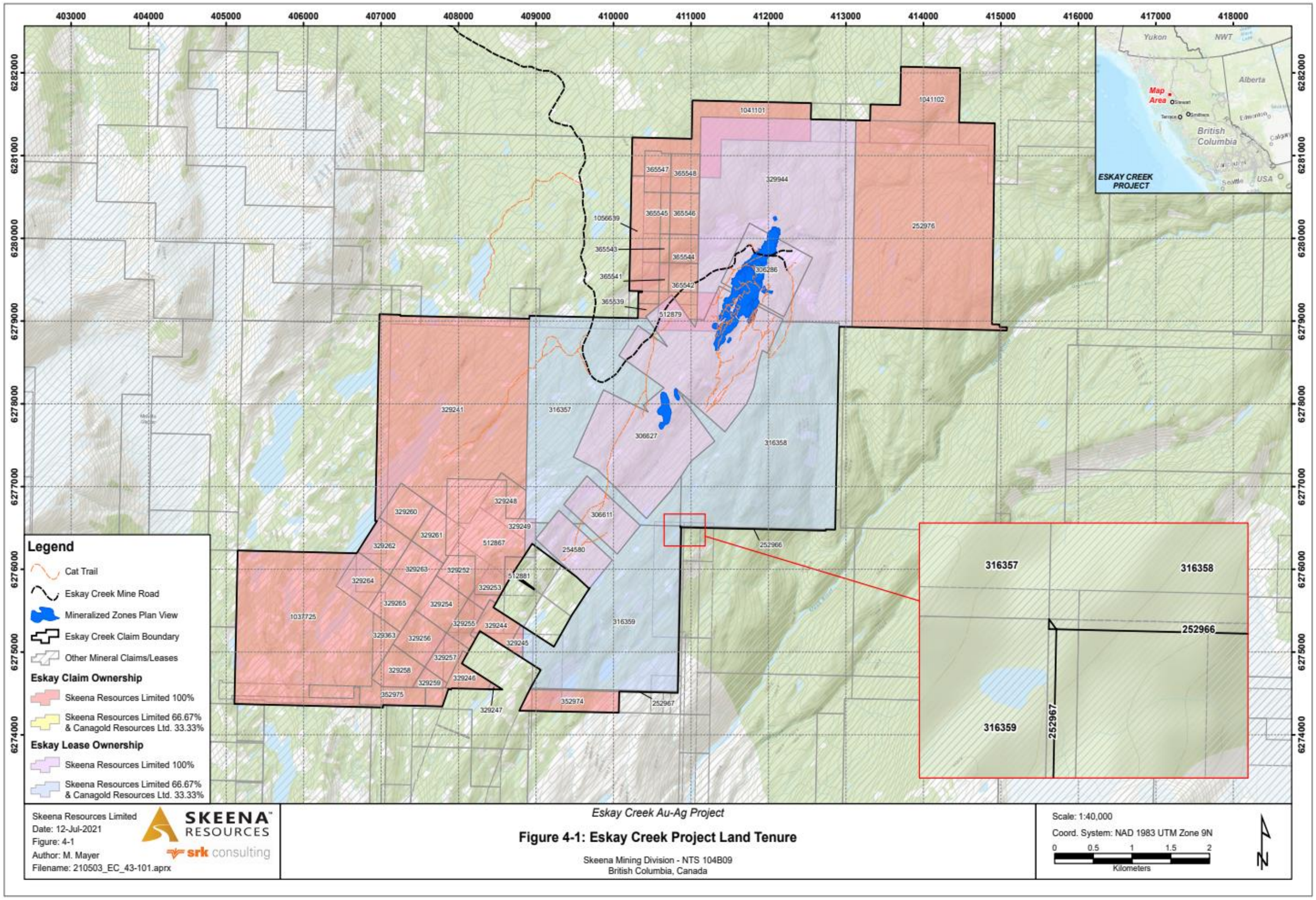
*Title Protection was applied to all claims due to COVID which extends all dates to an expiry date of Dec. 31, 2021.

Table 4-2: Mineral Tenure Summary

Tenure Number	Issue Date	Good to Date	Title Protection Expiry Date*	Area (Hectares)	Owner Name	Percent Ownership	Number of Owners
254580	1990-12-17	2020/DEC/17	2021/DEC/31	41.8	Skeena	100	1
306286	1991-08-13	2021/AUG/13	2021/DEC/31	73.56	Skeena	100	1
306611	1992-06-01	2021/JUN/01	2021/DEC/31	41.8	Skeena	100	1
306627	1992-06-01	2021/JUN/01	2021/DEC/31	355	Skeena	100	1
316357	1994-04-30	2022/APR/30		276.7	Skeena	66.67	2
316358	1994-04-30	2022/APR/30		367.7	Skeena	66.67	2
316359	1994-04-30	2022/APR/30		278.7	Skeena	66.67	2
329944	1994-12-06	2021/DEC/06	2021/DEC/31	395	Skeena	100	1

*Title Protection was applied to all claims due to COVID which extends all dates to an expiry date of Dec. 31, 2021

Figure 4-1: Mineral Tenure Location Plan



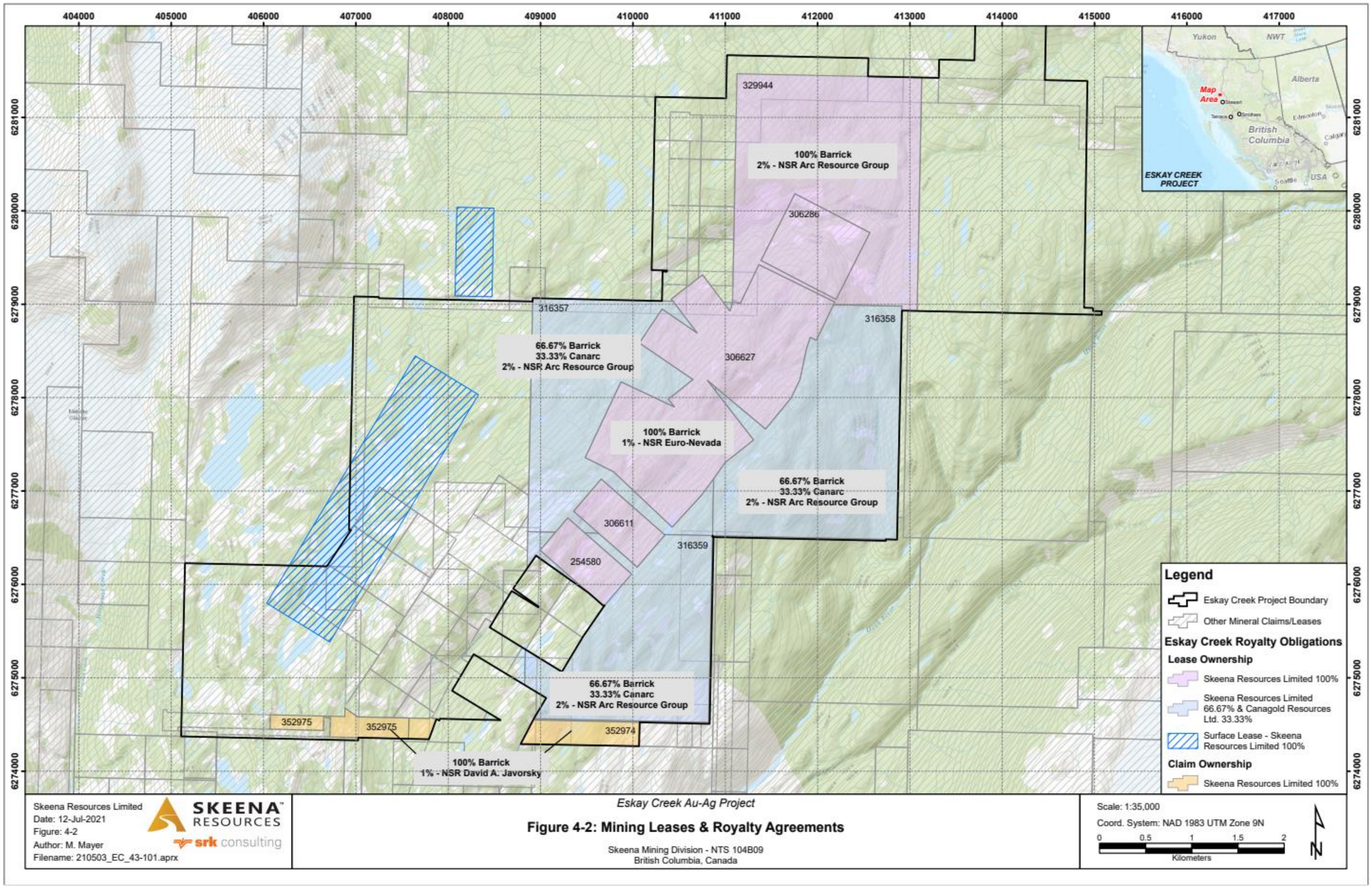
4.4 Surface Rights

Skeena holds the following surface rights interests:

- Surface lease number 634309: dated 24 December 1994 between the Province of BC and Prime Resources Group Inc.; interest assigned to Skeena;
- Surface lease number 740715: dated 25 July 2004 between the Province of BC and Optionor; interest assigned to Skeena;
- Special Use Permit S17635: for the use of the Eskay Creek road.

The locations of the surface leases are provided in Figure 4-2.

Figure 4-2: Mineral Tenure Plan Royalty Interests



District Lots underly the Eskay Creek tenures, and a title search indicates that there are no mineral or surface rights associated with the District Lots. Skeena will need to acquire surface rights in support of any future mining and processing activities.

Permit amendment for Surface Lease 740715 will be required to extend the boundary to include the surface area associated with the south end of the Tom Mackay tailings storage facility (TMSF).

4.5 Water Rights

Skeena currently holds two water licences:

- Conditional Water Licence 1017796: dated 2 March 1994 between the Province of BC and Prime Resources Group Inc.; interest assigned to Skeena on October 9, 2020;
- Conditional Water Licence 114327: effective 20 April 1999 between the Province of BC and Homestake Mining Company; interested assigned to Skeena on October 9, 2020.

Skeena anticipates needing to apply for additional Water Licences under the BC Water Sustainability Act for the proposed Project. Specifically, the following Water Sustainability Act authorizations will include:

- Section 2 Groundwater Well Registration and Groundwater Usage;
- Section 9 Authorization for Diversion and Use of Water;
- Section 10 Short Term Water Use;
- Section 11 Authorization for Working on or About Streams.

4.6 Royalties and Encumbrances

The Eskay Creek Project has NSR royalty obligations on five properties payable to third parties as shown in Table 4-3. The locations of the claims with royalty obligations were shown in Figure 4-2.

Table 4-3: Summary of Eskay Creek Project Royalty Obligations

Parcel	Royalty
Kay-Tok Property • Kay Mining Leases • Tok Mining Leases	1% NSR in favour of Franco-Nevada Corp. (1) w/o duplication of the following and depending on the handling of the product: 1% Net Smelter Returns, 1% Net Ore Returns, 1% Net Returns payable from the disposition of the beneficiated product of all metals, minerals and mineral substances. Barrick has the right to first refusal to purchase the royalty. No cap or buyout provision of this royalty.

Parcel	Royalty
IKS Property • IKS 1 Mining Lease • IKS 2 Mining Claim	2% NSR in favour of ARC Resource Corporation (2) Royalty also includes the are known as the IKS Gap. No cap on royalty payments. No buyout provision or rights of first refusal on the sale of the royalty.
GNC Property • GNC 1-3 Mining Leases	2% NSR in favour of ARC Resource Corporation (3) Interest: Barrick 66.67%; Canarc 33.33% No cap on royalty payments. No buyout provision or rights of first refusal on the sale of the royalty.
Star Property • Star 21, 22 • Silver West Mining Claims	1% NSR in favour of David A. Javorsky (4) No cap on royalty payments. The Option of Purchase the Royalty has expired.
Entire Eskay Creek Land Package	1% NSR in favour of Barrick Gold Corp. (5) Half the royalty may be repurchased from Barrick during the 24-month period after closing at a cost C$17.5 million.

1. Amended and Restated Eskay Creek Royalty Agreement dated May 5, 1995, between Prime Resources Group Inc. (now Barrick) and Euro-Nevada Mining Corporation Limited (now Franco-Nevada Corp.).
2. Transfer and Assignment Agreement dated December 22, 1994, between Prime Resources Group Inc. & Stikine Resources Ltd. (both now Barrick) and Adrian Resources Ltd.

   This agreement references the Royalty Deed dated August 1, 1990, between ARC Resource Group Ltd. And Adrian Resources Ltd.
3. Option and Joint Venture Agreement dated November 4, 1988, between Canarc Resources Corp and Calpine Resources Incorporated (now Barrick).
4. NSR Royalty Agreement w. Option to Purchase dated November 3, 2004, between Homestake Canada Inc. (now Barrick) and David A. Javorsky.
5. Royalty Agreement dated October 2, 2020, between Skeena Resources Limited, and Barrick Gold Inc.

4.7 Permitting Considerations

Permitting is discussed in Section 20.

4.8 Environmental Considerations

Environmental considerations are discussed in Section 20.

Skeena's current environmental liabilities are related to the ownership of the Project site and activities undertaken by Skeena. The key liabilities would be the Project's existing infrastructure, site closure and reclamation activities, and remediation of drill pads and access road. Skeena has posted an environmental bond with the relevant BC authorities in relation to the work programs that have been conducted.

4.9 Social Considerations

Social considerations are discussed in Section 20.

4.10 QP Comments on "Item 4; Property Description and Location"

The QP notes:

- Mineral concessions are valid and in good standing;
- Barrick retains a 1% NSR royalty on tenements otherwise not subject to royalty payments;
- There are underlying royalties payable to third parties on some of the tenures;
- Skeena will need to acquire additional surface rights in support of future mining and processing activities;
- Skeena anticipates needing to apply for additional Water Licences under the BC Water Sustainability Act for the proposed Project.

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

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

Access to the Eskay Creek Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction (refer to Figure 2-1). The Eskay Mine Road is a 59 km private industrial road that is operated by Coast Mountain Hydro Corp. (0 km to 43.5 km) and Skeena (43.5 km to 59 km).

There are two nearby gravel air strips: Bronson Strip which is about 40 km west of the mine site and Bob Quinn air strip, roughly 37 km northeast of the Project alongside Highway 37. Bronson Strip is a private air strip operated by Snip Gold Inc. It is 1,500 m long and in fair condition. It is accessible to the Project by helicopter only. The Bob Quinn Lake air strip is managed by the Bob Quinn Lake Airport Society, a not-for-profit organization consisting of government and local industry interests. The airstrip is about 1,300 m long and is in good condition. It is accessible to the Project by the Eskay Mine Road.

5.2 Climate

The mean annual total precipitation at the former mine site is estimated to be 2,500 ± 500 mm. The majority (55–71%) of annual precipitation falls as snow. Snowpack data collected between 1990–1993 indicated peak snowpack (April) of 1,425 ±567 mm. Cumulative snowfall data at the mine site collected between 1999 and 2006 indicates a range of about 7.5–17.5 m of snow can fall between September and May. A meteorology study for the environmental baseline studies is currently underway to update and support this information.

The average temperature range is from -10.4°C in January to +15°C in July (Environment Canada, 2013). Expected extreme temperatures range from -40°C to +30°C (SRK, 2019).

Exploration activities can be curtailed by winter conditions. The previous mining operation was conducted on a year-round basis, and it is expected that any future operations will also be year-round.

5.3 Local Resources and Infrastructure

Support services for mining and other resource sector industries in the region are provided primarily by the communities of Smithers (pop. 5,400) and Terrace (pop. 12,700). Both communities are accessible by commercial airlines with regular flights to and from Vancouver.

Labour in support of exploration activities can be locally sourced. British Columbia has a long mining history and experienced mining personnel can be found within the Province.

Volume freight service in the region is supported by rail connections that extend from tidewater ports in Prince Rupert and Vancouver. The closest tidewater port to the Project is in Stewart, approximately 260 km from the Project by road. Stewart is an ice-free shipping location and provides year-round access for bulk shipping.

The Project is in proximity to the new 287-kV Northwest Transmission Line operated by BC Hydro and Power Authority and three hydroelectric facilities operated by Coast Mountain Hydro.

Additional information on local resources and infrastructure is provided in Section 18.

5.4 Physiography

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (amsl), located on the eastern flank of the Boundary Ranges. The plateau is characterized by northeast-trending ridges with gently-sloping meadows occupying valleys between the ridges. Relief over the plateau area ranges from 500 m in the existing TMSF area to over 1,000 m in the Unuk River and Ketchum Creek valleys. The former Eskay Creek mine site is at approximately 800 m elevation.

Mountain slopes are heavily forested. Additional information on vegetation is included in Section 20.

Glacial features such as cirques, hanging valleys and over-steepened slopes, are present throughout the Project area. The plateau is surrounded by high serrate peaks containing cirque and mountain glaciers. The surficial geology in the area is varied, and includes till, colluvium at the base of bedrock outcrops and on steep slopes, organics in poorly-drained depressions, and alluvium along streams and the lake shorelines (Ulansky et al., 2018).

The Prout Plateau is drained by tributaries of the Stikine–Iskut and Unuk Rivers. Volcano Creek drains to the north into the Iskut River, a major tributary to the Stikine River system. The remainder of the plateau is drained almost exclusively by the Unuk River and its tributaries: the Tom MacKay, Argillite, Ketchum, Eskay and Coulter Creeks. The gradient of these drainages increases as the creeks descend from the moderate relief of the Prout Plateau into the deeply incised Unuk River valley. The plateau is occupied by the Tom MacKay, Little Tom MacKay and several smaller lakes as well as Argillite Creek, which collectively form the headwaters of the Tom MacKay Creek drainage system.

There are no known federal, provincial, or regional parks, wilderness or conservancy areas, ecological reserves, or recreational areas near the Eskay Creek Project.

The Tahltan Nation has asserted Indigenous title and rights to this area in the Declaration of the Tahltan Tribe in 1910. Previous operators have established formal agreements with the Tahltan Central Government regarding their ongoing participation at the mine site. More recently, Skii km Lax Ha Nation has produced maps indicating that the mine falls within their area of traditional land use (Rescan, 2009).

6 HISTORY

6.1 Exploration History

Table 6-1 is a summary of the known exploration work that had been undertaken on the Eskay Creek Project by various operators since 1932.

Table 6-1: Exploration Summary

Year	Owner	Work Area	Description
1932	Unuk Gold/Unuk Valley Gold Syndicate	Unuk & Barbara Group claims (Core property)	Prospecting
1933	Mackay Syndicate	Unuk & Barbara Claims	Trenching
1934	Mackay Syndicate/Unuk Valley Gold Syndicate	Unuk, Barbara & Verna D. Group Claims	Prospecting and core drilling (261.21 m)
1935-1938	Premier Gold Mining Co. Ltd.	Core property	Optioned property and conducted prospecting, trenching and core drilling (1,825.95 m). Defined and named over 30 mineralized showings
1939	MacKay Gold Mines Ltd.	#13 O.C./Mackay Adit	Financed by Selukwe Gold Mining and Finance Company Ltd. and acquired property. Conducted data review. Underground development of the MacKay Adit (84.12m) which is about 3 km south of the Eskay Creek mine site.
1940-1945			No activity due to World War II
1946	Canadian Exploration Ltd.	Mackay Adit	Optioned property. Conducted mapping and trenching. Underground development extended the Mackay Adit to 109.73 m and put a raise to surface at 46 m
1947-1952	American Standard Mines Ltd. / Pioneer Gold Mines of B.C. Ltd. / New York-Alaska Gold Dredging Corp.	Canab Group (36 claims of the Mackay Group)	Optioned and conducted property examination
1953	American Standard	Canab Group/Mackay Group 36 claims (No. 21, No. 22 & No. 5 areas)	Trenching (2,655.32 m). Open cutting in the 5, 21 and 22 zones. Core drilling (22 holes)

Year	Owner	Work Area	Description
1954-1962	Western Resources Ltd. (Western Resources)	Kay 1-18	Unknown – no work reported
1963	Western Resources	Kay 1-36; Emma Adit	Underground development of the Emma Adit (111.25 m) and road building (13 km) from Tom Mackay Lake to the property
1964	Stikine Silver Ltd. (Stikine Silver) / Canex Aerial Exploration Ltd.	Kay Group; Emma Adit	Optioned from Western Resources Ltd. Mapping, rock, stream, sediment, and soil sampling. Underground core drilling (224.64 m)
1965	Stikine Silver	Kay Group (40 claims); Emma Adit	Trenching (1,457.20 m in 18 trenches); core drilling (15.85 m); and underground development (extended the Emma Adit to 178.61 m)
1966	Stikine Silver		No activity
1967	Mount Washington Copper Co. / Stikine Silver	Kay 1-36 (Core Property)	Electromagnetic (EM 16) and magnetometer geophysical surveys; petrography
1968-1970	Newmont Mining Corp.	Kay 1-8	Surface and underground geological mapping; trenching (137.16 m)
1971-1972	Stikine Silver	22 Zone	Trenching and surface bulk sample
1973	Kalco Valley Mines Ltd.	22 Zone	Surface geological mapping and core drilling (299.62 m)
1974			No activity
1975-1976	Texasgulf Canada Ltd.	#5 O.C.; #6 O.C. Kay 11-18; Tok 1-22; Sib 1-16 claims	Mapping (1:5,000, Donnelly, 1976 B.Sc. Thesis, UBC); line cutting; rock sampling; EM and magnetic geophysical surveys. Core drilling (373.38 m).
1977-1978			No activity
1979	May-Ralph Resources Ltd.	22 Zone	Hand-cobbed bulk sample
1980-1982	Ryan Exploration Ltd. (U.S. Borax)	22 Zone; #6 Zone; Mackay Adit	Mapping; rock, stream sediment and soil sampling. Core drilling (452.32 m)
1983-1984			No activity
1985	Kerrisdale Resources Ltd.	#5 Zone; 21 Zone; 22 Zone	Mapping: rock and soil sampling. Core drilling (622.10 m)
1986	Consolidated Stikine Silver Ltd. (Consolidated Stikine)		No activity

Year	Owner	Work Area	Description
1987	Consolidated Stikine	#3 Bluff; 5 Zone; 21 Zone and 23 Zone	Stream sediment and soil sampling; core (all Kerrisdale) sampling; trench sampling
1988	Calpine Resources Inc. (Calpine)/ Consolidated Stikine	21A Zone; 21B Zone	Mapping; rock and soil sampling; core drilling (2,875.5 m). Discovery hole CA88-06 for the 21A Zone
1989	Calpine/Consolidated Stikine	21A Zone; 21B Zone; 22 Zone	Premier Resources acquired a controlling interest in Calpine in 1989 and took over managing the Eskay Creek project. Prime Resources merged with Calpine in April 1990. Homestake Canada Inc. (Homestake) acquired an equity position in Consolidated Stikine. Mapping; rock and soil sampling; airborne magnetic, EM, and very low frequency (VLF) geophysical surveys; ground magnetic VLF-EM, induced polarization (IP) geophysical surveys. Core drilling (44,338.9 m). Legal surveys
1990	Calpine/Consolidated Stikine	21B Zone; 21C Zone; PMP; Mack; proposed mill and mine site; GNC; Adrian	Mapping; rock and soil sampling; University of Toronto electromagnetic system (UTEM) geophysical survey. Core drilling (141,412.86 m). Environmental and terrane studies. Geotechnical and metallurgical studies Underground development in the 21B Zone. Bulk sample
1991	International Corona Corp. (Corona)	21B Zone; GNC	Mapping; rock and soil sampling; UTEM, seismic refraction and borehole frequency domain electromagnetics (FEM) geophysical survey. Core drilling (2,791 m) and core relogging core program. Start of underground core drilling.
1992	Corona	21B Zone; GNC	Mapping; rock and soil sampling; seismic refraction, gradient IP, transient EM and borehole FEM geophysical surveys. Core drilling (3,342 m). Homestake acquired Corona.
1993	Homestake	21B Zone; GNC	Mapping; rock sampling; resistivity, borehole FEM geophysical surveys. Core drilling (1,606.6 m). Feasibility study. Completion of Eskay mine road. T. Roth - MSc. thesis completed. R. Bartsch - MSc. thesis completed
1994	Homestake	21B Zone; Adrian; Albino Lake	Mapping; rock sampling; borehole FEM geophysical surveys. Core drilling (4,080.95 m)
1995	Homestake	21B Zone; NEX Bonsai	Mapping, rock sampling. Core drilling (3,468.1 m) Start of production on 21B Zone.

Year	Owner	Work Area	Description
1996	Homestake.	21B Zone; NEX; HW; Adrian; Bonsai	Mapping, rock sampling; trenching. Core drilling (21,280.80 m). Orthophoto Survey.
1997	Homestake	21B Zone; 21C Zone; 21E Zone; Adrian; GNC; Mack; Star	Prospecting; silt sampling. Core drilling (16,220.47 m)
1998	Homestake	21C; 21A; PMP; 5; 23; 22; 28; Mackay Adit; GNC; Mack; SIB Gaps; Star; Coulter	Mapping and prospecting; test gravity geophysical survey; Core drilling (21,909.63 m). Orthophoto survey.
1999	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting; structural study; geophysical compilation. Core drilling (17,363.96 m)
2000	Homestake.	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting. Core Drilling (25,893.93 m)
2001	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb; Felsite Bluffs; Sib Gaps; Pillow Basalt Ridge	Mapping and prospecting. Core drilling (22,035.48 m)
2002	Barrick Gold Corp. (Barrick)	21C; 21A; PMP; Deep Adrian; West Limb; 22 Zone; Mackay Adit	Mapping and prospecting. Core drilling (15,115.69 m). T. Roth PhD. thesis completed. Barrick acquired Homestake.
2003	Barrick	21C; 21A; PMP; Deep Adrian; West Limb; 22 Zone; Mackay Adit	Mapping and prospecting. IP and gravity geophysical surveys; line cutting. Core drilling (18,323.28 m)
2004	Barrick	22 Zone; Deep Adrian; West Limb; Ridge Block; Footwall	Mapping and prospecting; rock, soil, silt and vegetation sampling; topographic survey Borehole transient electromagnetics (TEM) geophysical survey. Core drilling (18,404.88 m)
2005	Barrick		Core drilling (16,000 m)
2006	Barrick		no activity
2007	Barrick		no activity
2008	Barrick		Mine closed in April. Reclamation commences.
2009-2016	Barrick		Mine reclaimed. Continuous care and maintenance

Year	Owner	Work Area	Description
2017	Barrick/Skeena		Skeena secures option
2018	Skeena	21A; 21C; 21B; 22 Zone	Skeena files Notice of Work, commences Phase 1 surface core drilling consisting of 45 holes (7,737.45m); Light detection and ranging (LiDAR) and photographic surveys. Initial Mineral Resource estimate.
2019	Skeena	21A; 21B; HW; 21E Zone; Tom MacKay; Tip Top; Eskay Porphyry	Updated Mineral Resource estimate. Prospecting; mapping; rock sampling. Surface core drilling consisting of 203 completed surface holes (14,091.87 m). Metallurgical leaching testwork, 2019 preliminary economic assessment (PEA) study.
2020	Skeena	22, 21A, 21C, 21B, 21E, HW, PMP, WTZ, LP, Tom MacKay	Surface core drilling: Phase 1 -197 holes for 36,582.45 m Phase 2 - 276 holes for 43,340.23 m Resistivity and IP geophysics surveys over Eskay Creek Project. Amended terms of the original option agreement; Skeena obtains 100% interest
2021	Skeena	21B; HW; PMP	Completion of Phase 2 surface core drilling (29 holes for 2,988.00 m)

6.2 Production

Underground mining operations were conducted from 1994 to 2008. From 1994 to 1997, ore was direct shipped after blending and primary crushing. From 1997 to closure in 2008, ore was milled onsite to produce a shipping concentrate.

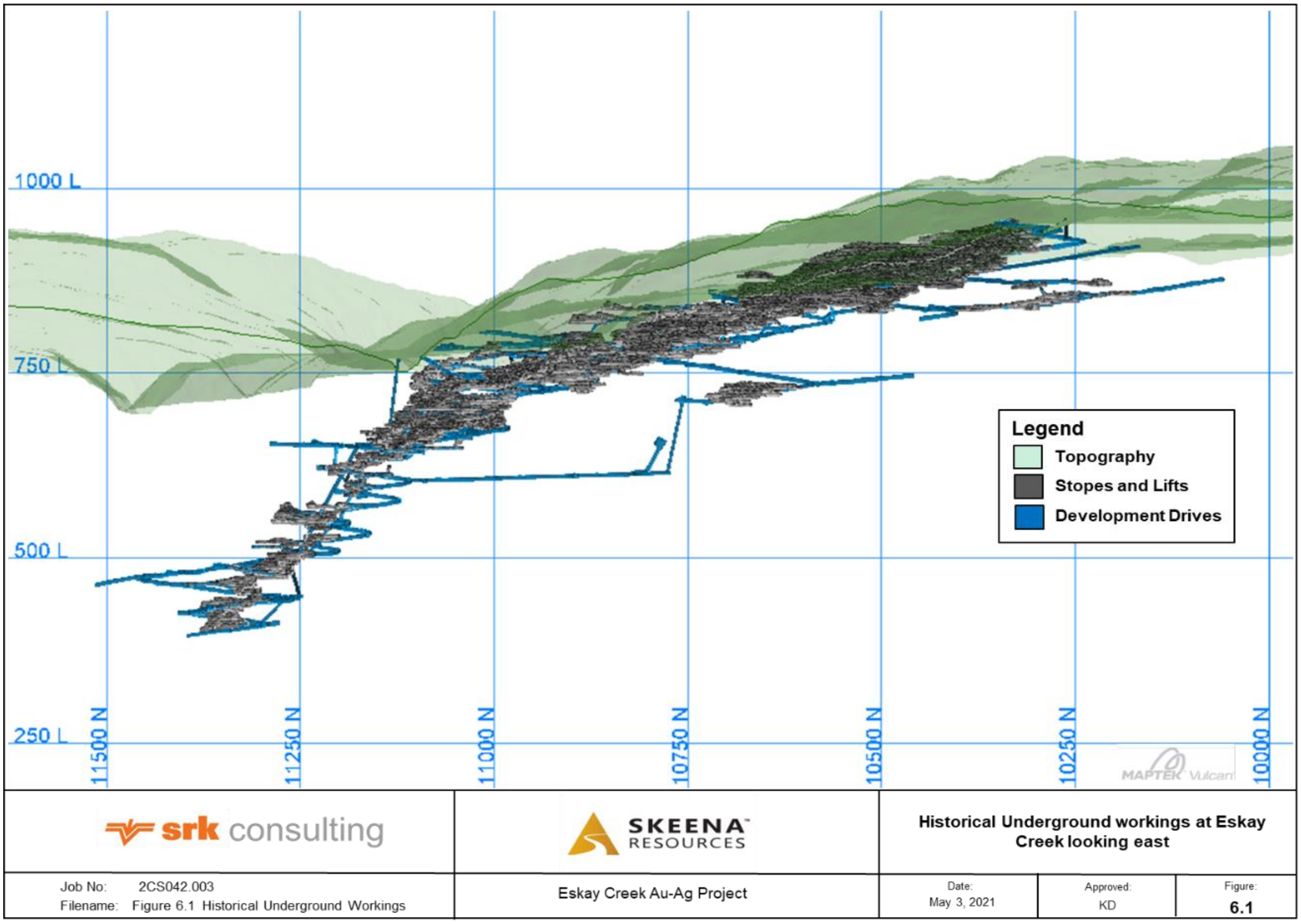
The Eskay Creek process plant began commercial production on 1 January 1998 at a 150 t/d rate. Production rates were incrementally increased from 1999 to 2004.

The Eskay Creek mine production is summarized in Table 6-2. Underground workings (stopes, lifts and development drives) are shown in Figure 6-1.

Table 6-2: Production History

Year Gold	Gold Produced (oz)	Gold Produced (kg)	Silver Produced (kg)	Silver Produced (oz)	Ore Tonnes Milled (t)	Ore Tonnes Shipped (t direct)
1995	196,550	6,113	309,480	9,950,401	0	100,470
1996	211,276	6,570	375,000	12,057,000	0	102,395
1997	244,722	7,612	367,000	11,799,784	0	110,191
1998	282,088	8,774	364,638	11,723,841	55,690	91,660
1999	308,985	9,934	422,627	13,588,303	71,867	102,853
2000	333,167	10,363	458,408	14,738,734	87,527	105,150
2001	320,784	9,977	480,685	15,454,984	98,080	109,949
2002	358,718	11,157	552,487	17,763,562	116,013	116,581
2003	352,069	10,951	527,775	16,969,022	115,032	134,850
2004	283,738	8,825	504,602	16,223,964	110,000	135,000
2005	190,221	5,917	323,350	10,396,349	103,492	78,377
2006	106,880	3,324	216,235	6,952,388	123,649	18,128
2007	68,000	2,115	108,978	3,503,861	138,772	0
2008	15,430	480	27,800	893,826	31,750	0
TOTAL	*3,272,628*	*102,112*	*5,039,065*	*162,016,018*	*1,051,892*	*1,205,604*

Figure 6-1: Historical Underground Workings, (Looking East)

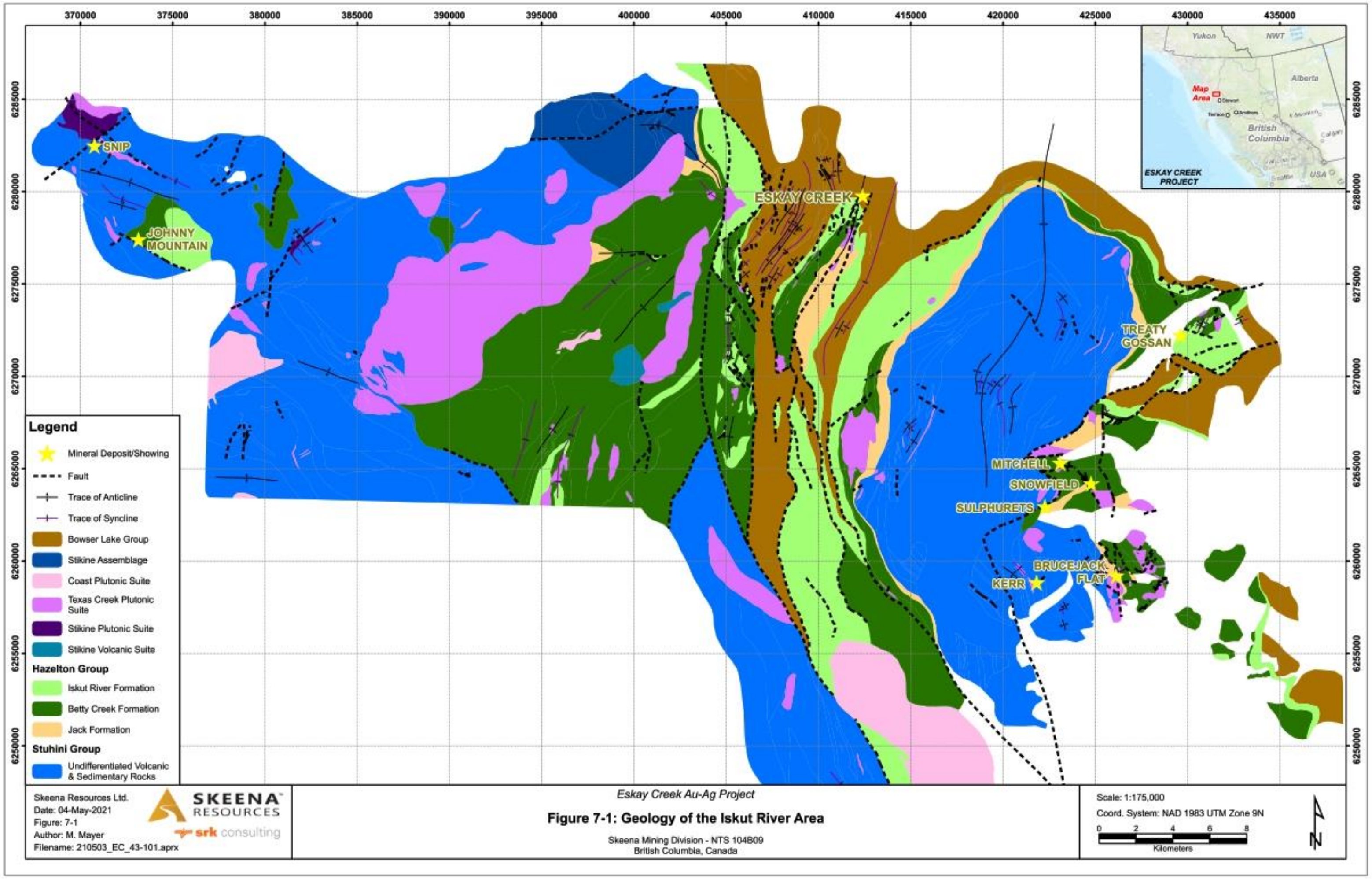


7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Iskut River region is located along the western margin of the Stikine Terrane, within the Intermontane Tectonic Belt of the Northern Cordillera (Figure 7-1). Anderson (1989) divides this area of the Stikine Terrane into four unconformity-bounded tectonostratigraphic elements. Deformed and metamorphosed sedimentary and volcanic rocks of the Paleozoic Stikine Assemblage are overlain by volcano-sedimentary arc complexes of the Stikinia Assemblage (Triassic Stuhini Group and Lower to Middle Jurassic Hazelton Group). These units are subsequently overlain by Upper Jurassic to Lower Cretaceous siliciclastic sedimentary rocks of the Bowser Lake Group that formed an overlap assemblage following the amalgamation of the Stikine and Cache Creek Terranes (Table 7-1). Six distinct plutonic suites have been recognized in the area and commonly intrude all assemblages (Table 7-2).

Figure 7-1: Regional Geology



Note: Eskay Creek deposit is held by Skeena. Other mines and deposits shown are owned by third parties.

Table 7-1: Regional stratigraphy of the Iskut River region (after Anderson, 1989 and Nelson et al., 2018)

Assemblage	Age	Rock Units
Coast Plutonic Complex	Tertiary	Post tectonic, felsic plutons
"Bowser Overlap" Assemblage (includes Bowser Lake Group)	Late Jurassic to Early Cretaceous	Deformed, siliciclastic sediments
"Stikinia" Assemblage (Includes Stuhini & Hazelton Groups)	Triassic to Middle Jurassic	Deformed volcanics, and intrusive rocks and basinal sediments
Stikine Assemblage	Early Devonian to Early Permian	Highly deformed limestone and volcanic rocks

Table 7-2: Iskut River region plutonic rock suite (after MDRU, 1992)

Suite Name	Lithologies	Age
Coast Plutonic Complex	Lamprophyres, gabbro-syenite	Tertiary (13-25 Ma)
Hyder	Monzogranite, monzonite, granodiorite	Tertiary (36-57 Ma)
Eskay Creek	Monzodiorite	Middle Jurassic (185 ± 2 Ma)
Sulphurets	Felsic intrusives/extrusives rocks	Middle Jurassic (185.9 Ma)
Texas Creek	Calc-alkaline granodiorite and quartz monzodiorite commonly cut by andesite dikes	Early Jurassic (189-195 Ma)
Stikine	Clinopyroxene-gabbro, diorite, monzodiorite and monzonite. Co-spatial with the Stuhini volcanic rocks	Late Triassic (210 Ma)

Lower greenschist facies metamorphism is common throughout the area and is likely related to the Cretaceous deformation that formed the Skeena fold and thrust belt (Rubin et al., 1990; Evenchick, 1991). Deformation in the Iskut River area is characterized by regional upright anticlinoria and synclinoria, related thrust faults, mesoscopic folds and normal faults, and cleavage development.

The regional-scale McTagg anticlinorium is the dominant structural feature, located in the eastern part of the Iskut River area.

Given the important relationship of the Hazelton Group to mineral deposits throughout the area, there have been many local mapping campaigns through the years, completed by different workers and at different scales. The resulting stratigraphic framework, although detailed in parts, contained numerous inconsistencies, and resulted in a poor ability to correlate stratigraphy and units on a regional scale. Working to resolve many of these issues, Nelson et al. (2018) completed a comprehensive regional investigation of the Hazelton Group, resulting in a new stratigraphic framework that contains six formations, detailed in Table 7-3.

Table 7-3: Stratigraphic framework for the Hazelton Group in the Eskay Creek Harrymel Creek area (after Nelson et al., 2018)

Formation	Lithologies	Sub-units	Age
Quock Formation. (Hazelton Group)	The highest unit in the Hazelton Group, consisting of 50-100 m of thinly bedded, dark grey siliceous argillite with pale felsic tuff laminae, and radiolarian chert. Commonly identifiable by presence of alternating dark and light-colored beds. Located in areas proximal to, but outside of the Eskay rift.		~164-170 Ma
Mt. Dilworth Formation. (Hazelton Group)	Dacite and rhyolite that form laterally continuous exposures; distinguished from felsic units of the Iskut River Fm. by its regional extent and lack of interfingering with mafic units. Located in areas proximal to, but outside of the Eskay rift.		174 Ma
Iskut River Formation. (Hazelton Group)	A several kilometer thick successions of interlayered basalt, rhyolite, and sedimentary rocks that occupy a narrow, fault-bounded north-trending belt known as the Eskay Rift. It consists of a highly variable succession of mafic and felsic volcanic and sedimentary units in differing stratigraphic sequences, often with multiple stratigraphic repetitions.	Willow Ridge mafic unit - Voluminous basalts located at varying stratigraphic levels; present in the hanging wall to the Eskay Creek deposit.	170-173 Ma
		Mount Madge sedimentary unit - Thinly bedded black argillaceous mudstone and felsic tuff (host to the stratiform mineralization at Eskay Creek in the Contact Mudstone); similar thin, discontinuous lenses enclosed within volcanics occur elsewhere in the Iskut River Formation	171-175 Ma
		Eskay Rhyolite Member - A linear flow dome complex of coherent to brecciated flows that show peperitic contacts with the overlying argillites; distinct geochemical signature compared to other felsic bodies in the area (Al/Ti>100). Associated with the mineralizing event at Eskay Creek.	175 Ma
		Bruce Glacier felsic unit - Non-welded to welded lapilli tuff, felsic volcanic breccia and coherent flows, and volcanic conglomerates. Located in the footwall of the Eskay Creek deposit.	173-179 Ma
Spatsizi Formation. (Hazelton Group)	Volcanic sandstone, conglomerate, and local bioclastic sandy limestone, mudstone-siltstone rhythmites, and limestone.		~174-187 Ma
Betty Creek Formation (Hazelton Group)	Can be subdivided into three informal units which have been observed as multiple bodies at different stratigraphic levels.	Brucejack Lake felsic unit - Flow dome complex believed to represent the extrusive and high-level intrusive products of a local magmatic centre; consists of k-spar, plagioclase, and hornblende phyric flows, breccias and bedded welded to non-welded felsic tuffs that are intruded by flow-banded coherent plagioclase phyric bodies (grade upward into flows).	183-188 Ma
		Johnny Mountain dacite unit - Generally located upsection of the Unuk River andesite consisting of bedded dacite lapilli tuff and breccia.	~194 Ma
		Unuk River andesite unit - Pyroclastic and epiclastic deposits often located unconformably overtop of the Jack Formation.	187-197 Ma
Jack Formation. (Hazelton Group)	Basal siliciclastic unit characterized by cobble-boulder granitoid-clast conglomerates, quartz-bearing arkosic sandstone, greywackes, and thinly bedded siltstones and mudstones, units sometimes weather to an orange colour. Some sections contain interbedded andesitic volcaniclastics.		196-203 Ma

7.2 Project Geology

7.2.1 Stratigraphy

The Eskay Creek deposit is located near the northern margin of the Eskay Anticline, just below the stratigraphic transition from volcanic rocks of the uppermost Hazelton Group to marine sediments of the Bowser Lake Group (Table 7-4 and Figure 7-2).

Descriptions of units from the local mine stratigraphy have been compiled in Table 7-4 from Roth et al. (1999) with regional stratigraphic nomenclature taken from Nelson et al. (2018). A stratigraphic section through the Project area is included in Figure 7-3.

Table 7-4: Stratigraphic Units

Regional Stratigraphy	Local Mine Stratigraphy	Description
Recent	Recent	In-situ soils and transported tills
Bowser Lake Group	Bowser Group Sediments	Mudstones and conglomerates
Willow Ridge mafic unit	Hanging Wall Andesite & Hanging Wall Sediments	Has both extrusive and intrusive phases, ranges from aphanitic to medium-grained with local feldspar phenocrysts, and in places exceeds 150 m thickness. Near the top of the sequence, well-preserved pillow flows and breccias, hyaloclastite, and basaltic debris flows containing minor mudstone and rhyolite clasts interspersed with thin argillite beds occur.
Mount Madge sedimentary unit	Contact Mudstone	Basal contact consists of a black-matrix breccia, comprising matrix-supported white rhyolite fragments set in a siliceous black matrix. Overlying the rhyolite and black matrix breccia are black mudstone and intercalated graded volcaniclastic sedimentary rocks. Within these volcaniclastic intervals, the presence of coarser rhyolite breccia fragments is interpreted to represent debris flows. The Contact Mudstone is the host unit for stratiform mineralization in the 21A, B, C, E, NEX and Hanging Wall (HW) Zones. It is characterized by laterally extensive, well-laminated, carbonaceous mudstone that is variably calcareous and siliceous and ranges from less than 1 m to more than 60 m in thickness.
Eskay Rhyolite member	Rhyolite	Up to 200 m thick. Linear set of flow-dome complexes, with locally preserved flow bands, flow lobes, breccias, hyaloclastite, spherulites, and perlitic textures. Located in the immediate footwall to the economically significant stratiform mineralized bodies, and also hosts stringer-style discordant mineralization.
Datum Mudstone	Lower Mudstone	Thin (5–15 m thick) black mudstone horizon
Datum Dacite	Dacite	Amygdaloidal, aphanitic dacite flow or sill
Bruce Glacier felsic unit	Dacite	Characterized by pumice-rich block and lapilli tuffs and heterogeneous epiclastic rocks that are locally fossiliferous
Spatsizi Formation	Even Lower Mudstone	Marine shales and interbedded coarse clastic sedimentary, volcaniclastic, and calcareous rocks
Betty Creek Formation	Footwall Andesite	Exposed in the core of the Eskay Anticline. Characterized by a thick sequence of coarse, monolithic andesite breccias and heterolithic volcaniclastic rocks.

Figure 7-2: Property-Scale Geology of the Eskay Creek Project Area

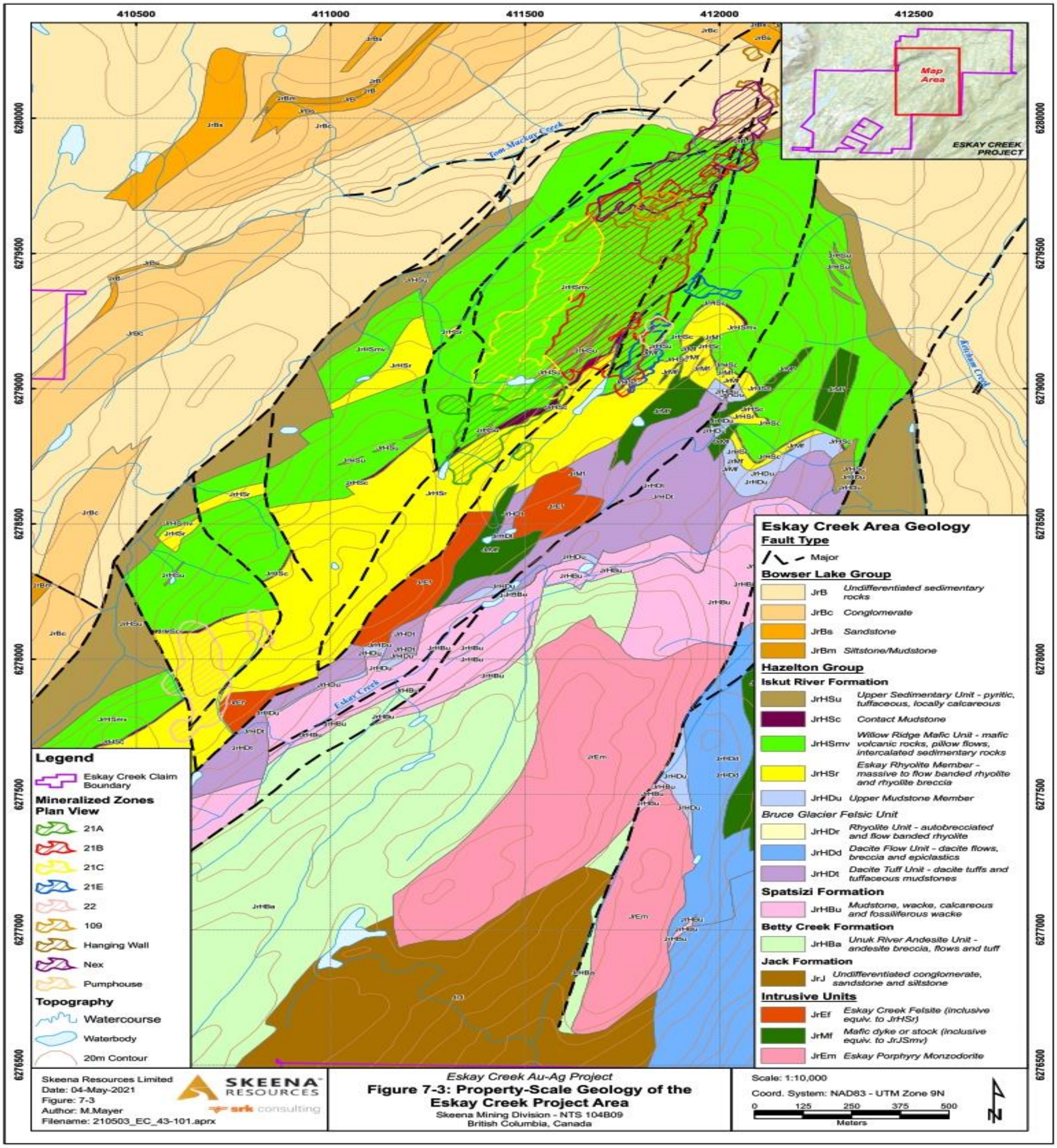
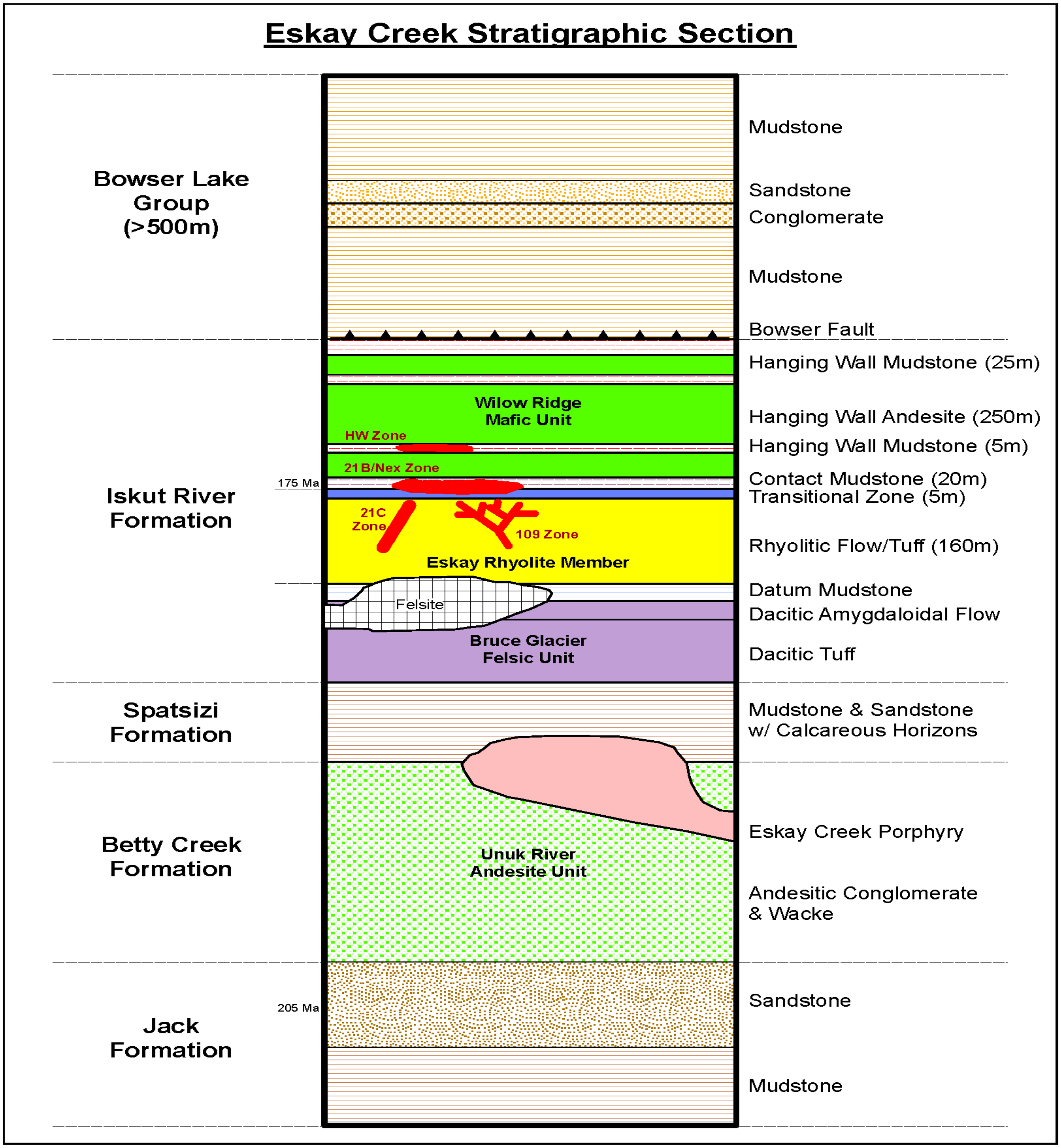


Figure 7-3: Eskay Creek Stratigraphic Section (modified after Gale et al., 2004)



7.2.2 Intrusive Rocks

Intrusive units are common through the stratigraphic sequence. The 184 ± 5 Ma (MacDonald et al., 1992; Childe, 1996) Eskay monzodiorite porphyry is perhaps the most voluminous intrusive on the property and is exposed in the core of the Eskay Anticline just south of the 21 Zone deposits. It predates the Eskay Rhyolite and mineralization located in the 21 Zone deposits, by 6-16 million years.

On the West Limb of the Eskay Anticline, a series of north-northeast trending felsic intrusive rocks form a series of prominent gossanous bluffs which extend for 7 km to the southwest of the Eskay Creek deposit. These felsic intrusive rocks are chemically indistinguishable from the Eskay Rhyolite (Bartsch, 1993, Roth, 1995) and display strong quartz, pyrite, and potassium feldspar alteration with minor sericite. Bartsch (1993) and Edmunds et al. (1994) believe these intrusive units represent sub-volcanic portions, or feeders, to the Eskay Rhyolite.

Basaltic dikes and sills linked to the Hanging Wall Andesite (Willow Ridge mafic unit) are also observed throughout the Eskay Creek stratigraphic section. Where they cut the Contact Mudstone (Mount Madge sedimentary unit), their contacts are frequently brecciated and peperitic, suggesting the mudstone was still wet at the time of intrusion (Roth et al., 1999).

7.2.3 Structure

Two structural events are recognized:

- D1: A Mid-cretaceous north-northwest compression event that formed northeast trending, syncline-anticline couples, and a spaced pressure solution cleavage (Edmunds and Kuran, 1992). The cleavage is axial planar to the bedding-defined Eskay Creek Anticline and is pervasive within the phyllosilicate-rich lithologies and even through the massive sulphide horizons. Faulting late in the D1 event resulted in the development of east-dipping thrust sheets, such as the Coulter Creek Fault, south of Eskay Creek. Regional metamorphism during the D1 event also resulted in the formation of porphyroblastic prehnite and calcite.

- D2: North-northeast directed compression event, locally re-oriented the D1 cleavage planes, and formed prominent north and northeast trending, steeply-dipping faults. Crosscutting relationships suggest that the north set of faults are early with apparently consistent sinistral displacement (Edmunds and Kuran, 1992). The later northeast trending set of faults commonly display oblique normal displacement. These faults form strong topographic lineaments and displace both stratigraphic contacts and mineralized zones.

7.2.4 Alteration

Alteration in the footwall volcanic units is characterized by a combination of pervasive quartz-sericite-pyrite, potassium feldspar, chlorite, and silica. Zones of most intense alteration are associated locally with sulphide veins that contain pyrite, sphalerite, galena, and chalcopyrite (Roth et al., 1999).

Alteration zonation is perhaps most apparent in the Rhyolite (Roth et al., 1999), closely associated with the 21 Zone deposits. Rhyolite located lateral to and at deeper levels beneath the area of stratiform mineralization is commonly moderately silicified and potassium feldspar altered. Silica alteration occurs as extremely fine-grained quartz flooding and densely developed quartz-filled micro veinlets. Potassium feldspar occurs as fine-grained replacement of plagioclase phenocrysts (Gale et al., 2004). Fractures that cut potassium feldspar-silica altered rhyolite typically have sericitic alteration envelopes and contain very fine-grained pyrite. Where alteration is most intense, chlorite replaces sericite in these fracture envelopes.

An intense tabular-shaped blanket of chlorite-sericite alteration, up to 20 m thick, occurs in the Rhyolite, immediately below the contact with the main stratiform sulphide mineralization. In these areas, magnesium chlorite has completely replaced the rhyolite to form a dark green, waxy rock consisting of clinochlore (Roth et al., 1999). This blanket coincides spatially with an area of greater rhyolite thickness and where extensive brecciation has developed in the upper part of the rhyolite unit. This zone of increased brecciation likely created more pathways for hydrothermal fluids, and therefore greater surface area for fluid-rock interaction, resulting in development of the stronger alteration zone.

7.3 Deposit Descriptions

Several distinct styles of stratiform and discordant mineralization are present at the Eskay Creek Project, defined over an area approximately 1,400 m long and up to 500 m wide (The main body of mineralization, the 21B Zone, is a stratiform tabular body of gold-silver-rich mineralization roughly 900 m long, 60 to 200 m wide, and locally exceeding 20 m thick. Individual clastic sulphide beds range from 1-100 cm thick and become progressively thinner up sequence) (Figure 7-4). Ore is composed of beds of clastic sulphides and sulphosalts containing variable amounts of barite, rhyolite, and mudstone clasts. Imbricated, laminated mudstone rip-up clasts were observed locally at the base of the clastic sulphide-sulphosalt beds, indicating turbiditic emplacement of some beds. In the thickest part of the orebody, pebble and cobble-sized clasts occur in a northward trending channel overlying the Eskay Rhyolite. The beds grade laterally over short distances into thinner, finer-grained, clastic beds and laminations.

Gold and silver occur as electrum and amalgam while silver mainly occurs within sulphosalts. Precious metal grades generally decrease proportionally with decrease in total sulphides and sulphosalts. Clastic sulphoside beds contain fragments of coarse-grained sphalerite, tetrahedrite, lead-sulphosalts with lesser freibergeite, galena, pyrite, electrum, amalgam, and minor arsenopyrite. Stibnite occurs locally in late veins, as a replacement of clastic sulphides, and appears to be confined to the central, thickest part of the deposit., suggesting a locus for late hydrothermal activity. Cinnabar is rare and is found associated with the most abundant accumulations of stibnite. Barite occurs as isolated clasts, in the matrix of bedded sulphides and sulphosalts, and also as rare clastic or massive accumulations of limited extent. Barite is more common towards the north end of the deposit.

Early exploration efforts focused on discordant-style, precious metal mineralization hosted in sulphide veins within the rhyolite, felsic intrusions, and the footwall volcanic units. Following recognition of more significant stratiform mineralization, exploration expanded further to the north, defining the 21 Zone deposits. Distinct zones were defined by variations in location, mineralogy, texture, and precious metal grades (Edmunds et al, 1994).

The main characteristics and stratigraphic locations of the ore zones are well summarized by Roth et al. (1999), and updated by Skeena, shown in Table 7-5.

Table 7-5: Summary of Mineralized Zones

Zone	Associated Elements	Characteristics	Stratigraphic Position
21A	As-Sb-Hg-Au-Ag	Stratiform lenses of massive to semi-massive sulphides (realgar, stibnite, cinnabar, arsenopyrite).	At the base of the Contact Mudstone
		Disseminated stibnite, arsenopyrite, tetrahedrite, and veinlets of pyrite, sphalerite, galena, tetrahedrite ± chalcopyrite.	Hosted within the underlying rhyolite
21B	Au-Ag-Zn-Pb-Cu-Sb	Stratiform, bedded clastic sulphides and sulfosalts including, sphalerite, tetrahedrite-freibergite, Pb sulfosalts (including boulangerite, bournonite, jamesonite), stibnite, galena, pyrite, electrum, and amalgam.	At the base of the Contact Mudstone
21Be	Ag-Au-Zn-Pb-Cu	Fine-grained massive to locally clastic sulphides and sulphosalts. Massive pyrite flooding in rhyolite grading upwards into massive sulphides and sulphosalts.	Within a fault-bounded block, mainly at contact between mudstone and rhyolite
21C	Ba (Pb-Zn-Au-Ag)	Bedded massive to bladed barite associated with very fine-grained disseminated sulphides including pyrite, tetrahedrite, sphalerite and galena. Located sub-parallel to and down-dip of the 21B zone.	Within the Contact Mudstone
		Localized zones of cryptic, disseminated, precious metal-bearing mineralization.	Hosted within the underlying rhyolite
21E	Sb-Ag-Au	Fine-grained stratabound sulphide lenses dominated by stibnite, pyrite, sphalerite, galena, chalcopyrite and arsenopyrite and associated with silica and carbonate alteration. This zone has generally lower gold-silver grades relative to the 21 Zones. Disseminated stibnite, arsenopyrite, and veinlets of pyrite, sphalerite, galena, tetrahedrite and chalcopyrite	Hanging-wall sediments Hosted within the underlying rhyolite
NEX	Au-Ag-Zn-Pb-Cu	The NEX stratiform mineralization is similar to the 21Be, and locally the 21B zone. Contains fewer sulphosalts and has a local overprint of chalcopyrite stringers.	At the base of the Contact Mudstone
HW	Pb-Zn-Cu	Massive, fine-grained stratabound sulphide lens dominated by pyrite, sphalerite, galena, and chalcopyrite (mainly as stringers). This zone has generally lower gold-silver grades and higher base metals relative to the 21 Zones.	Hanging-wall sediments
PMP	Fe-Zn-Pb-Cu	Veins of pyrite, sphalerite, galena, and tetrahedrite. Commonly banded; locally with colloform textures. Local zones of very fine-grained mineralization in rhyolite. Underlies the 21Be zone.	Hosted within the underlying rhyolite
109	Au-Zn-Pb-Fe	Veins of quartz, sphalerite, galena, pyrite, and visible gold associated with silica flooding and fine-grained amorphous carbon. Underlies the north end of the 21B and HW Zones.	Hosted within the underlying rhyolite
22 Zone	Au-Ag	Silica altered rhyolite with quart veinlets and micro veinlets and precious metals associated with pyrite-arsenopyrite	Hosted within the underlying rhyolite
LP	Zn-Pb-Cu-Fe-Au-Ag	Semi-massive base metals with associated gold - silver and sericite alteration.	Hosted within the Lower Mudstone, Even Lower mudstone and dacitic conglomerates/tuffs
WTZ	Au-Ag	Feeder style, discordant mineralization in sericitized and silicified rhyolite breccias.	Hosted within rhyolite

7.3.1 Stratiform Mineralization Zones

Stratiform style mineralization is hosted in black carbonaceous mudstone and sericitic tuffaceous mudstone of the Contact Mudstone located between the Rhyolite and the Hanging Wall Andesite. The stratiform-hosted zones include the 21B Zone, the NEX Zone, the 21A Zone (characterized by arsenic-antimony-mercury sulphides), the barite-rich 21C Mudstone Zone, and the 21Be Zone. Stratigraphically above the 21B Zone and usually above the first basaltic sill, the mudstones also host a localized body of base metal-rich, relatively precious metals-poor, massive sulphides referred to as the Hanging Wall or HW Zone.

7.3.1.1 21A Zone

The 21A Zone can be subdivided into stratiform- and feeder-style mineralization types. Stratiform mineralization is characterized by a gold-silver-rich sulphide lens that sits on the flank of a small depression at the Rhyolite-Contact Mudstone contact, located 200 m south of the 21B Zone. Stratiform-style, mudstone-hosted mineralization averages 10 m in thickness and is bound to the east by the Pumphouse fault. The sulphide lens consists of semi-massive to massive stibnite-realgar ± cinnabar ± arsenopyrite and local angular mudstone fragments. Areas with more concentrated stibnite-realgar ± cinnabar appear to be focused above the interpreted vent locations with relatively limited extent. Visible gold is rare.

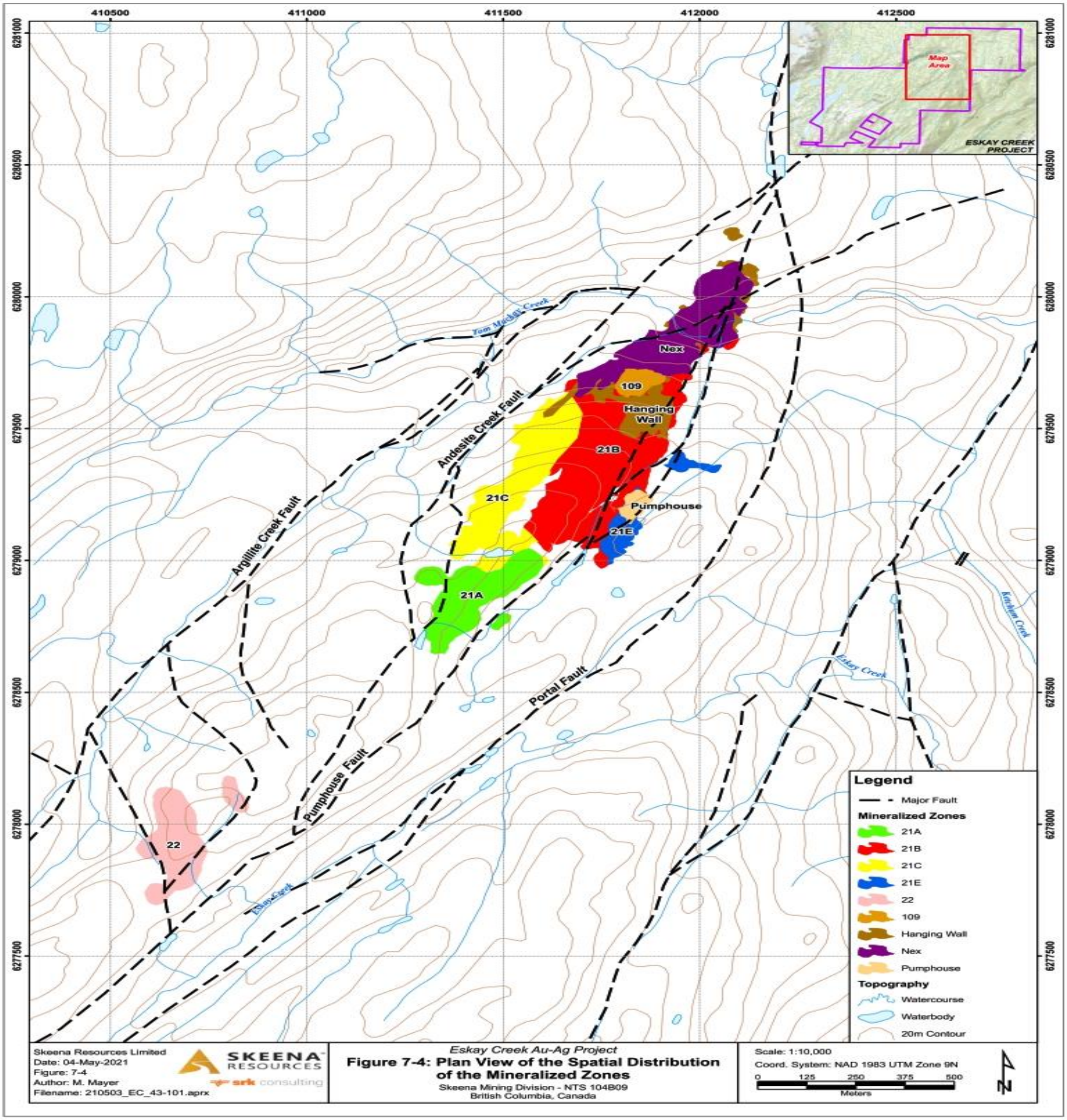
The mudstone is underlain by a discontinuous zone of intense Magnesium chlorite alteration and stockwork veining in the Eskay Rhyolite Member. Disseminated stibnite, arsenopyrite, and tetrahedrite also occur in the immediate footwall of the sulphide lens within the intensely sericitized rhyolite. Cinnabar and stibnite are observed in late fractures that cut the sulphide lens, the surrounding mudstone, and locally the rhyolite. Realgar-calcite veinlets locally cut the mudstone in a restricted area adjacent to the sulphide lens.

7.3.1.2 21B Zone

The main body of mineralization, the 21B Zone, is a stratiform tabular body of gold-silver-rich mineralization roughly 900 m long, 60 to 200 m wide, and locally exceeding 20 m thick. Individual clastic sulphide beds range from 1–100 cm thick and become progressively thinner up sequence). Ore is composed of beds of clastic sulphides and sulphosalts containing variable amounts of barite, rhyolite, and mudstone clasts. Imbricated, laminated mudstone rip-up clasts were observed locally at the base of the clastic sulphide-sulfosalt beds, indicating turbiditic emplacement of some beds. In the thickest part of the orebody, pebble to cobble-sized clasts occur in a northward trending channel overlying the Eskay Rhyolite. The beds grade laterally over short distances into thinner, finer-grained, clastic beds and laminations.

Gold and silver occur as electrum and amalgam while silver mainly occurs within sulphosalts. Precious metal grades generally decrease proportionally with the decrease in total sulphides and sulphosalts. Clastic sulphide beds contain fragments of coarse-grained sphalerite, tetrahedrite, lead-sulphosalts with lesser freibergite, galena, pyrite, electrum, amalgam, and minor arsenopyrite. Stibnite occurs locally in late veins, as a replacement of clastic sulphides, and appears to be confined to the central, thickest part of the deposit, suggesting a locus for late hydrothermal activity. Cinnabar is rare and is found associated with the most abundant accumulations of stibnite. Barite occurs as isolated clasts, in the matrix of bedded sulphides and sulphosalts, and also as rare clastic or massive accumulations of limited extent. Barite is more common towards the north end of the deposit.

Figure 7-4: Plan View of the Spatial Distribution of the Mineralization Zones (after Roth et al., 1999)



7.3.1.3 21C Zone

The 21C Zone is dominantly characterized by stratabound to stratiform barite-rich mineralization with associated disseminated base and precious metal-rich mineralization in the Rhyolite. It occurs at the same stratigraphic horizon as the 21B Zone but is located down-dip and subparallel to it. The two zones are separated by 40 to 50 m of barren Contact Mudstone, roughly 8 to 15 m thick. Mineralization is associated with mottled barite-calcite ± tetrahedrite beds in and near the base of the Contact Mudstone. Precious metal grades are variable. Local areas of brecciation are infilled with sulphides including sphalerite, pyrite, galena, and tetrahedrite. Mineralization in the underlying Rhyolite forms a cryptic, tabular body, sub-concordant to stratigraphy. Aside from containing 1-2% very fine-grained pyrite and trace sphalerite, tetrahedrite, and galena, the rhyolite appears similar to adjacent unmineralized areas. Drill holes have intersected intervals containing up to 35 g/t Au from these seemingly barren rhyolites.

7.3.1.4 21Be Zone

Precious-metal mineralization near the north end of the 21B Zone extends over top of the anticline into a block bound by segments of the north-south oriented Pumphouse faults. Mineralization of the 21Be Zone is found within a steeply dipping, fault-bounded slab of Contact Mudstone that is complexly folded and faulted.

While some of the mineralization within the 21Be Zone appears similar to the 21B Zone, the majority is steeply dipping and dominated by fine-grained, massive sulphosalts that grade downward into massive pyrite. There is a direct correlation of sulphosalts with higher-grade precious metal concentrations. The silver: gold ratio for the zone is approximately 100 times greater than in the 21B Zone. Stringers of chalcopyrite and chalcopyrite-galena-sphalerite overprint the mineralization. Fine-grained pyrargyrite occurs locally in hairline fractures cutting the mudstone and hosts ore-grade mineralization. Many of the textures observed in this zone suggest that the sulphides were introduced by replacement processes, perhaps along early faults.

7.3.1.5 HW Zone

The HW Zone forms massive sulphide horizons hosted in the mudstone interbeds within the Hanging Wall Andesite, at a higher stratigraphic level above the Contact Mudstone. Its geometry is disrupted by fault structures associated with the fold closure. Sulphides are typically fine-grained, finely banded, and consist of semi-massive to massive pyrite, sphalerite, galena, chalcopyrite, and tetrahedrite. Sphalerite is reddish-brown, suggesting a higher iron content compared to sphalerite encountered in other zones. The HW Zone has a higher base metal content compared to other zones, except where tetrahedrite ± sulphosalts are observed, which are associated with significantly higher precious metal grades.

7.3.1.6 North Extension Zone (NEX) Zone

The ~300 m long NEX Zone is geometrically complicated by numerous faults that cut the nose of the Eskay Anticline. Textures, mineralogy, and precious-metal grades are somewhat variable and show similar characteristics to parts of the 21Be Zone and distal parts of the 21B Zone, suggesting synchronous deposition. Pyrite and chalcopyrite are more common whereas antimony-mercury bearing minerals are less common. Chalcopyrite occurs in stringers that overprint earlier clastic mineralization and may be related to the formation of the HW Zone. Much of the contained pyrite may also have been introduced during this later event.

7.3.1.7 21E Zone

The 21E Zone sits on the eastern most block. Locally, mudstone interbeds within the Hanging Wall Andesite host fine-grained to massive and locally clastic sulphides and sulphosalts. Sulphides include fine laminae of tetrahedrite, replacement to dendritic style stibnite, and minor blebs or replacements of sphalerite-galena-chalcopyrite and arsenopyrite and associated silica and carbonate alteration. This zone generally has lower gold-silver grades relative to the other 21 Zones. In the underlying Rhyolite, the mineralization is associated with disseminated stibnite, arsenopyrite and veinlets of pyrite.

7.3.1.8 Lower Package Zone

The Lower Package (LP) stratabound-style mineralization is hosted stratigraphically below the Rhyolite and is hosted within the Lower Mudstone, Dacite, Even Lower Mudstone and Footwall Andesite. Mineralization is comprised of semi-massive base metal-rich beds with associated gold and silver. Metal content appears to be stronger near bounding faults of the Eskay Creek basin (in particular the Pumphouse fault), and related conjugate fault sets.

7.3.2 Discordant-Style Mineralization

Stockwork and discordant-style mineralization at Eskay Creek is hosted in the Rhyolite within the PMP, 109, 21A-Rhyolite, 21C-Rhyolite, 21E-Rhyolite, Water Tower, and 22 Zones. The PMP Zone is characterized by pyrite, sphalerite, galena, and chalcopyrite-rich veins and veinlets hosted in strongly sericitized and chloritized rhyolite. The 109 Zone comprises gold-rich quartz veins with sphalerite, galena, pyrite, and chalcopyrite associated with abundant carbonaceous material hosted mainly in siliceous rhyolite. The 21A and 21C-Rhyolite Zones consist of very fine-grained cryptic pyrite with rare sphalerite and galena in sericitized rhyolite. The 22 Zone consists of cross-cutting arsenopyrite, stibnite and tetrahedrite veins hosted in massive to pyroclastic facies rhyolite.

Descriptions of the following discordant mineralized zones are modified after Roth et al. (1999).

7.3.2.1 PMP Zone

The PMP Zone is a discordant zone of diffuse vein and disseminated sulphide mineralization hosted in the Rhyolite beneath the eastern part of the 21B Zone and just north of the 21Be Zone. Precious metal grades are generally lower than in other zones. Patchy sulphide mineralization is observed locally through the rhyolite in the form of veins containing pyrite, sphalerite, galena, and lesser sulphosalts such as tetrahedrite. Chalcopyrite content increases with depth. Sphalerite is generally darker (more iron-rich) than in the overlying 21B Zone. Mineralization is commonly banded and is locally characterized by colloform textures. Locally, areas of very fine-grained disseminated sulphide mineralization enriched in precious metals occur; these are similar to footwall hosted mineralization observed in the 21C Zone.

7.3.2.2 109 Zone

The 109 Zone is named after the discovery drill hole of the same name. The zone is characterized by a distinct siliceous stockwork of crustiform quartz veins with coarse-grained sphalerite, galena, minor pyrite, and chalcopyrite. The 109 Zone is hosted entirely within the Rhyolite, beneath the north end of the 21B and the HW Zones. Gold and silver occur in electrum and sulphosalts.

7.3.2.3 22 Zone

The 22 Zone is located 2 km southeast of the 21A Zone, with mineralization hosted exclusively in the silicified Rhyolite. It is believed to represent a feeder zone intimately related to conjugate faults occurring between the north-south trending basin bounding faults (Pumphouse and Andesite Creek). Gold and silver mineralization are hosted within barren-looking quartz micro-veinlets and disseminated fine-grained pyrite and blebby sphalerite. Fine grained arsenopyrite and stibnite are occasionally observed. Higher vein densities generally indicate better gold grades.

7.3.2.4 WT Zone

The WT Zone is located on the western side of the property and occurs as steeply dipping, feeder-style, discordant mineralization within intensely altered rhyolite breccias. Mineralization is hosted within quartz veinlets and disseminated fine-grained pyrite with blebby sphalerite.

7.4 Prospects/Exploration Targets

Exploration potential is discussed in Section 9.

7.5 QP Comments on "Item 7: Geological Setting and Mineralization"

In the opinion of the QP, the understanding of the Eskay Creek deposit setting, lithologies, and geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources.

8 DEPOSIT TYPES

The Eskay Creek deposit is known as an outstanding example of a high-grade, precious metal rich epithermal volcanogenic massive sulphide (VMS) deposit that formed in a shallow submarine setting. The deposit has features and characteristics typical of a classic VMS deposit: it formed on the seafloor in an active volcanic environment with a rhyolite footwall and basalt hanging wall, having chlorite-sericite alteration in the footwall and sulphide formation within a mudstone unit at the seafloor interface. What differentiates the Eskay Creek deposit from other VMS deposits are the high concentrations of gold and silver, and an associated suite of antimony, mercury, and arsenic. These mineralization features, along with the high incidence of clastic sulphides and sulphosalts, are more typical of an epithermal environment with low formation temperatures.

The processes responsible for the formation of the Eskay Creek deposit are not unique in the VMS environment, but require the coincidence of several favourable conditions to optimize the precious metal grade in the deposit. Roth et al., (1999) hypothesized that the maintenance of a low temperature environment was of primary importance for the active and continued transport of gold. Heat was continually removed at the vent site due to the collapse and dismemberment of chimneys and mounds; an outcome which would have prevented the hydrothermal system from sealing. The redeposition of clastic sulphides adjacent to the vent site would have prevented the system from increasing in temperature beyond the range permissible for gold deposition.

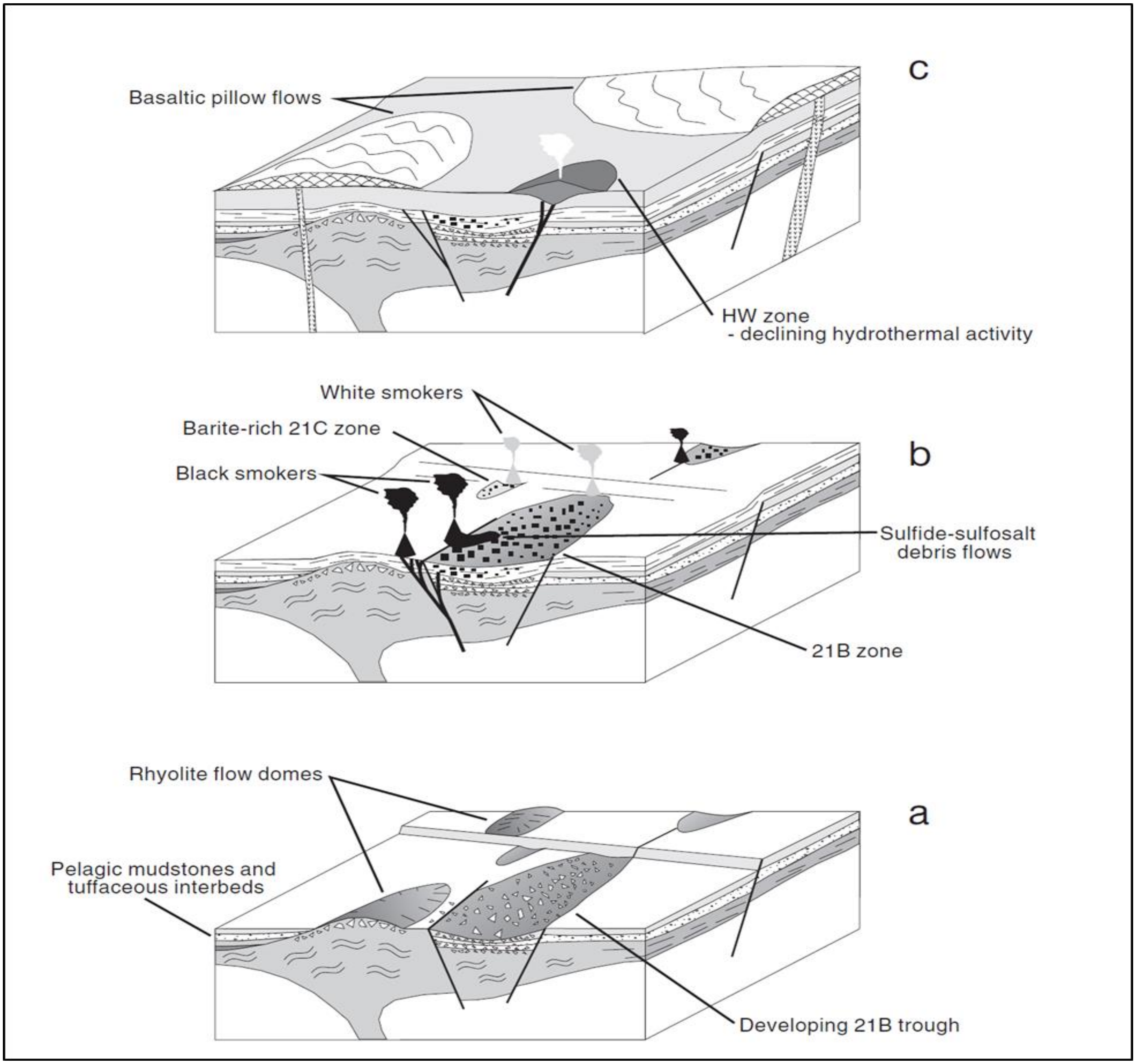
Roth et al., (1999) developed a deposit genesis model for the 21 Zones, that included the following phases:

- Rifting, basin development and intrusion and extrusion of rhyolite flow domes. Coarse volcaniclastic debris from extrusive portions of the rhyolite domes are deposited along the developing 21b Zone trough (part a of Figure 8-1);

- Hydrothermal activity is focused through rift faults forming chimneys and mounds on the seafloor. Collapse or disruption of these mounds forms clastic sulphide-sulfosalt debris which is redeposited in the 21B Zone trough. Other smaller basins provide the sites for similar mineralization and barite-rich zones (21c) related to white smokers (part b of Figure 8-1);

- The HW zone of massive sulphide forms higher in the mudstone stratigraphy and basaltic magmatism begins (dykes and flows) during the waning stages of hydrothermal activity (part C of Figure 8-1).

Figure 8-1: Genetic Model



Note: Figure from Roth et al., 1999.

8.1 QP Comments on "Item 8: Deposit Types"

The QP is of the opinion that exploration programs that use either a VMS or a hot-spring deposit model are applicable to the Project area.

9 EXPLORATION

9.1 2018

9.1.1 Grids and Surveys

McElhanney Consulting Services Ltd. (McElhanney) of Vancouver, B.C flew an airborne light detection and ranging (LiDAR) and photo acquisition survey in December 2018. The resulting topography map was compiled to 0.1 m accuracy.

LiDAR and photo acquisition were collected simultaneously with equipment co-mounted on the sampling aircraft. Sixty flight lines comprising 539-line kilometres were completed, covering the 100 km^2 survey area.

9.2 2019

9.2.1 Mapping and Grab Sampling Program

9.2.1.1 Tom MacKay

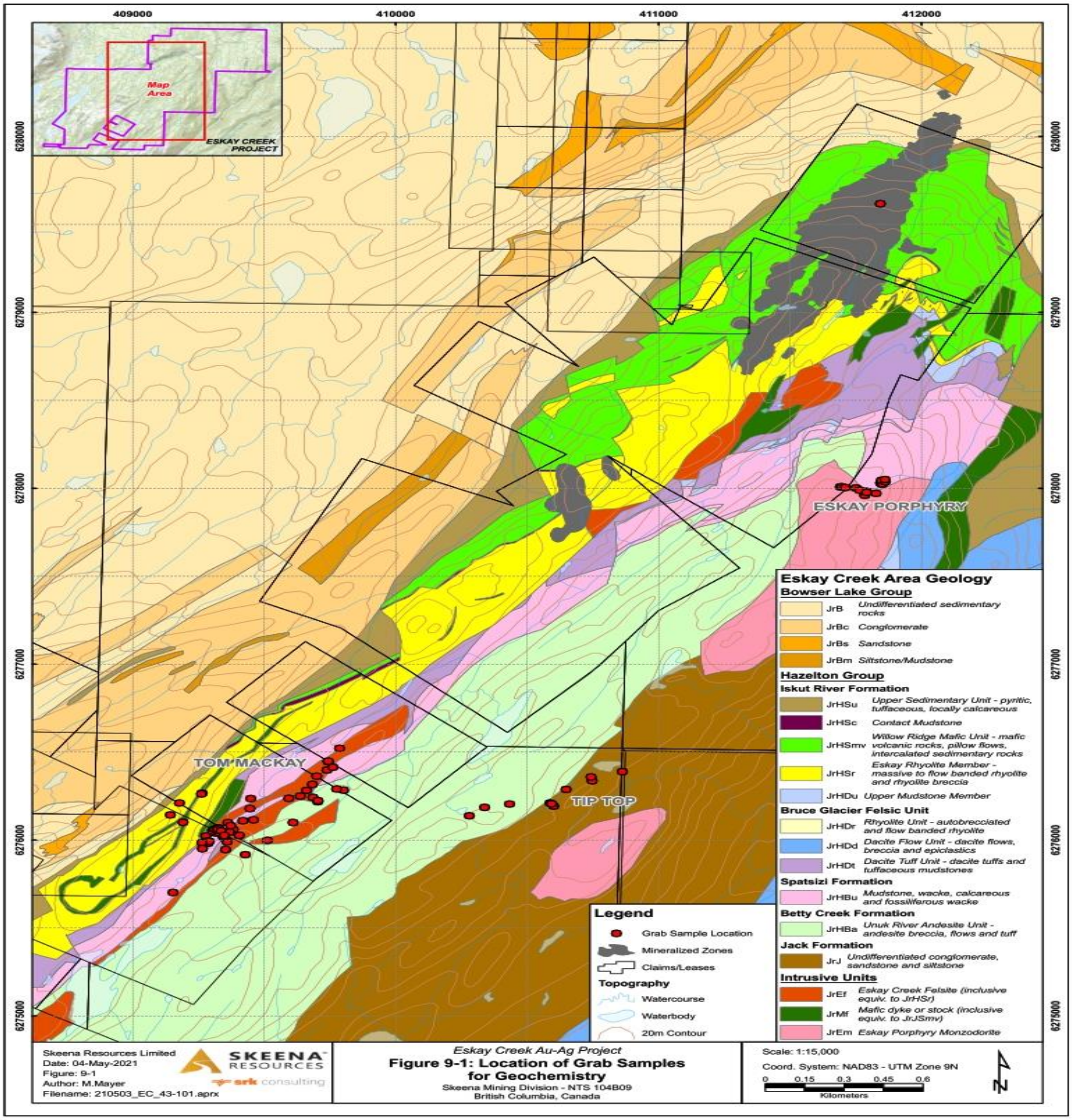
In mid-October 2019, geological mapping and grab samples were collected by Skeena geology staff in the Tom MacKay area, located approximately 2.2 km south of the 22 Zone. Historical drill holes in the adit area contained anomalous gold values primarily within felsite which generally lies subvertical, dipping towards the east. The purpose of the program was to determine the relationship of the felsite dykes to the Eskay Rhyolite and collect rocks for whole rock geochemistry analysis.

Mapping and sampling were conducted over a 0.45 km^2 area. A total of 51 grab samples were collected from outcrops for whole rock analysis and analysed by multi-acid multi-element inductively coupled plasma (ICP) and 44 structural measurements were taken (Figure 9-1).

The results of field mapping show the Eskay Rhyolite varied slightly from the mapped and historically logged felsite dyke. The structural data taken support an anti clinical environment with foliations and north-south faults dipping sub-vertically to the east. The strongest visual mineralization appeared to be associated with the brecciated felsite dyke within the structural corridor.

Sampling returned a number of anomalous gold and silver grades. Following the favourable geochemical assays, it was recommended to drill to the northeast of the Tom MacKay area.

Figure 9-1: Location of Grab Samples



9.2.1.2 Eskay Porphyry and Tip Top

In August 2019, geological mapping and grab sampling was carried out on the Tip Top and Eskay Porphyry targets, located 700 m east of the 21 Zone deposits (see Figure 9-1). The Eskay Porphyry is a monzodiorite exposed in the core of the Eskay anticline, intruding into the Footwall Andesite. The Tip Top prospect is located along the same structural trend towards the southwest.

Twenty-eight grab samples were collected from Tip Top and 14 grab samples were collected from the Eskay Porphyry, a number of which had anomalous gold and silver values.

9.3 2020

9.3.1 Geophysics

During, late summer 2020, Dias Geophysical Limited (Dias) carried out a 3D direct-current (DC) resistivity and induced polarization (DCIP) survey on the Eskay Creek Project using the DIAS32 system in the UTM Zone 9N WGS84.

The geophysical program was designed to detect the electrical resistivity and chargeability signatures associated with potential targets of interest. This was achieved using the DIAS32 acquisition system, in conjunction with the DIAS transmitter, to produce up to 7.0 kW of total power. The survey was completed using a rolling distributed partial 3D DCIP array with a pole-dipole transmitter configuration. The survey covered approximately 5 km^2.

Dias completed a partial 3D rolling distributed pole-array in common voltage reference (CVR) mode. The survey layout consisted of a total of five northeast-southwest oriented receiver lines, spaced at 200 m (Figure 9-2). Along the receiver lines, the electrode stations were spaced 100 m apart. The injection lines ran perpendicular to the five receiver lines and offset by 50 m from the receiver nodes.

After thorough quality control, all the accepted data were used to produce a set of unconstrained 3D DC and IP models using the SimPEG inversion code.

9.4 Exploration Potential

There is remaining exploration potential in the Eskay Creek deposit.

Skeena considers that well-defined, mineralized syn-volcanic feeder structures that propagate through the volcanic pile have not been sufficiently explored at depth and along strike. Examples of this well-documented mineralization style include the 22 Zone, Water Tower Zone, 21C Zone and mudstone in the HW Zone where these feeders propagate.

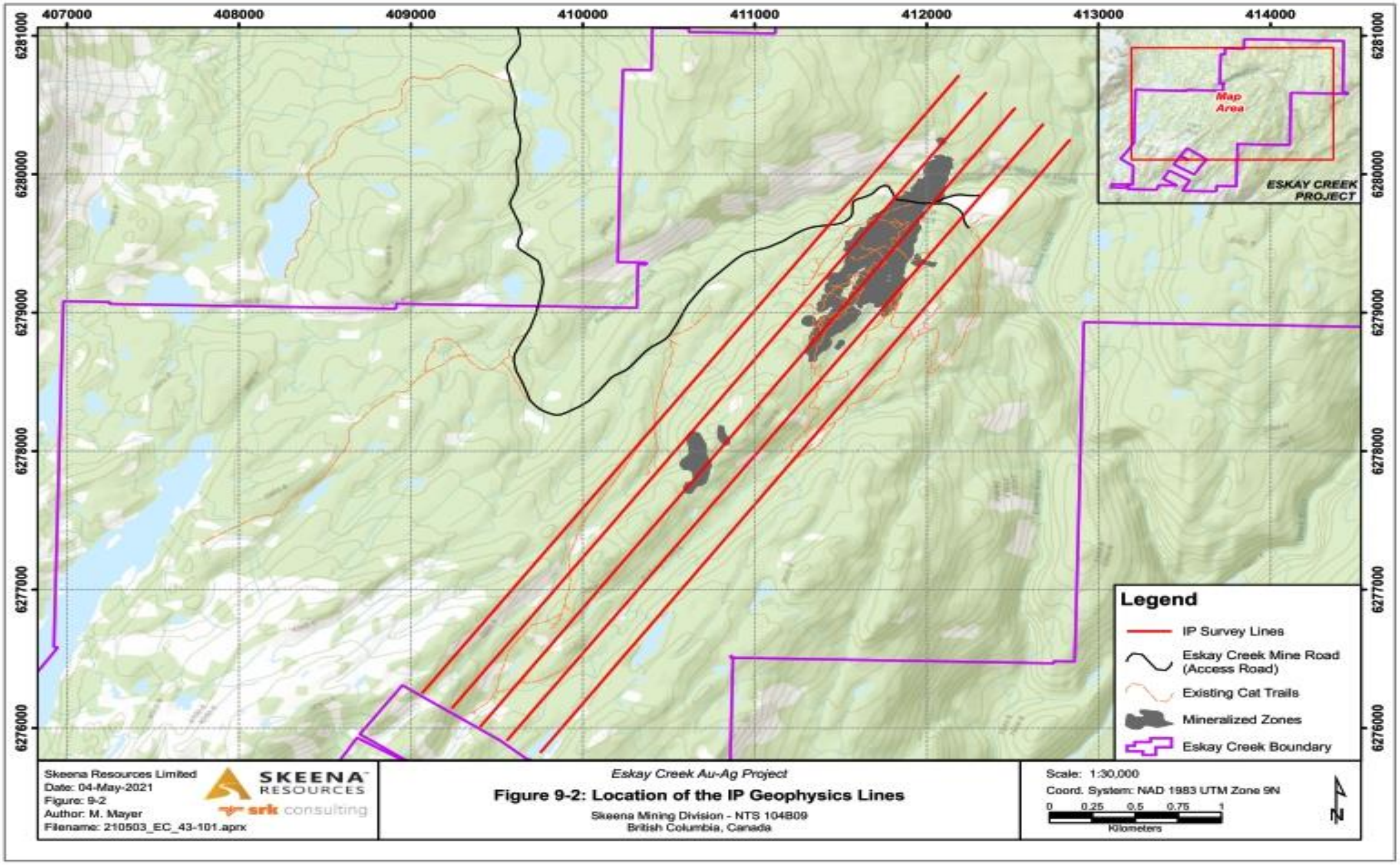
In addition, the underexplored Lower Mudstone is situated ~100 m stratigraphically below the more well-known Contact Mudstone. and represents a horizon with potential to host similar exhalative style mineralization. Exploratory target ranking will be influenced by areas where known synvolcanic feeder structures intersect this unit, as these locales will offer the highest potential for development of additional exhalative mineralization.

Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

9.5 QP Comments on “Item 9: Exploration”

The exploration programs completed to date are appropriate to the style of the deposit and prospects. Additional exploration has a likelihood of generating further exploration successes.

Figure 9-2: Location of the IP Geophysics Lines



10 DRILLING

10.1 Introduction

Surface drilling has been carried out by multiple operators, with the first drilling on the property by Unuk Gold in 1932. Data collected prior to Skeena's project interest is referred to as legacy data.

Legacy drilling consists of 1,522 surface core drill holes totalling 342,119 m, drilled from 1932 until 2004. Since 2018, Skeena has drilled 751 surface drill holes totalling 104,740 m. Table 10-1 summarizes the surface drilling on the Eskay Creek Project arranged by operator and year (modified after Gale et al., 2004). Figure 10-1 shows the location of the surface holes.

Table 10-1: Drill Summary Table

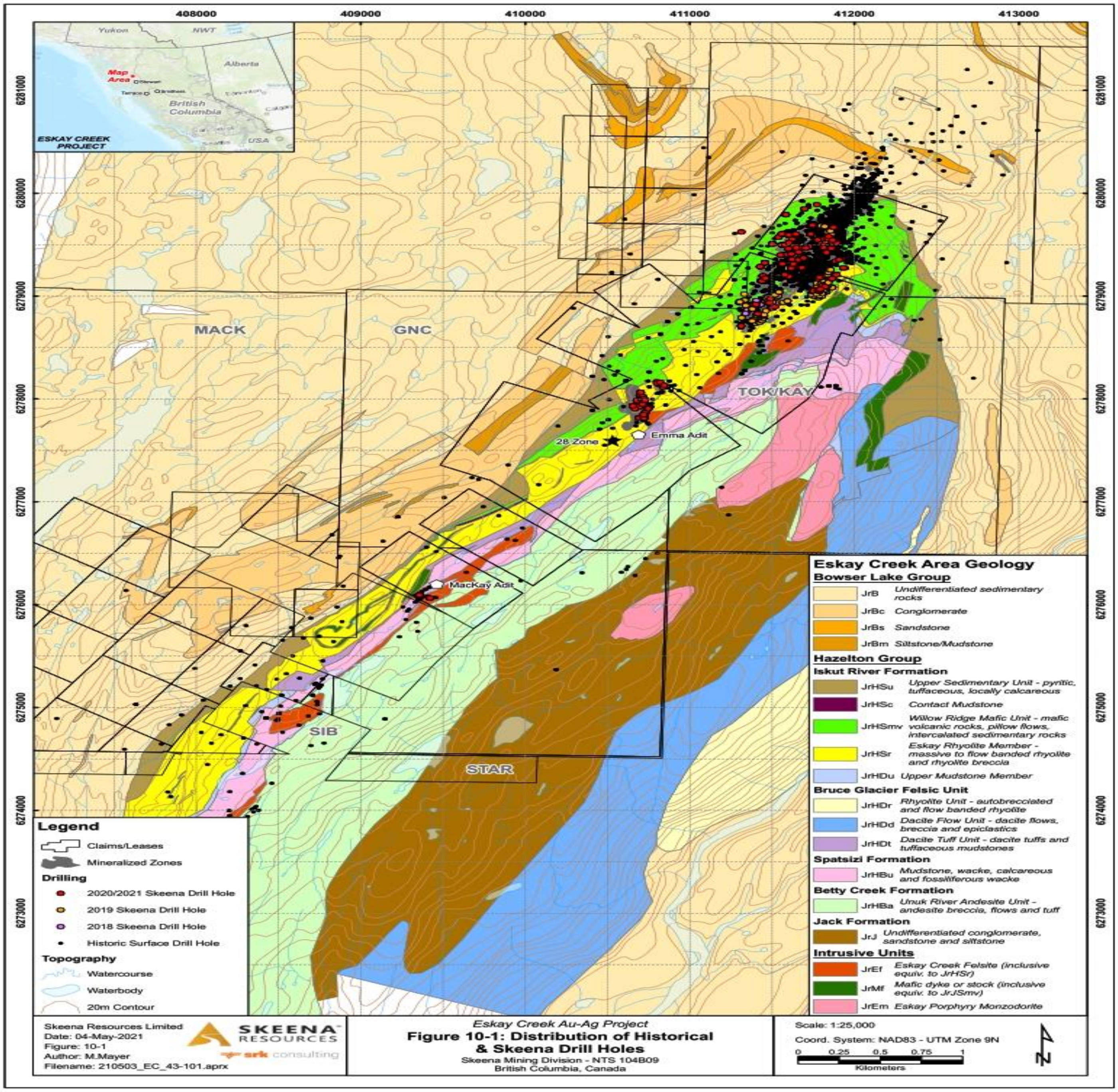
Period of Work	Company	Area of Work	Number of Holes	DDH #'s	Metres Drilled
1932–1934	Unuk Gold/Unuk Valley Gold		11	Unuk 1-11	261.21
1935–1938	Premier Gold Mining Co. Ltd.		38	P 12-49	1,825.95
1964	Stikine Silver Ltd. (Stikine Silver) / Canex Aerial Exploration Ltd.	Emma Adit	6	C-1 to C-6	224.64
1965	Stikine Silver	Emma Adit	3	?	15.85
1973	Kalco Valley Mines Ltd.	22 Zone	7	KV-1 to KV-7	299.62
1975–1976	Texasgulf Canada Ltd.	#5 O.C./#6 O.C.	7	K76-1 to K76-7	373.38
1980–1982	Ryan Exploration Ltd. (U.S. Borax)	22 Zone/6 Zone	7	MR-1 to MR-7	452.32
1985	Kerrisdale Resources Ltd.		5	KDL 85-1 to 85-5	622.1
1988	Calpine Resources Inc. (Calpine) / Consolidated Stikine Silver Ltd. (Consolidated Stikine)	21A/21B Zones	16	CA88-01 to CA88-16	2,875.50
1989	Calpine / Consolidated Stikine	21A/21B/22 Zone	179	CA 89-17 to CA 89-196	43,017.90

Period of Work	Company	Area of Work	Number of Holes	DDH #'s	Metres Drilled
				CA 89-198 to CA 89-205	
			7	CA 8922-01 to CA 8922-07	1,321.00
1990	Calpine / Consolidated Stikine	21B/21C	513	CA 90-197	115,272.26
		PMP		CA 90-206 to CA 90-691	
		Mack		MK 90-01 to MK 90-04	
		Proposed Mill Site		PMS 90-01 to PMS 90-06	
				KP-1 to KP-16	
			3	CA 90-692, 693, 696	1,036.60
		GNC	19	GNC 90-01 to GNC 90-19	3,318.00
		Adrian	35	AD 90-01 to AD 90-35	21,786.00
1991	International Corona Corp. (Corona)	21B Zone	12	C 91-700 to C 91-711	2,791.00
		GNC	5	GNC 91-20 to GNC 91-24	
1992	International Corona Corp.	21B	1	C 92-712	3,342.00
		GNC	7	GNC 92-25 to GNC 92-31	
1993	Homestake Canada Inc. (Homestake)	21B	2	C 93-713- to C 93-714	1,606.60
		GNC	3	GNC 93-32 to GNC 93-34	
1994	Homestake	Adrian	6	AD 94-35 to AD 94-40	3,531.70
		21B	5	KP 94-1 to KP 94-5	549.25
1995	Homestake	21B/NEX Zone	21	C 95-715 to C 95-735	3,468.10

Period of Work	Company	Area of Work	Number of Holes	DDH #'s	Metres Drilled
				(formerly labelled NEX 95-1 to 18 and QZ 95-1 to 3)	
		Bonsai	5	BZ 95-1 to BZ 95-5	
1996	Homestake Canada Inc.	21B/NEX/HW	94	C 96-736 to C 96-829	21,280.80
		Adrian	19	AD 96-41 to AD 96-59	
		Bonsai	1	BZ 96-06	
1997	Homestake	21B/21C/21E	42	C 97-830 to C 97-871	16,220.47
		Adrian	14	AD 97-60 to AD 97-73	
		GNC	1	GNC 97-30X	
		Mack/Star	2	MP 97-01 to MP 97-02	
1998	Homestake	Core Property	79	C 98-872 to C 98-950	21,909.63
		GNC	2	GNC 98-35 to GNC 98-36	
		Mack	8	MP 98-03 to MP 98-09	
		Star	1	SP 98-01	
1999	Homestake	Core Property	64	C 99-951 to C 99-1014	17,363.96
2000	Homestake	Core Property	77	C001012W	25,893.93
				C001015 to C001088	
2001	Homestake	22 Zone	61	C011089 to C011145	22,035.48
		21C Zone			
2002	Barrick Gold Corp. (Barrick)	21C Zone	47	C02-1146 to C02-1178	15,115.69
		21A Zone		C02-920X, C02-975X	

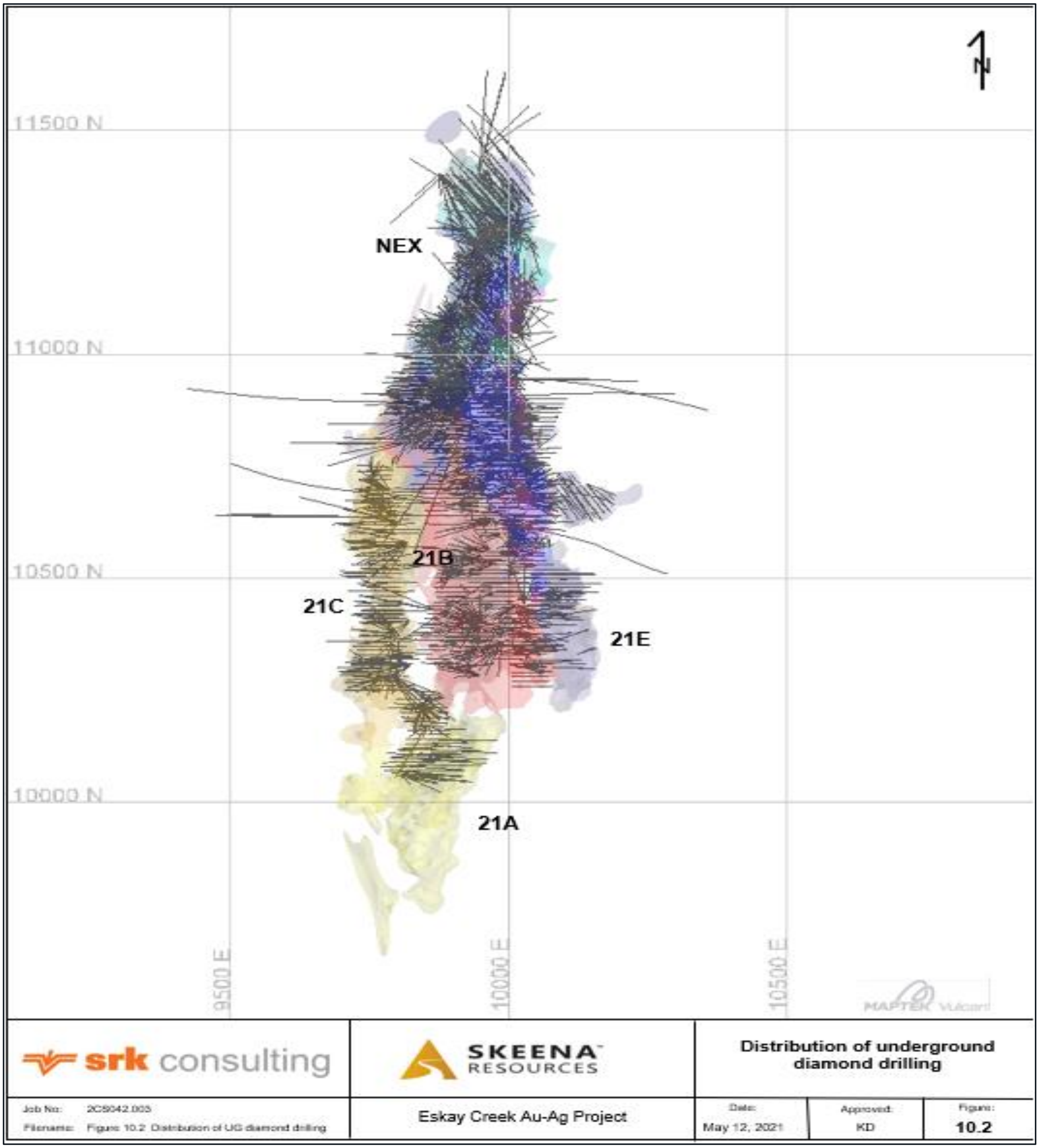
Period of Work	Company	Area of Work	Number of Holes	DDH #'s	Metres Drilled
		Deep Adrian			
2003	Barrick	22 Zone	71	C03-1179 to C03-1245	18,323.28
		21A Zone		C03-919X	
		21C Zone			
2004	Barrick	22 Zone	55	C04-1261 to C04-1298	18,404.88
		Ridge Block		C04-1020X, C04-1196X	
		21C/21E Zones		C04-1206X	
		Deep Adrian		5702, 6461, 6464	
2018	Skeena	21A / 21B / 21C / 22 Zones	46	SK-18-001 to SK-18-043; SK-18-048 to SK-18-051	7,737.45
2019	Skeena	21A / 21B / 21E / HW Zones	203	SK-19-044 to SK-19-047; ~SK-29-052 to SK-19-247	14,091.87
2020	Skeena	21A / 21B / 21C / 21E / HW / PMP / WT / MAC / 22 Zones	473	~ SK-20-248 to SK-20-788	79,922.79
2021	Skeena	21B / HW / PMP Zones	29	~ SK-21-789 to SK-21-816	2,988.00

Figure 10-1: Surface Drill Hole Location Plan



A total of 6,061 underground drill holes were drilled totalling 309, 213 m. All underground drilling is legacy. Figure 10-2 shows the locations of the underground core drill holes. Underground drill holes are generally less than 100 m in length and drilled with an average spacing of 10 m using BGM (~40 mm) core diameter. In highly complex areas where mining was active, drill spacing was locally reduced to 5 m.

Figure 10-2: Underground Drill Hole Location Plan



10.2 Logging Procedures

10.2.1 Legacy

Limited information is available for procedures used during the exploration programs carried out prior to 2004. The drill core was logged using DLG computer programs for data entry as well as for drill log printing. The data were entered directly into laptop computers and the rock units coded with four-digit geology codes. Mineralized sections were logged separately as nested units of primary units. Textural descriptions, rock colour and structure were also coded with two-character fields. Remarks were typed into separate fields to characterize unique geology structure or mineralization features.

All collar and survey information were tabulated in master files within the DLOG computer program. Completed logs were printed and the information was exported int ACAD and Vulcan software to facilitate plotting drill hole location maps and cross section.

The only data that remains from the legacy data is the collar, survey, the four-digit lithology code and assay data.

10.2.2 Skeena

All core logging and technical tasks were completed by geologists and supervised geological technicians employed by Skeena. Once the initial assessment was completed, core was measured, and 1 m intervals were marked directly on the core with China markers. The start and end meterage of each core box was marked on the upper left and lower right, respectively. A metal tag, noting hole identification, box number, and metreage was stapled to the top end of the core box for easy identification while stored. Geotechnical data was collected by a supervised geotechnician or by the logging geologist. Data collected for all drill holes included recovery, rock quality data, magnetic susceptibility, and specific gravity. The logging geologist also recorded lithology, alteration, mineralization, and structural data. The geologist marked sampling intervals for assay analyses, and inserted quality assurance and quality control (QA/QC) samples at regular intervals along the core.

Once logging and sampling was completed, the core was photographed wet, with the hole ID, box number, and start/end meterages clearly visible. The core boxes were transferred from the logging facility to the core cutting shack and stacked in numerical order to prevent confusion when cutting the core. Tagged and labelled sample bags were provided to the core cutting technician specific to the drill hole being sampled. The core was cut in half and placed into the clear plastic sample bags. The remaining half core was placed back into the core boxes and stacked outside the core shed on a wooden palette. Once a complete hole was cut, the core boxes were capped, banded, and taken to the core storage location.

10.3 Surface Drilling Methods

10.3.1 Legacy

Limited details are available regarding drilling contractors and drilling procedures specific to each campaign prior to 1995.

10.3.1.1 1995-1997

Most of the drilling around the mine workings was completed by Hy-Tech Drilling of Smithers, B.C. (Hy-Tech). Hy-Tech used up to three drill rigs that included a JKS-300 which drilled BQTK (thin wall) core (40.7 mm), and two F-15 drill rigs which drilled NQTK (thin wall in 1996) (50.6 mm) and NQ2 (50.6mm) in 1997.

In 1996, Advanced Drilling of Vancouver completed four holes using a Boyles 56 drill rig.

No casing for the 1995 program survived the winter snow removal since they were all located near, or on, the mine access road. Casing was left in most of the holes from 1996 and 1997. All holes were grouted provided that the casing was still intact. All holes drilled in 1996 and 1997 were marked with a yellow wooden stake and aluminum tags marked with the drill hole number.

10.3.1.2 1998

Hy-Tech completed all holes of the 1998 campaign using four drill rigs including two JKS-300 rigs which drilled BQTK (thin wall) core and two F-15 rigs which drilled NQ2 core (with the capability of reducing to BQTK or BQ (36.4 mm).

None of the holes completed during the 1998 drilling campaign were grouted. This was due partially to the ineffectiveness of the material used during the 1997 campaign and also due to the initiation of the mine closure plan.

10.3.1.3 2004

Hy-Tech completed all drill holes during the 2004 summer and winter drilling campaigns. Three drill rigs were used, including one JKS-300 rig which drilled both BQTK (thin wall) core and NQ2, and two modified F-15 rigs, which drilled NQ2 core (with the capability of reducing to BQTK or BQ).

All the drill holes were sealed using Volclay grout and a 15 m cement cap at the overburden/bedrock interface. The casings were left in for holes C04-1248 to C04-1272, but they were removed for all other holes and plugged with a yellow or orange steel cap with the appropriate drill hole number marked on the surface. In the longer holes (i.e., Deep Adrian and Deep West Limb), an additional 15 m cement plug was placed in the HW Andesite unit, immediately below the Bowser fault.

10.3.2 Skeena

10.3.2.1 2018

From August 15 to November 6, 2018, Skeena completed 46 exploration core drill holes from 12 drill platforms totalling 7,737.45 m. Drilling targeted the 21A, 21C and 22 Zones. The purpose of the drill program was to infill areas with low drill density and to collect fresh material for a metallurgical characterization program.

Drilling was conducted by DMAC Drilling Ltd. (DMAC) of Aldergrove, B.C. and Hy-Tech. DMAC used a Hydracore 2000 hydraulic skid-mounted drill rig on the 21A and 21C Zones and converted the drill to a fly rig for drilling on the 22 Zone. Hy-Tech used a Tech 5000 fly rig. Drill hole collars were initially located using handheld global positioning system (GPS) units and surveyed at the end of the drill program using a Trimble differential GPS (DGPS) instrument. Down hole orientation surveys were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool.

Drill core was logged and sampled at core logging facilities located just inside the Eskay Creek Mine site gate, proximal to Argillite Creek. Drill core is a combination of NQ (47.6 mm) (Hy-Tech) and NQ2 (DMAC) diameter core. As weather conditions deteriorated with the onset of winter, all logging and sampling operations were moved to the QuestEx Gold and Copper Ltd.’s core facilities located at the McLymont Creek staging area in the Iskut Valley. Core is stored at both the Eskay Creek Mine site carpentry shop and McLymont Creek staging area.

Helicopter drill moves, and daily drill support was provided by Silver King Helicopters Inc. of Smithers, B.C. (Silver King) using a Eurocopter AS350 B2 helicopter.

10.3.2.2 2019

From August to December 2019 Skeena completed 203 exploration diamond drill holes totalling 14,091.87 m. The purpose of the drill program was to infill areas with low drill density and upgrade the mineral resource categories. The drilling targeted the 21A, 21B, 21E and HW Zones.

Drilling was conducted by Tahltech Drilling Services Ltd. (Tahltech) (a partnership between the Tahltan Nation Development Corporation – TNDC and Geotech Drilling Services Ltd.), using a Hydracore 2000 hydraulic skid-mounted drill rig. Drill hole collars were initially located using handheld GPS units and surveyed at the end of the drill program using a Trimble DGPS. Down hole orientation surveys were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool. Drill core was NQ size.

Drill core was logged and sampled exclusively at core logging facilities located at the McLymont Creek staging area in the Iskut Valley. All drill core is stored at McLymont Creek Staging area.

10.3.2.3 2020

From February 7 to October 10, 2020, with a hiatius from March to July due to Covid 19 restrictions, Skeena completed 197 core drill holes totalling 36,582.56 m from their Phase 1 drill program. Drilling targeted zones outside of the 20 m buffer zone imposed by Barrick around the underground workings.

From October 10 to December 31, 2020, Skeena drilled 276 holes for 43,340.23 m from their Phase 2 drill program targeting zones inside the 20 m buffer zone.

The purpose of the 2020 drilling was to support upgrade of the mineral resource confidence categories in the 22, 21A, 21C, 21B, 21E, PMP Zones, as well as to test for mineralization in the Lower Mudstones below the 21A Zone and the Water Tower Zone. Exploration drilling in the Tom MacKay area was also conducted.

Three contractors were used throughout the year including: Tahltech, ITL Diamond Drilling Ltd., (ITL) and Konaleen Drilling Ltd. (Konaleen).

Tahltech used Hydracore 2000 drills for both skid and heli drilling, ITL used a DrillCo rig, which was used only for heli drilling, and Konaleen Drilling used a Zinex A5 drill for skid drilling. Helicopter drill moves, and daily drill support was provided by Silver King using Eurocopter AS350 B2 helicopters. All drill core was NQ in size.

Drill hole collars were initially located using handheld GPS units and surveyed at the end of the drill program using a Trimble DGPS. Down hole orientation surveys were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool. Drill core was NQ size.

Drill core was logged and sampled exclusively at core logging facilities located at the McLymont Creek staging area in the Iskut Valley. All drill core is stored at McLymont Creek Staging area.

10.3.2.4 2021

The remaining 29 holes for 2,988.00 m of the Phase 2 program was completed between January 1 to January 11, 2021, by Tahltech and Konaleen using the same rigs and procedures as the 2020 drilling.

10.3.3 Discharge of Water

In exploration activities near underground workings, discharge of artesian flows from drill holes that intersect mine workings may constitute an unauthorised waste discharge under the B.C. Environmental Management Act. In accordance with this applicable legislation, and Skeena's Principles for Responsible Exploration, artesian flow from exploration drill holes had to be controlled and prevented. Skeena instructed each drill contractor to comply with the following procedure: all drill holes had to be cased into solid bedrock and then cemented. Once the cement had cured the drill holes had to be pressure tested to a minimum of 300 psi to test for leakage. If the pressure test failed, casing was re-drilled and the whole casing-pressure testing process was repeated until the pressure test passed. Upon completion of each hole a Van Ruth plug was installed at depth, the drill hole above the plug was filled to surface using Microsil anchor grout, and then capped with a threaded metal cap.

10.3.4 Site Reclamation

Upon completion of the drill holes, all man-made materials and set-up timbers were removed from the drill sites and all trees felled were cut into 1.3–2 m lengths. Before and after pictures were taken at each site and then submitted to the BC provincial government as part of the Notice of Closure.

10.4 Recovery

10.4.1 Legacy

Skeena does not currently have access to the legacy rock quality designation (RQD) and recovery data.

10.4.2 Skeena

Drilling undertaken by Skeena during 2018 to 2021 had excellent core recoveries, with core recovery averaging 95%.

10.5 Sample Lengths/True Thickness

10.5.1 Legacy

Drill hole spacing throughout the deposit ranges from 5 m, where underground production drilling encountered complex areas, to 25 to 50 m at the surface. The average drill spacing throughout the deposit is 10–15 m.

For surface holes, mineralization true width varies but approximates 70–100% of drilled width; for underground drill holes positioned on singe platforms and drilled in radiating fans, true drilling widths are more variable.

10.5.2 Skeena

The sample lengths were determined during logging by the geologist. The average sample length for drill holes ranged from 1.0 m in the Contact Mudstone, 1.5 m in the Eskay Rhyolite and 3.0 m in the Hanging Wall Andesites. Samples were generally broken on geological contacts, leading to some samples being as short as 18 cm. As the holes cut the mineralization at different angles, they all have different true widths. In general, the true width is estimated to be 70–100% of the interval length.

10.6 Underground Drilling

10.6.1 Legacy

Underground drilling began in 1991. Information regarding field procedures are largely incomplete or missing. Little detail is known about the amount of definition drilling completed per year or the type of drill rigs used.

The deposit is drilled at an average spacing of 10 m using BGM (~40 mm) core diameter. In highly complex areas where mining was active, drill spacing was locally reduced to 5 m. Underground drill holes are generally less than 100 m in length.

Collar location surveys were performed by the mine surveyors. These provided accurate collar locations for the drill holes, and a check on the initial azimuth and dip was recorded for each hole. Prior to 2004, most of the drill holes in the database were surveyed downhole using a Sperry Sun Single Shot instrument, with readings taken every 60 m, or by acid tubes, with readings every 30 m. In early 2004, downhole surveying used an Icefield Tools M13 instrument. This provided azimuths and dips for each hole every 3 m down the drill hole. Readings were reviewed by staff and inaccurate entries were removed from the database.

10.6.2 Skeena

No underground drilling has been undertaken by Skeena.

10.7 QP Comments on "Item 10: Drilling"

The QP considers that the quantity and quality of the logging, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation. Drill orientations are generally appropriate for the mineralization style for the bulk of the deposit area. No factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Sample preparation, analyses, and security results and protocols for drilling campaigns before 2018, the year that Skeena optioned the Eskay Creek Project, are documented in Appendix B. Skeena performed a rigorous analysis of the historical data prior to adopting into their database.

11.1 2018 – Early 2021 Analysis

11.1.1 Sample Preparation and Assaying Procedures

Skeena's sampling and assay quality control guideline for the Eskay Creek drill core programs was reviewed by SRK (Skeena, 2019). This quality control guideline is a comprehensive document that is designed to assist staff in the implementation and ongoing monitoring of assay quality data for all present and future drill programs. The guideline provides definitions and instructions for all stages of core handling, preparation and analysis with which Skeena personnel are expected to follow (see Section 10.3 for details on drill rig specifications and drill site procedures as well as core storage locations).

Drill core logging, photography, and sampling are conducted in a systematic and vigilant manner. When drill core arrives at the core shack, the geologist rearranges the core so that the pieces fit back together as best as possible. The geologist then checks the core for any depth marker discrepancies or core interval mix-ups before making the applicable correction(s). Boxes are labelled at the start and end of the boxes, in metres, and then cleaned of any mud or contaminants. The core is photographed under wet conditions. The core is logged by a geotechnician for recovery, RQD, longest stick, and magnetic susceptibility. Specific gravity samples are collected at the rate of one in every 20 m down the hole. A whole piece of NQ-sized competent core 10–15 cm in length is selected and measured using the water displacement method.

A geologist is assigned to a drill hole and logs the core for lithology, alteration, veining, mineralization, and structural features. All metrics, depending on the geological feature being evaluated, are assessed in percent abundance or intensity rankings as well as orientation and thickness. One-metre assay intervals are established when visible mineralization is first observed, and then uniform intervals are continued down the drill length until there is no evidence of mineralization. Assay intervals honour geological contacts to a minimum of 0.5 m and a maximum of 1.5 m. Skeena records geological and geotechnical information into a GeoSpark database.

Skeena geologists mark the centre line of the core in red wax pencil in preparation for core cutting. All drill core is halved with a diamond core cutting saw. One assay sample ticket stub is placed into the sample bag with the half core and the other matching ticket stub is stapled into position onto the core box marking the appropriate assay interval.

Samples are shipped using the following procedure: groups of samples are placed in a large rice bag and secured with tie wraps. The sample number series within the sack are marked on the outside of the rice bag and a laboratory sample submission form is placed in the first rice bag in sequence. The laboratory is emailed in advance of the shipment, and when the laboratory receives the shipment a confirmation email is returned. Assay sample shipments are shipped to the assay facility in Kamloops twice per week. Samples were transported by truck from the Eskay site to the McLymont staging area by Skeena personnel and then loaded onto trucks driven by Rugged Edge Holdings (Skeena's expediter). The samples are then delivered to Bandstra in Smithers and transported from there to the ALS preparation facility in Kamloops (ALS Kamloops). All samples are initially sent and prepared at ALS Kamloops, after which the pulp samples are split and shipped for analysis to the ALS laboratory in Vancouver (ALS Vancouver), an ISO/IEC 17025:2005 accredited laboratory for selected analytical techniques. ALS is independent of Skeena.

Reject and pulp materials are temporarily stored with ALS Vancouver for up to one year after the original sample has been tested. All temporarily stored materials are discarded thereafter; however, most original half core is appropriately maintained at the McLymont Creek staging area.

At the preparation facility in Kamloops the entire sample is dried and then crushed using a two-stage Terminator crusher. Crushing is done to better than 70% passing a 2 mm Tyler 10 mesh screen, and then the crushed material is put through the riffle splitter to 1,000 g. Roughly 1,000 g is taken and pulverized to better than 85% passing a 75-µm Tyler 200 mesh screen (PREP-31BN). The LM2 pulverizing mill is equipped with a B2000 bowl.

At ALS Vancouver, gold assays were performed on 50 g samples by fire assay and atomic absorption (ALS code: Au-AA26) with a lower and upper detection limit of 0.01 g/t and 100 g/t, respectively. For assays above the upper detection limit then samples were analysed by fire assay with a gravimetric finish (ALS code: Au-GRA22) with lower and upper detection limits of 0.05 g/t and 10,000 g/t Au, respectively.

Silver assays were performed on 50 g samples by fire assay and gravimetric finish (ALS code: Ag-GRA22) with lower and upper detection limits of 5 g/t and 10,000 g/t, respectively. For assays above the upper detection limit, a concentrate and bullion grade fire assay and gravimetric finish were performed (ALS code: Ag-CON01) with lower and upper detection limits of 0.7 g/t Ag and 995,000 g/t Ag, respectively.

Multi-element assays were performed using a combination of digest and finish methods: a 0.25 g sample using a four-acid digest followed by an ICP-AES finish (ALS code: ME-ICP61), and a 0.1 g sample using lithium borate fusion followed by an ICP-MS finish (ALS code: ME-MS81). This combination in assay methods for the multi-elements ensured that the range of concentrations for all elements of interest, particularly for antimony, were covered. In the database, the ICP-AES finish method took precedence.

A limited number of samples exceeded the upper limits for silver, arsenic, copper, lead, and zinc. For these samples, the laboratory was instructed to apply overlimit methods on a 0.4 g sample (ALS code: OG62) using a four-acid digest and ICP or AAS finish. Sulphur overlimits were re-analysed using the total sulphur Leco furnace method using a 0.1 g sample (ALS code: S-IR08) with a lower detection limit of 0.01% and upper detection limit of 50%.

Mercury was separately analysed using low temperature aqua regia digestion followed by an ICP-AES finish (ALSO code: Hg-ICP42) with a lower detection limit of 1 ppm and an upper detection limit of 100,000 ppm.

11.1.2 QA/QC Verifications 2018–Early 2021

Skeena implemented formal QA/QC programs for all phases of drilling between 2018 and early 2021. In total four drilling phases were completed, including 2018, 2019, 2020 Phase 1, and 2020 Phase 2.

The QA/QC programs contained the following types of quality control samples: sample blanks, certified reference materials (CRMs), and check assays. In addition to the Skeena-introduced QC samples, ALS Vancouver inserted their own independent check samples.

The blank material used was a marble garden rock obtained from Canadian Tire in Smithers, BC. Approximately 1 kg of this material was used for each blank sample. Three blanks were inserted for every 100 samples, typically at the “20”, “60” and “00” numbers in the sample tag sequence. Assays for blanks should be less than 10 times the detection limit of the analytical method for gold.

CRMs were inserted for every 100 samples, typically at the "10", "30", "50", "70" and "90" numbers in the sample tag sequence. CRMs were usually inserted in rotation, except where high-grade intervals above approximately 20 g/t Au were encountered; here high-grade CRMs (CDN-GS-25) were inserted.

CRMs and blanks were monitored when batches of assay data were first received. CRM or blank control charts were routinely updated for the following elements: gold, silver, copper, lead, and zinc; other elements were analysed on an as needed basis. Table 11-1depicts the 10 CRMs used and their expected values and standard deviation for gold and silver.

Table 11-1: List of Certified Reference Materials (Au and Ag recommended values)

Certified Reference Material	Year	Gold (g/t)			Silver (g/t)		
		Recommended value	+ 3 Std dev	- 3 Std dev	Recommended value	+ 3 Std dev	- 3 Std dev
CDN-GS-1T	2018	1.08	1.23	0.93	n/a	n/a	n/a
CDN-GS-25	2018-2020	25.60	27.01	24.19	99.5	110.5	88.3
CDN-GS-5T	2018	4.76	5.075	4.445	126	141	111
CDN-ME-1312	2018-2020	1.27	1.495	1.045	22.3	24.85	19.75
CDN-ME-1601	2018	0.613	0.682	0.544	39.6	42.3	36.9
OREAS 603b	2019-2020	5.21	5.837	4.583	297	321	273
OREAS 622	2019-2020	1.85	2.048	1.652	102	111.9	92.1
CDN-ME-1902	2020	5.38	6.01	4.75	356	384.5	327.5
CDN-GS-13A	2020	13.2	14.28	12.12			
Arsenic							
Cd-1	2019-2020	3.57					

Control charts for CRMs were prepared using the acceptable value plus or minus three standard deviations, to provide the acceptable range. If analyses were outside of the acceptable range after checking for data entry errors, then repeat assays were requested. Where two or more consecutive CRMs were both biased high or low (more than 105% of the expected value or less than 95% of the expected value) repeat assays were requested. The laboratory was instructed to retrieve five pulp samples before and after the QC failure.

Two kinds of duplicates were processed during all drilling program: preparation and pulp duplicates. The preparation duplicate is a split that the laboratory takes from the reject material at a rate of one in every 50 samples. The pulp duplicate is an exact repeat of the primary pulp sample analysed immediately after the original sample. Pulp repeat insertion rates are at the discretion of the Laboratory Manager. Preparation and pulp duplicate data sets were routinely charted using X-Y scatterplots, relative percent difference versus average graphs and quartile-quartile plots. Skeena monitored the laboratory's performance and reported any concerns to the Laboratory Manager.

Five CRMs were used during the 2018 Phase 1 drilling program, all of which were obtained from CDN Resource Laboratories in Langley, British Columbia (CDN). One CRM was certified for gold only (CDN-GS-1T), two were certified for gold and silver only (CDN-GS-5T and CDN-GS-25), and two were polymetallic CRMs certified for gold, silver, copper, lead and zinc (CDN-ME-1312 and CDN-ME-1601). All CRMs were purchased from CDN; they were selected to best match the rock matrix seen at Eskay Creek, as well as to match the analytical method used on the samples.

A total of 112 control blanks, 196 CRMs, 206 preparation duplicates, and 1,178 pulp duplicates were inserted and analysed in 2018 (Table 11-2). The combined quality control samples equate to 51% of the total assays submitted in 2018.

Table 11-2: QC Samples Phase 1 Drilling Program, 2018

QC Sample	Type	Subtotal	Total	% of Total
Total Blanks			112	7%
CRMsl	CDN-GS-1T	2		
	CDN-GS-25	44		
	CDN-GS-5T	58		
	CDN-ME-1312	48		
	CDN-ME-1601	44		
Total CRMs			196	12%
Duplicates (internal ALS)	Prep	206		
	Pulp	1,178		
Total Duplicates			1,384	82%
Total QC			1,692	100%

Five CRMs were used in the 2019 Phase 1 drilling program, two of which originate from CDN, and two from Ore Research & Exploration Pty Ltd. (OREAS), through Analytical Solutions Ltd. in Ontario. An additional high-grade antimony CRM (Cd-1) was obtained from Natural Resource Canada in Ottawa, Ontario and inserted, at the geologist's discretion, in zones of massive stibnite. Cd-1 originates from stibnite-bearing quartz veins in greywacke and slate from Lake George mine, New Brunswick (Skeena, 2019a). A total of 281 control blanks, 466 CRMs, 28 preparation duplicates, and 1,504 pulp duplicates were inserted and analysed in 2019 (Table 11-3). The percentage of combined quality control samples equates to 27% of the total assay samples submitted in 2019.

Table 11-3: QC Samples Phase 1 Drilling Program 2019

QC Sample	Type	Subtotal	Total	% of Total
Total Blanks			281	12%
CRMs	CDN-GS-25	123		
	CDN-ME-1312	112		
	OREAS 603b	115		
	OREAS 622	114		
	Cd-1	2		
Total CRMs			466	20%
Duplicates (internal ALS)	Prep	28		

	Pulp	1,504		
Total Duplicates			1,532	67%
Total QC			2,279	100%

Five CRMs were used during the 2020 Phase 1 and Phase 2 drilling programs, three of which originate from CDN, and two from OREAS (Skeena, 2020a; Skeena, 2020b). A total of 1,132 control blanks, 2,708 reference samples, 115 preparation duplicates, and 1,152 pulp duplicates were inserted and analysed in 2020 (Table 11-4). The percentage of combined quality control samples equates to 14% of the total assay samples submitted in 2020.

Table 11-4: QC Samples Combined Phase 1 & 2 Drilling Programs 2020

QC Sample	Type	Subtotal	Total	% of Total
Total Blanks			1,132	22%
CRMs	CDN-GS-25	664		
	CDN-ME-1312	678		
	OREAS 603b	689		
	OREAS 622	667		
	CDN-GS-13A	10		
Total CRMs			2,708	53%
Duplicates (internal ALS)	Prep	115		
	Pulp	1,152		
Total Duplicates			1,267	25%
Total QC			5,107	100%

11.2 Specific Gravity Analysis

Specific gravity (SG) measurements were routinely collected from drill core during Skeena's 2018, 2019 and 2020 Phase 1 and Phase 2 drilling campaigns. Sections of whole drill core up to 10 cm long were used to determine SG. The core was first weighed in air on a top-loading balance, and then weighed in water. A total of 4,965 SG measurements were taken and categorized according to dominant lithology type. Table 11-5 shows the nine dominant lithology types versus their average SG values.

Table 11-5: Specific Gravity versus Lithology

Rock Type	No. of Samples	Specific Gravity
Bowser Group Sediments	3	2.74
Hanging-wall Andesite	2,460	2.80
Hanging-wall Sediments	606	2.72
Contact Mudstone	225	2.78
Rhyolite	1,496	2.66

Rock Type	No. of Samples	Specific Gravity
Lower Mudstone	36	2.79
Footwall Andesite	43	2.75
Footwall Dacite	78	2.78
Even Lower Mudstones	18	2.75
Average	**4,965**	**2.75**

Specific gravity was coded into the resource model using these rock types. An additional unit was designed and inserted for the predominantly barite-rich unit: the 21C mudstone. In addition, a default value of 2.67 was applied to blocks for which lithology had not been coded. This is the average value of unmineralized rhyolite and mudstone host rocks combined.

Resource models prior to 2021 used an SG formula derived experimentally from actual measurements and analyses when the Eskay Creek mine was in historical production. This formula was used for all Mineral Reserves estimated on site so that SG could be determined for mineralized intervals that did not have directly measured values. The formula historically used was:

$$SG = (Pb + Zn + Cu) * 0.03491 + 2.67 \text{ (where all metals are reported in \%)}.$$

11.3 SRK Comments

In the opinion of SRK the sampling preparation, security and analytical procedures used during the years 2018 to 2021 are consistent with generally accepted industry best practices and are therefore adequate for resource estimation use. The quality control programs established for Skeena's 2018, 2019, 2020 Phase 1 and Phase 2 programs adequately tested for sample mix-ups, contamination, sample bias, sample accuracy and precision using a collection of reference materials and blanks. All quality control issues were immediately addressed, and repeat batches were conducted if questionable data was encountered. Monthly quality control reports documented the type, quantity, and outcome of the quality control assessment, all of which show good performance and assay data integrity.

12 DATA VERIFICATION

12.1 Verifications by SRK

The database used for the 2021 Mineral Resource estimate was submitted to SRK on March 9, 2021 (the close out date for the database) for a final review before Skeena proceeded with generating mineralization domains. Skeena ensured that the database inherited from the historical Operator was verified using historical assay certificates and logs. SRK conducted an independent review of the historical database as well as the current database used for the 2018, 2019, 2020 Phase 1 and Phase 2 drilling programs. In addition, SRK reviewed the historical and current QA/QC programs and independently analysed the results from these programs. After the review, SRK concluded that the database was sufficiently reliable for resource estimation.

Note that although the resource has been estimated for the base metals (lead, copper, and zinc) and deleterious metals (arsenic, mercury, and antimony), and metallurgical elements (iron and sulphur), the database verifications and validations are primarily focused on gold and silver assays. At the request of SRK, the units for arsenic and antimony were changed from percent to ppm.

12.1.1 Current Database

The current database was provided to SRK in .csv format and included collar, survey, assay, and geology files for the 751 drill holes drilled during the 2018, 2019, 2020 Phase 1 and Phase 2 drilling programs, as well as all historical holes (for a total of 8,334 holes). SRK inspected the data for collar survey discrepancies, erroneous downhole deviation paths, and overlapping or missing assay and lithology intervals.

SRK performed an independent analysis of Skeena's database relevant to the 2019 and 2020 Phase 1 and Phase 2 drilling programs, whereby the database was compared directly with the provided assay certificates. Certificates for the 2019 and 2020 Phase 1 and Phase 2 drilling programs were imported into an SRK SQL database and validations were performed for the following eight assay values: Ag_Best_ppm, As_Best_ppm, Au_Best_ppm, Cu_Best_%, Pb_Best_%, S_Best_ppm, Sb_Best_ppm, and Zn_Best_%.

A total of 729 final certificates were imported, and out of the matching sample IDs, 100% of the 294,640 values had zero errors when programmatically compared.

12.1.2 Historical Database

The historical database was provided to SRK in .csv format and included collar, survey, assay, and geology files.

SRK conducted routine verifications to ascertain the reliability of the electronic drill hole database provided by Skeena. All assays in the database were verified against Eskay Mine laboratory and independent laboratory assay certificates, where assay certificates were available. No significant errors or omissions were discovered; however, the large number of missing assay certificates is a limitation on the validation effort.

The database was checked for missing values, duplicate records, overlapping intervals, sample intervals exceeding maximum collar depths, borehole deviations, drill holes collars versus topography, laboratory certificate versus database values, and special values (i.e. non-numeric or less than zero). Minor errors were reviewed with Skeena's Resource Geologist and resolved prior to geological modelling and resource estimation. All modifications to the database were checked to ensure appropriate allocation; these included assay priorities ranking and accurate, consistent lower detection limit (LDL) updates. The LDL is the lowest quantity of a substance that can be distinguished with a stated confidence level.

SRK viewed the collar locations of underground drill holes by means of 50 m sections with drill hole volume projections of 25 m. There was no obvious discrepancy between collar location and underground workings. Viewed on 50 m sections, the drill holes collars originating from the surface appear to correlate reasonably well with the topography layer. There are, however, several drill holes that occur approximately 20 m above or below the surface layer. Given the fact that the collar locations have more accurate spatial resolutions than the topography surface, this discrepancy is not thought to be a material concern. SRK cross-checked the UTM and mine grid coordinates from the McElhanney report with the final Skeena database. The checks confirmed that the UTM-mine grid shift had been done accurately.

12.1.3 Site Visit #1

Ms. S. Ulansky, P.Geo., visited the Eskay Creek Project on June 27 and June 28, 2018, with two representatives from Skeena (Ms. K. Dilworth and Mr. J Himmelright). The purpose of the visit was to see localities that had been described in earlier reports first-hand and to validate the areas with independent checks. The following areas were visited and verified:

- Approximately 50 drill hole collars, located on twenty-two drill pads, were located and resurveyed. GPS readings were taken along with general azimuth and dip orientations of the remaining casing. These independent GPS readings agreed within 1 m of the collar coordinates in the database, noting that the handheld GPS used by SRK had an accuracy of 5 m. All the drill holes surveyed were cased, although many casing caps were missing or not placed there in the first place. Seventeen of the drill holes had labels etched onto the casing caps and some of these locations were photographed (Figure 12-1);

- Five east–west trenches were visited, and their localities verified;

- The borrow pit that was used for making mine laboratory assay 'blank' samples;

- The historical regional exploration camp at km 45, which is now in the possession of another exploration company;

- Albino Lake, where all drill core and low-grade waste material was disposed.

Figure 12-1: Drill Hole Locations with Labelled Casing

Note: Figure prepared by S. Ulansky, June 27, 2018.

12.1.4 Site Visit #2

Ms. Ulansky's most recent site visit was conducted between July 27 and July 30, 2020. During this latest site visit, Ms. Ulansky reviewed surface and underground drill core to confirm the presence and nature of mineralization and appropriateness of the interpreted geological framework. She observed abundant mineralization in drill core, verifying the presence, and nature of gold and silver mineralization at the Eskay Creek Project.

In addition, while on site, she verified Skeena's drilling, sample preparation, handling, security, and chain of custody procedures, surface drill hole locations and core logs. Figure 12-2 depicts the core logging facility at the McLymont Creek staging area.

Figure 12-2: Core Logging Facility at the McLymont Creek Staging Area

Note: Photograph by Ms. S. Ulansky, 30 July, 2021.

12.1.5 Verifications of Analytical Quality Control Data

Skeena made available to SRK the historical assay results for analytical quality control data accumulated on the Eskay Creek property between 1997 and 2004. Although not complete, the Eskay Creek mine did initiate QA/QC measures into their sample stream in 1997. With progressive years the QA/QC protocol became more comprehensive and detailed. SRK independently compiled and summarized the QA/QC assays directly from the available assays for the years 1999, 2001, 2002, 2003 and 2004. The QA/QC data for the years between 1995 and 2004 showed satisfactory duplicate, blank and standard results.

SRK also independently verified Skeena's 2018, 2019, 2020 Phase 1 and Phase 2 drilling program QA/QC measures.

Table 12-1 summarizes all the QA/QC procedures in place in relation to the years that the samples were inserted.

Table 12-1: Drilling and Sampling Years versus QA/QC Procedure in Place

Year	Lab(s)	Type(s)	Certificate Availability
1997	Eskay mine lab	Repeat (pulp?)	No certificates found
1998	Eskay mine lab Bondar Clegg IPL MIN-EN ALS Chemex	Round robin standards, blanks, field, and pulp duplicates	No certificates found
1999	Eskay mine lab	Pulp repeats	Certificates found
2001	Eskay mine lab	Pulp repeats	Certificates found
2002	Acme Analytical	In-house standards, in-house pulp repeats	Certificates found
2003	Eskay mine lab	Unknown standards and blanks	Certificates found
	Acme Analytical	In-house standards, in-house prep, pulp and reject repeats	Certificates found
2004	Eskay mine lab	Standards, blanks, prep, pulp and reject repeats	Certificates found
	Acme Analytical	In-house standards, in-house prep, pulp and reject repeats	Certificates found
2018 - 2021	ALS Global	Reference material, blanks, in-house prep, and pulp repeats	Certificates available

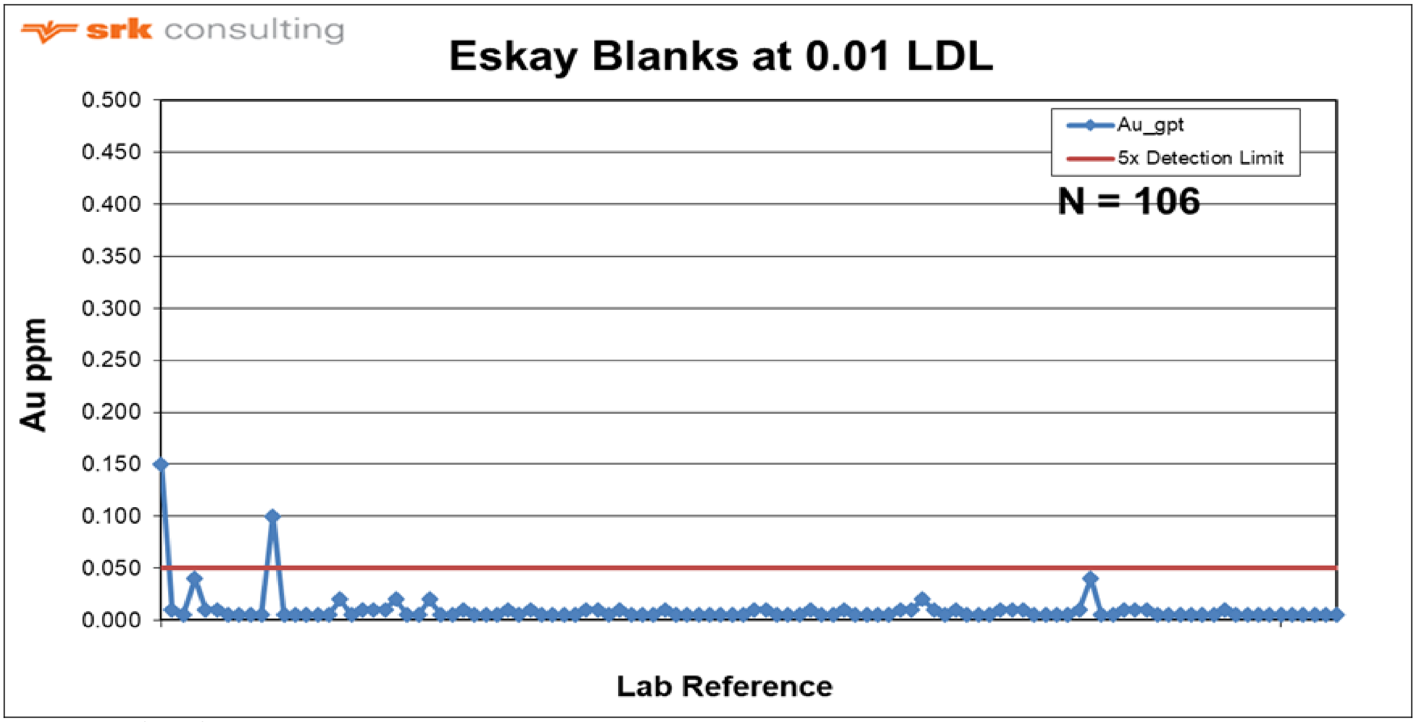
12.1.6 2018 – Early 2021 QA/ QC

Official QA/QC programs were undertaken in 2018, 2019, 2020 Phase 1 and Phase 2, whereby Skeena added standards and blanks to the sample stream and submitted them to the primary assay laboratory, ALS Global, for preparation and analysis. Preparation and pulp duplicates were processed at ALS Global during the routine sampling process. An additional laboratory (SGS Canada, located in Burnaby, BC(SGS Canada)) was used to independently test pulp duplicates and a select number of standards.SGS Canada is an ISO 9001:2015 accredited laboratory and is independent of Skeena.

12.1.6.1 2018

An analysis of 106 blank gold samples confirmed that the least amount of contamination was transferred from sample to sample (Figure 12-3). Two samples contained greater than five times the detection limit and follow up investigations show that one of them occurred immediately following a high-grade sample. Since the elevated blank sample was <1% of the previous high-grade sample result, it was deemed to be acceptable. No re-assays were requested for the blank results for the 2018 Phase 1 drill program.

Figure 12-3: **2018 Drilling Campaign Blank Results**



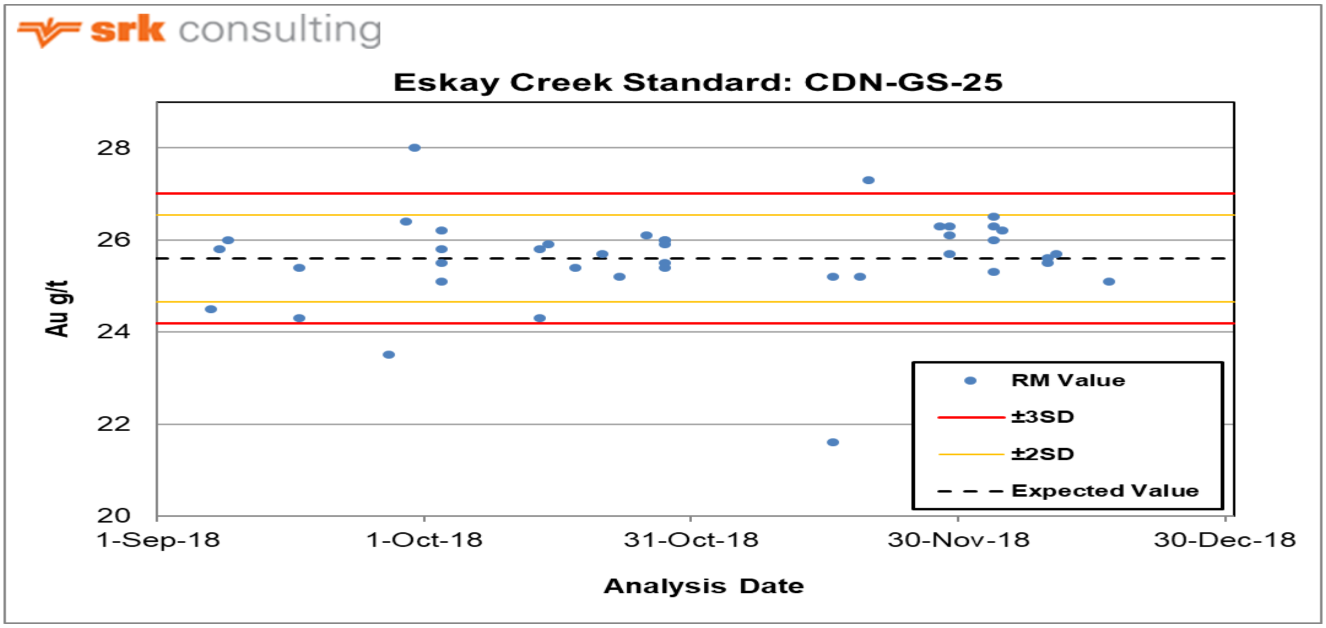
Note: prepared March 23, 2021

Five commercially produced CRMs were inserted into the sample stream during the 2018 Phase 1 drilling program. An analysis of CRM charts for gold showed no obvious errors or bias (Figure 12-4, Figure 12-5, Figure 12-6, and Figure 12-7). Several CRMs were mislabeled which were duly corrected during Skeena's QA/QC routine procedures. CRM CDN-GS-25 demonstrated an even spread about the expected value for gold, although several samples occurred outside of the three standard deviation limits (Figure 12-4). These samples were, however, within 10% of the expected value and are considered acceptable. One sample occurred outside of the 10% of the expected value but this sample was considered acceptable since it was introduced into a stream of low-grade assays.

CRM CDN-GS-5T demonstrates acceptable results for gold with one sample outside the three standard deviation limits but within 10% of the expected value (Figure 12-5). CRM CDN-ME-1312 showed one standard deviation more than 10% of the expected value and occurred within a series of medium- to high-grade gold assays (Figure 12-6). This CRM was re-assayed along with five to nine surrounding samples on each side of the failed samples. The re-assay results fitted within the acceptable limits.

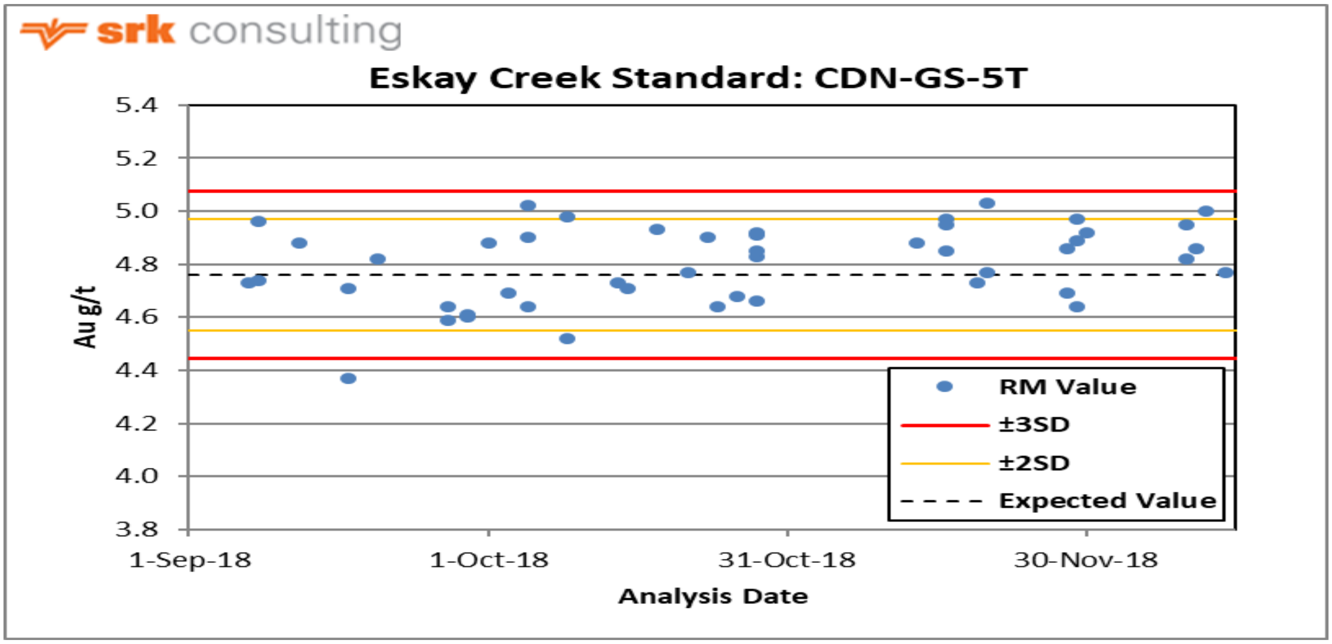
CRM CDN-ME-1601 resulted in several sample mislabels, which were duly corrected (Figure 12-7). Four samples occurred above the three standard deviation limit and above 10% of the expected value. These four samples occurred within low-grade assays, and it was not considered necessary or material to retest the surrounding assays.

Figure 12-4: Standard CDN-GS_25 From the 2018 Drill Campaign



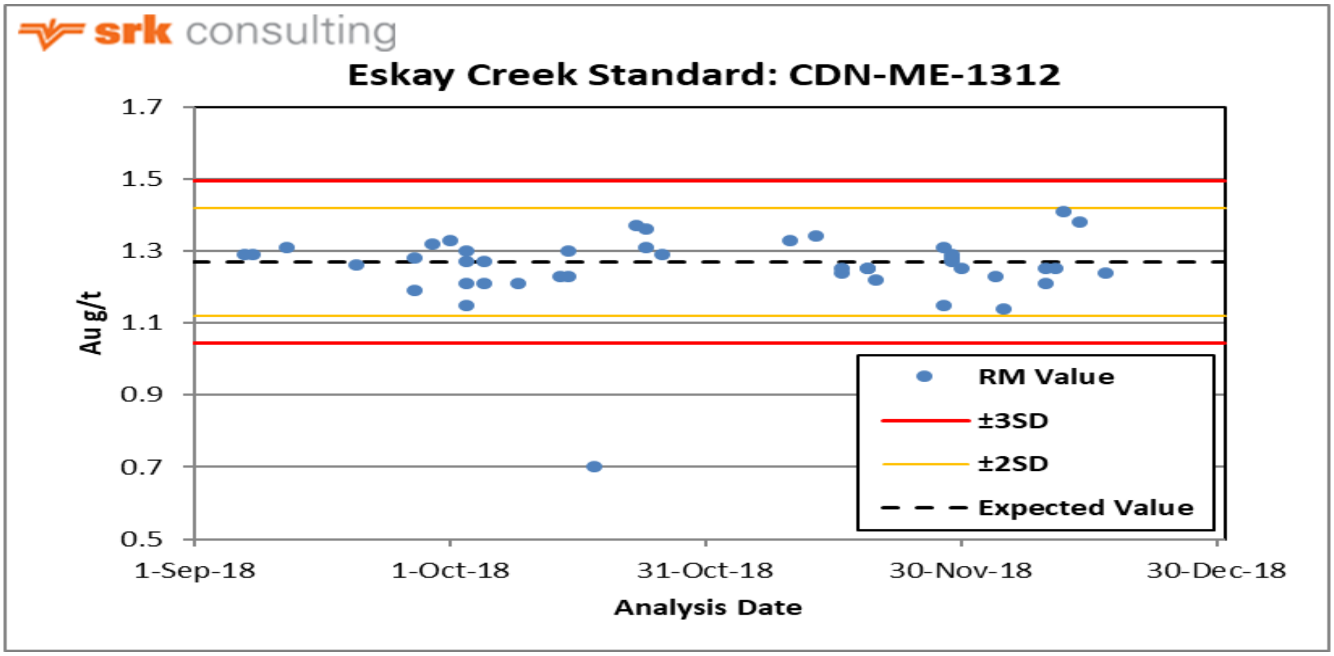
Date: prepared March 23, 2021

Figure 12-5: Standard CDN-GS-5T From the 2018 Drill Campaign



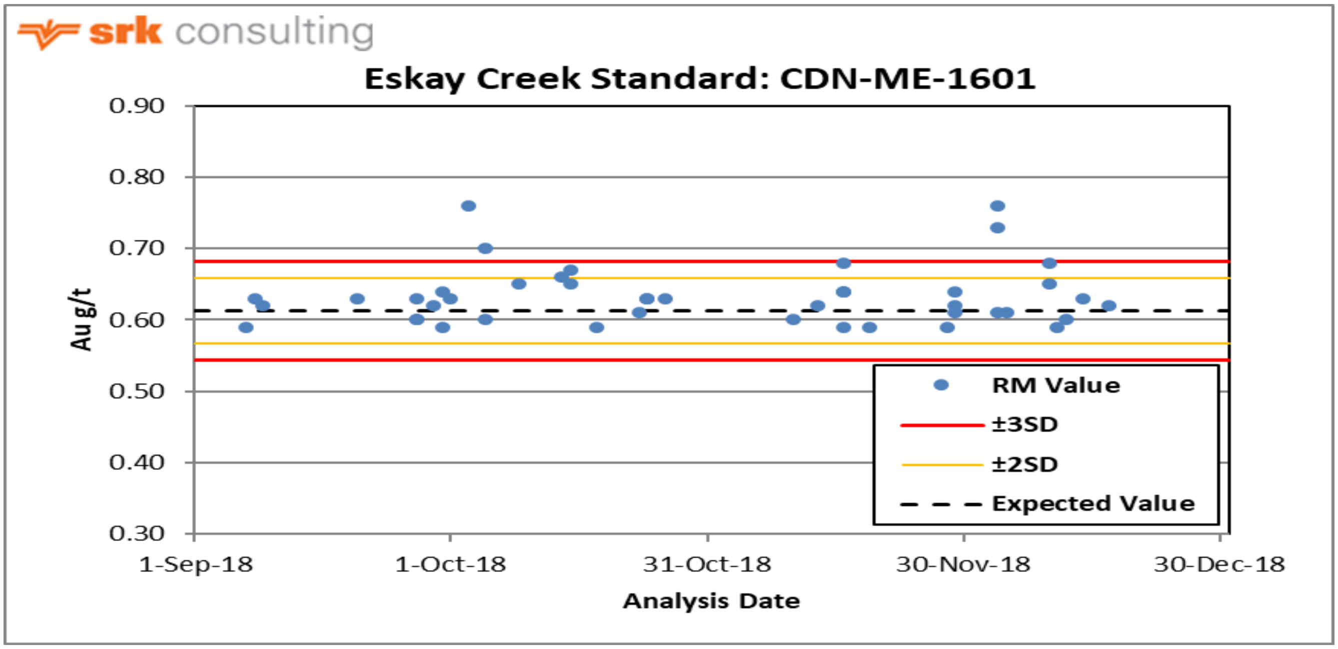
Date: prepared March 23, 2021

Figure 12-6: Standard CDN-ME 1312 From the 2018 Drill Campaign



Date: prepared March 23, 2021

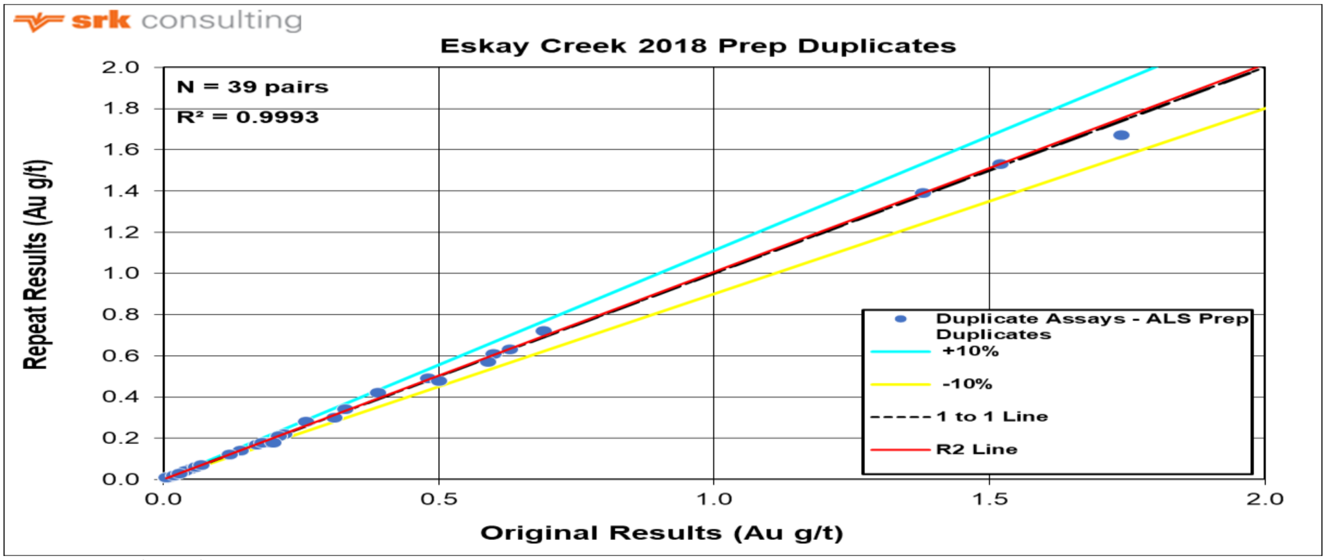
Figure 12-7: Standard CDN-ME-1601 From the 2018 Drill Campaign



Date: prepared March 23, 2021

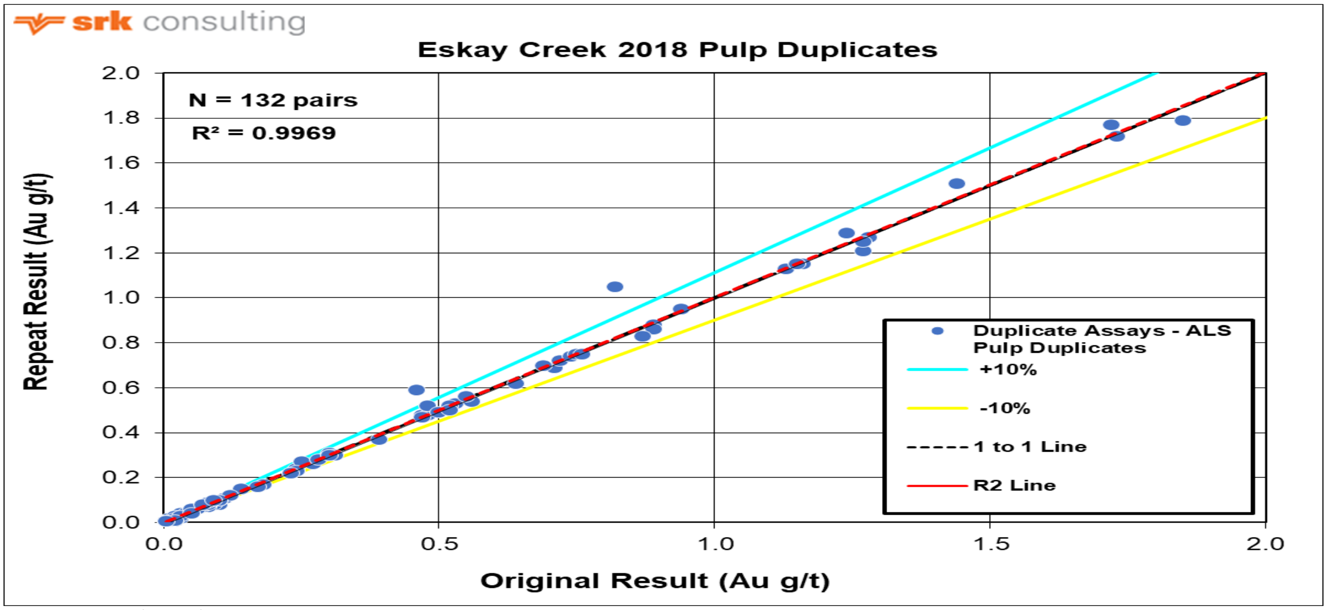
Preparation (rejects) and pulp duplicates were routinely run at ALS as part of the laboratory's internal QA/QC procedures. Paired preparation and pulp data performed within acceptable tolerance criteria at both lower grade and higher-grade values (Figure 12-8 and Figure 12-9).

Figure 12-8: Gold Prep Duplicate Samples From The 2018 Drill Campaign



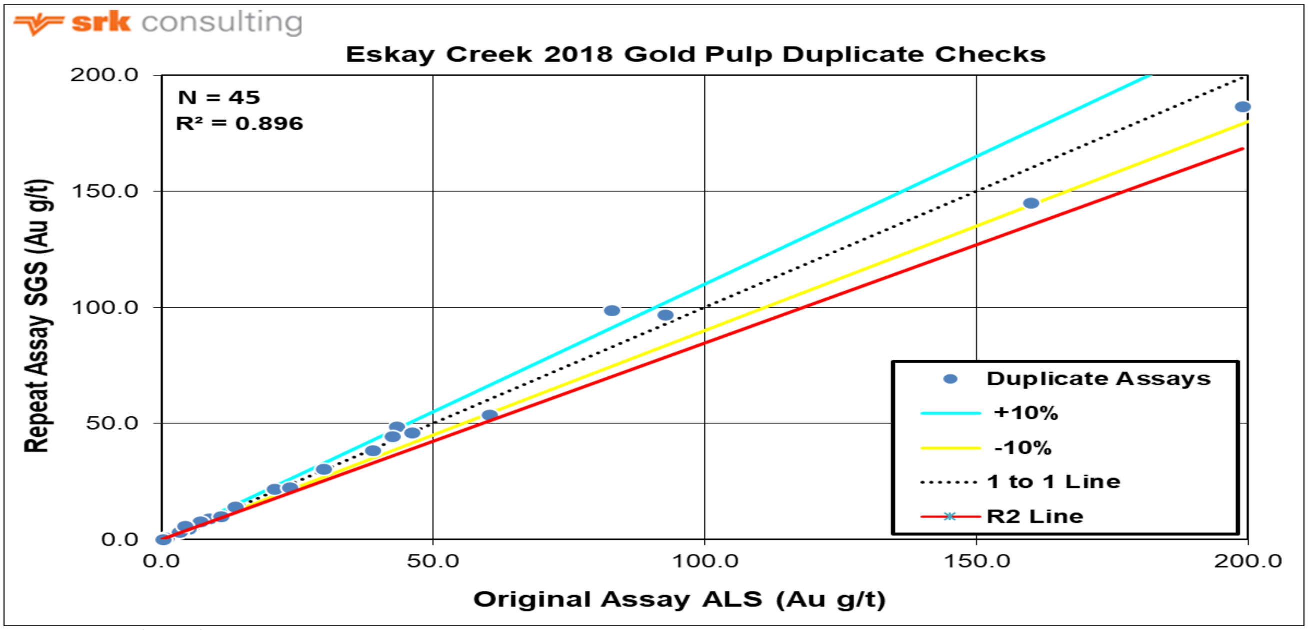
Date: prepared March 23, 2021

Figure 12-9: Gold Pulp Duplicate Samples From The 2018 Drill Campaign



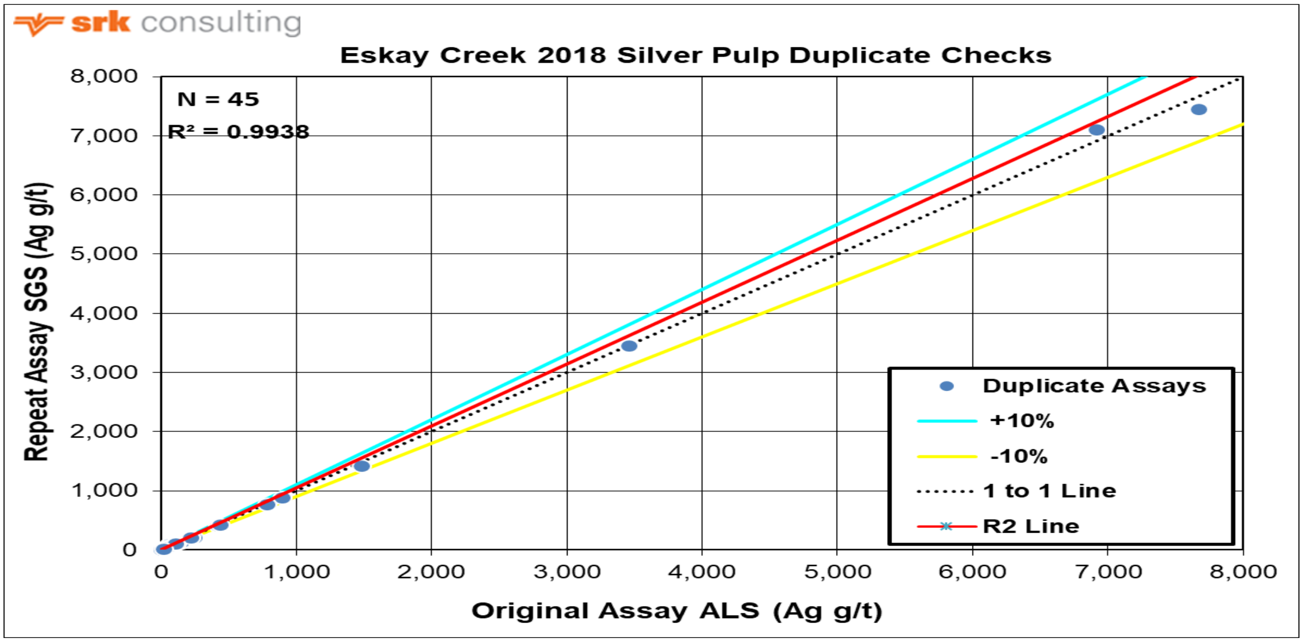
Date: prepared March 23, 2021

Figure 12-10: Gold Pulp Duplicate Check Samples From The 2018 Drill Campaign



Date: prepared March 23, 2021

Figure 12-11: Silver Pulp Duplicate Check Samples From The 2018 Drill Campaign

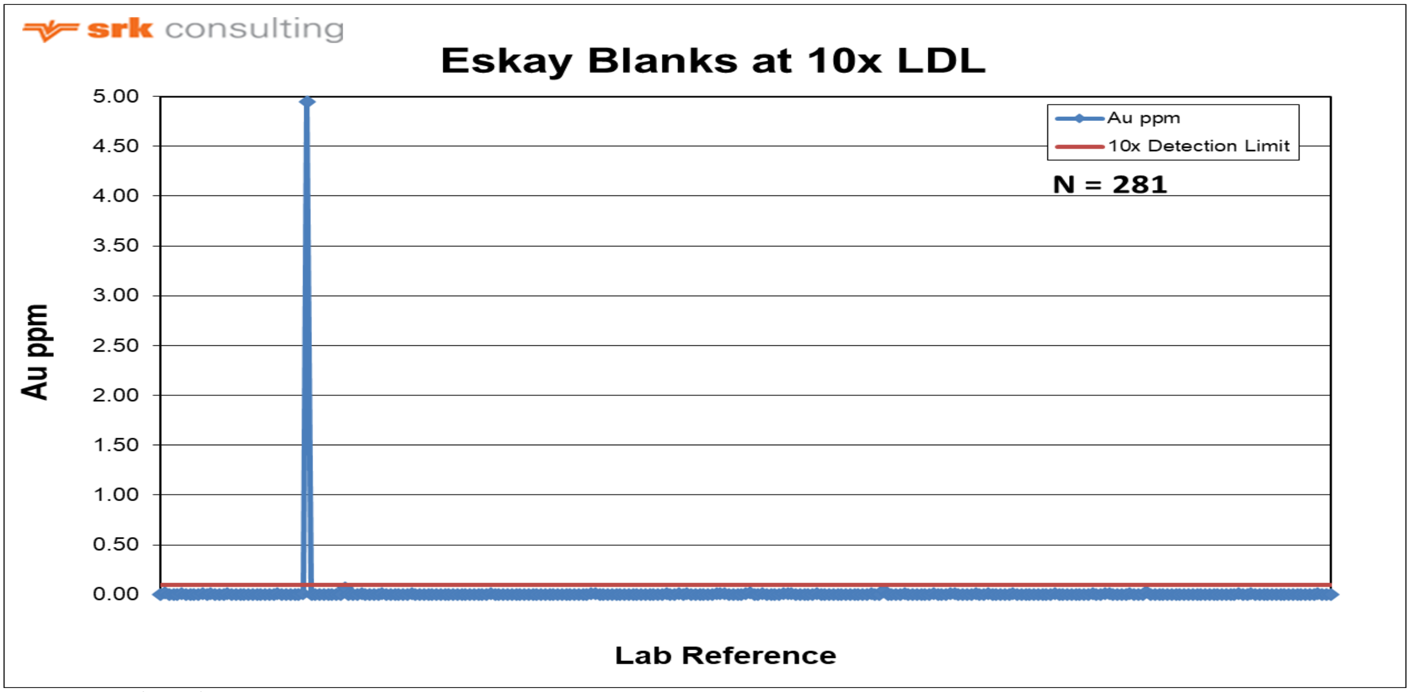


Date: prepared March 23, 2021

12.1.6.2 2019

A total of 281 control blanks were inserted during the 2019 drilling campaign. All except one sample returned less than 10x the detection limit (Figure 12-12). One gold control blank sample registered 4.95 g/t Au; however, this sample immediately followed an extremely high-grade result of 1,380 g/t Au (Skeena, 2019). It is reasonable to expect up to 1% carry over in a blank sample, and hence, no re-assays were run.

Figure 12-12: 2019 Drilling Campaign Blank Results

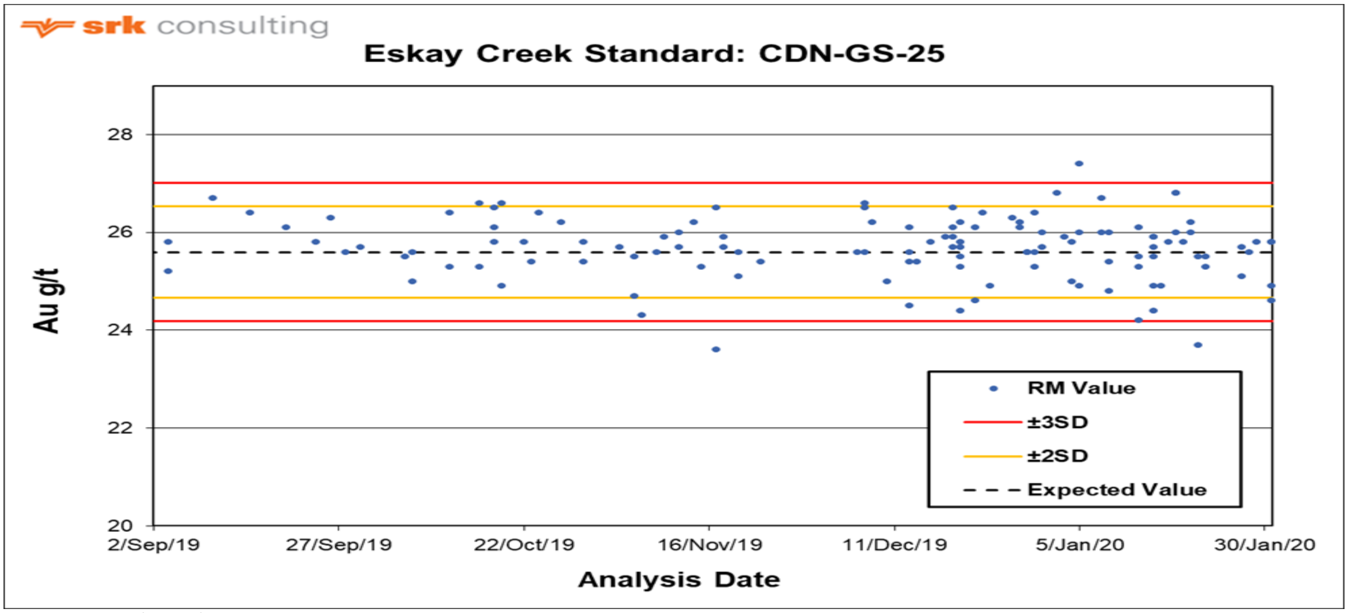


Date: prepared March 23, 2021

Five commercially produced CRMs were inserted into the sample stream during the 2019 drilling program. An analysis of 3 CRM charts for high, medium, and low gold grades showed no obvious errors or bias (Figure 12-13, Figure 12-14, and Figure 12-15). The overall failure rate for gold standards in the 2019 program was 0.6%, an inconsequential number of samples outside of the three standard deviation limits. CRM CDN-GS-25 demonstrated an even spread about the expected value for gold, although a few samples occurred outside of the three standard deviation limits (Figure 12-13).

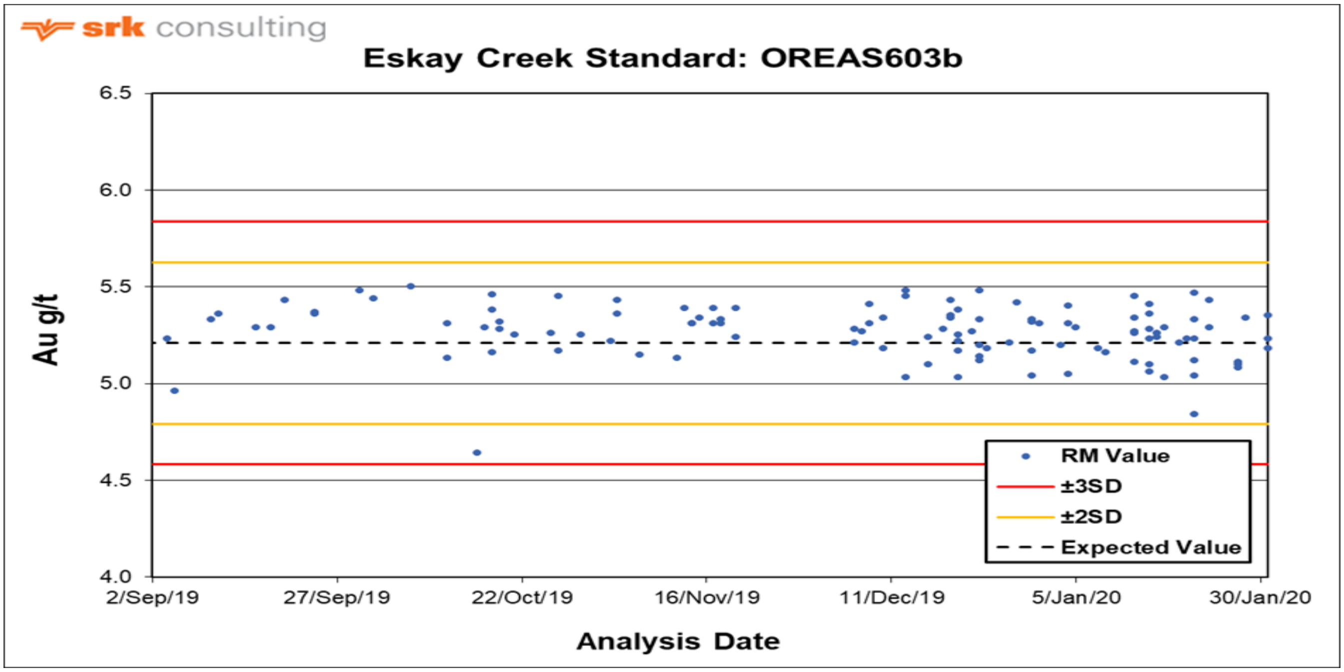
CRM OREAS603b demonstrated acceptable results for gold with all samples falling within the three standard deviations value (Figure 12-14). Similarly, CRM CDN-ME-1312 results are evenly spread about the expected value and occur wholly within the three standard deviation limits (Figure 12-15).

Figure 12-13: Standard CDN-GS-25 From The 2019 Drill Campaign



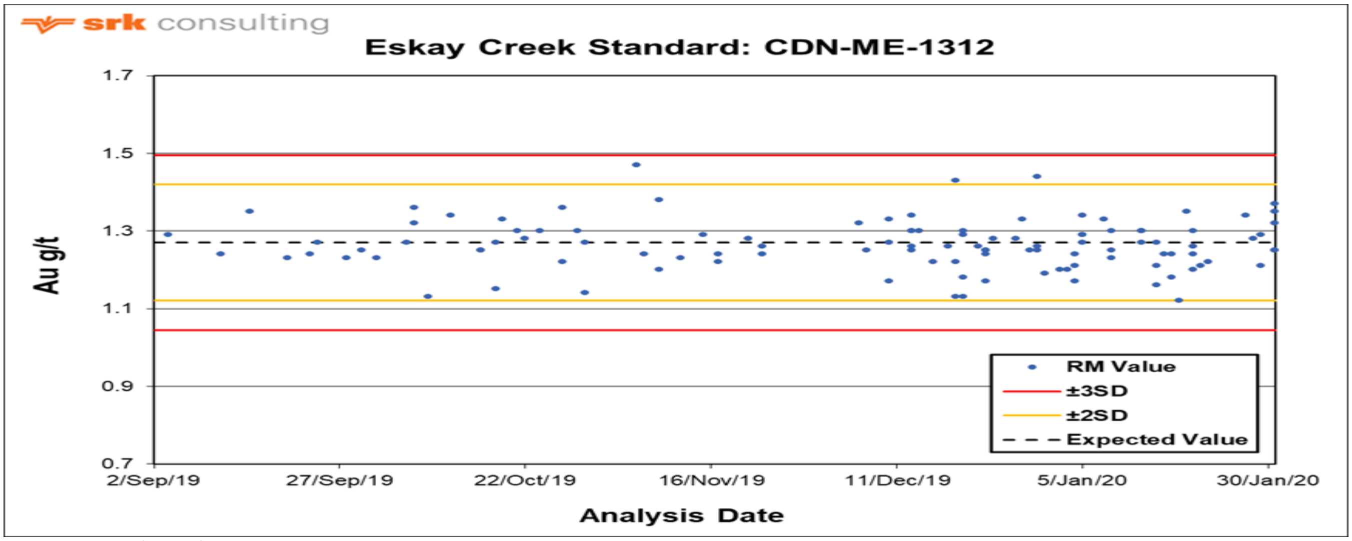
Date: prepared March 23, 2021

Figure 12-14: Standard OREA S603b From The 2019 Drill Campaign



Date: prepared March 23, 2021

Figure 12-15: Standard CDN-ME-1312 From the 2019 Drill Campaign

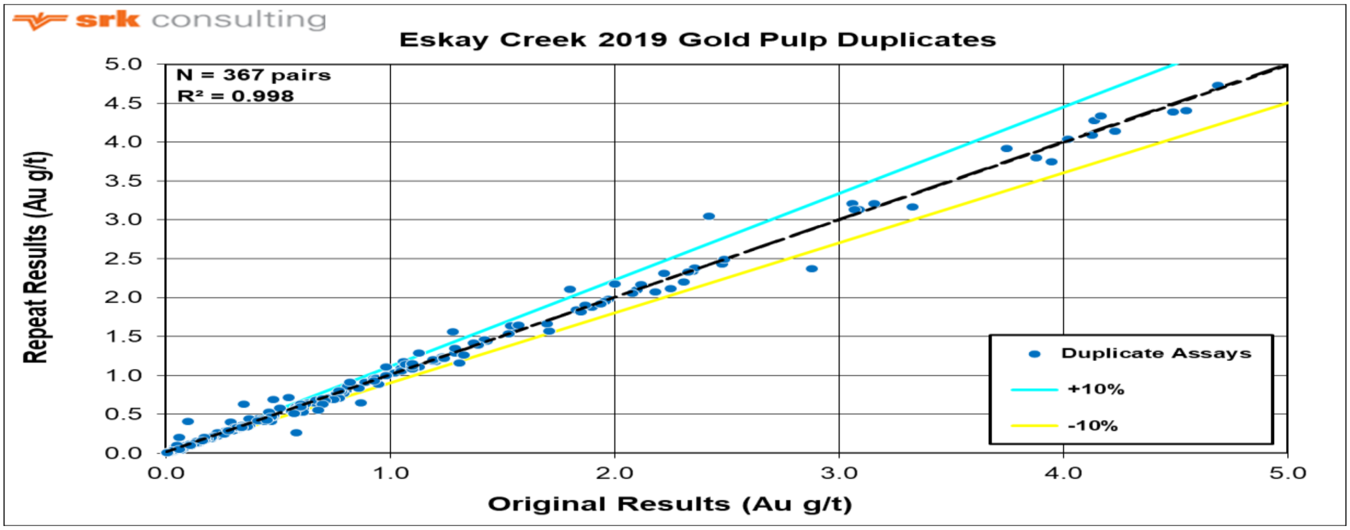


Date: prepared March 23, 2021

Paired preparation and pulp data performed in 2019 occurred within acceptable tolerance criteria at both lower-grade and higher-grade values (Figure 12-16 and Figure 12-17).

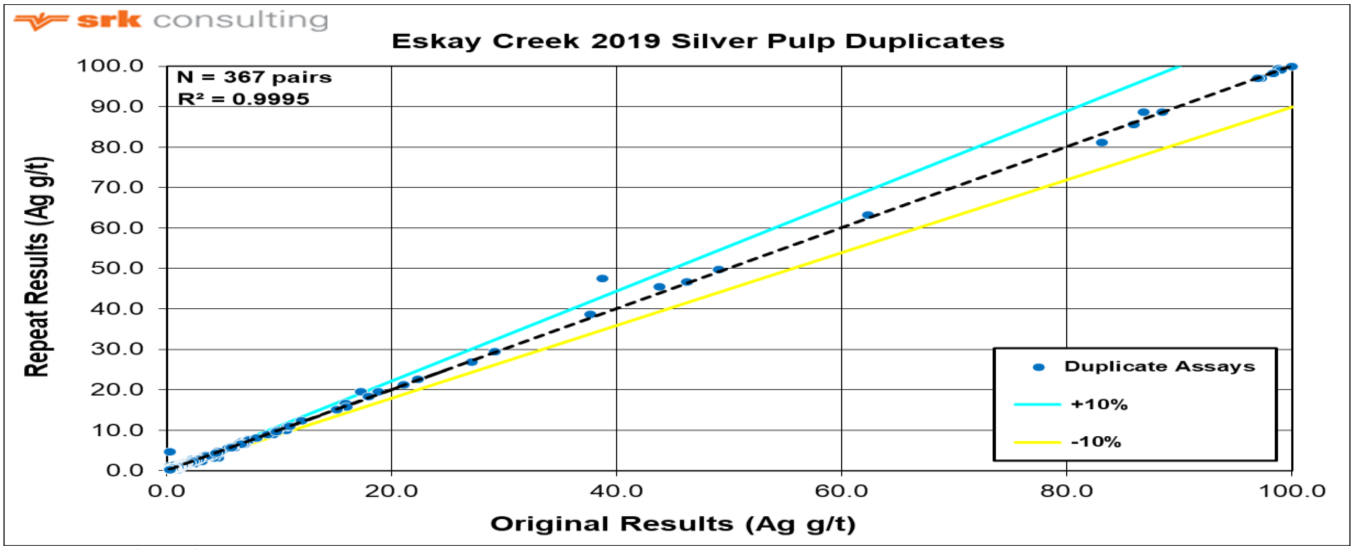
At the end of the 2019 Eskay Creek drill program, a random selection of 2.5% of all assay samples, of which 1.5% occurred within moderate to higher gold grades, were selected and sent to SGS Canada, (Skeena, 2019b). A total of 215 pulps were checked against pulps originally processed at ALS Vancouver, and 10 reference materials were sent along with the check assay samples. Overall, the check assays performed within acceptable limits.

Figure 12-16: Gold Pulp Duplicate Samples From The 2019 Drill Campaign



Date: prepared March 23, 2021

Figure 12-17: Silver Pulp Duplicate Samples From The 2019 Drill Campaign

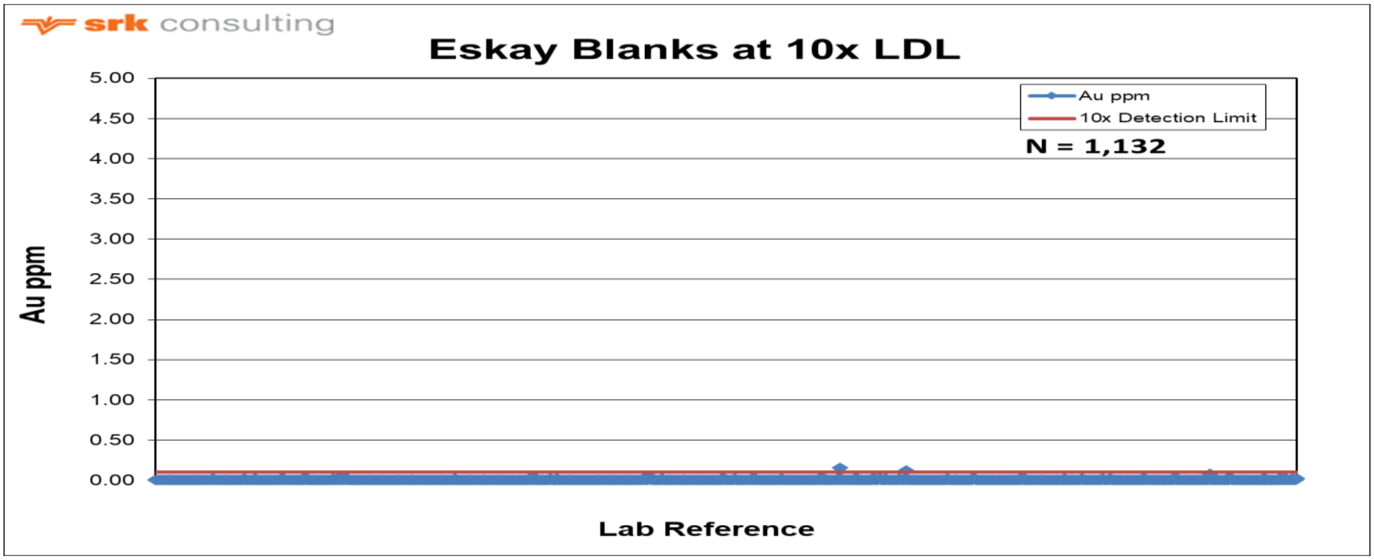


Date: prepared March 23, 2021

12.1.6.3 2020

A total of 1,132 control blanks were inserted during the two 2020 drilling campaigns. Two samples registered slightly above the 10x detection limit, however these samples occurred within a series of non-QC samples that registered below the detection limit (Figure 12-18). Having no effect on the resource estimate, they were, therefore, not retested.

Figure 12-18: 2020 Drill Campaign Blank Results

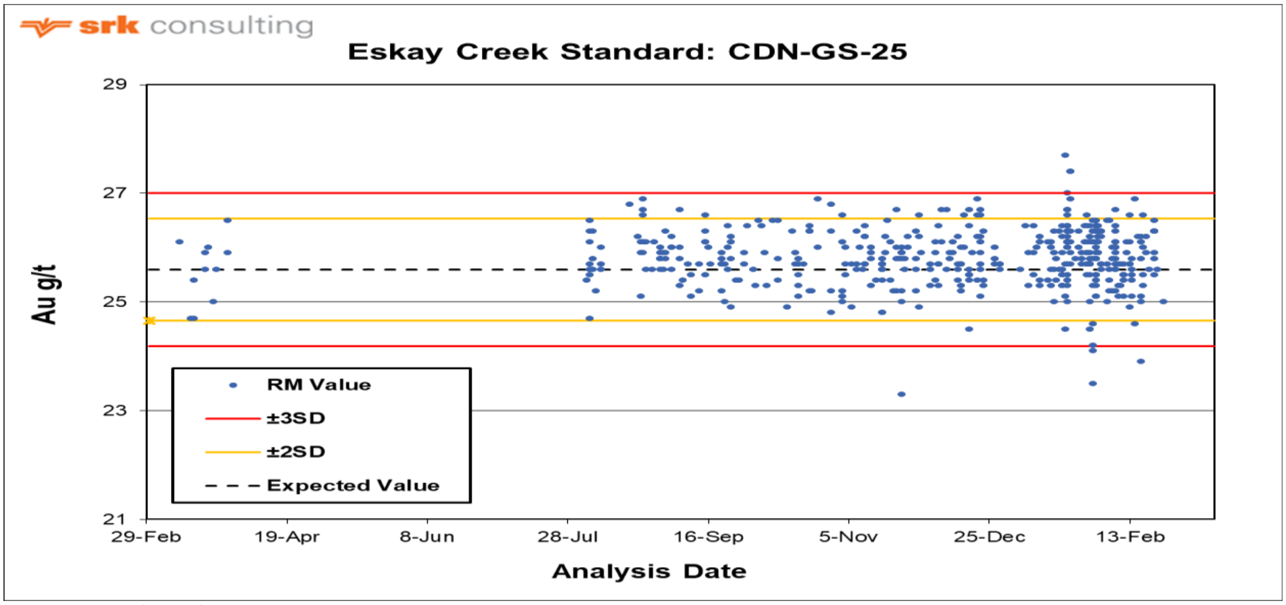


Date: prepared March 23, 2021

Five different types of CRMs were inserted into the sample stream during the 2020 drilling programs. An analysis of three CRM charts for high, medium, and low gold grades showed no obvious errors or bias (Figure 12-19, Figure 12-20, and Figure 12-21). CRM CDN-GS-25 demonstrated even spread about the expected value for gold, although four samples occurred below the three standard deviation limits (Figure 12-19).

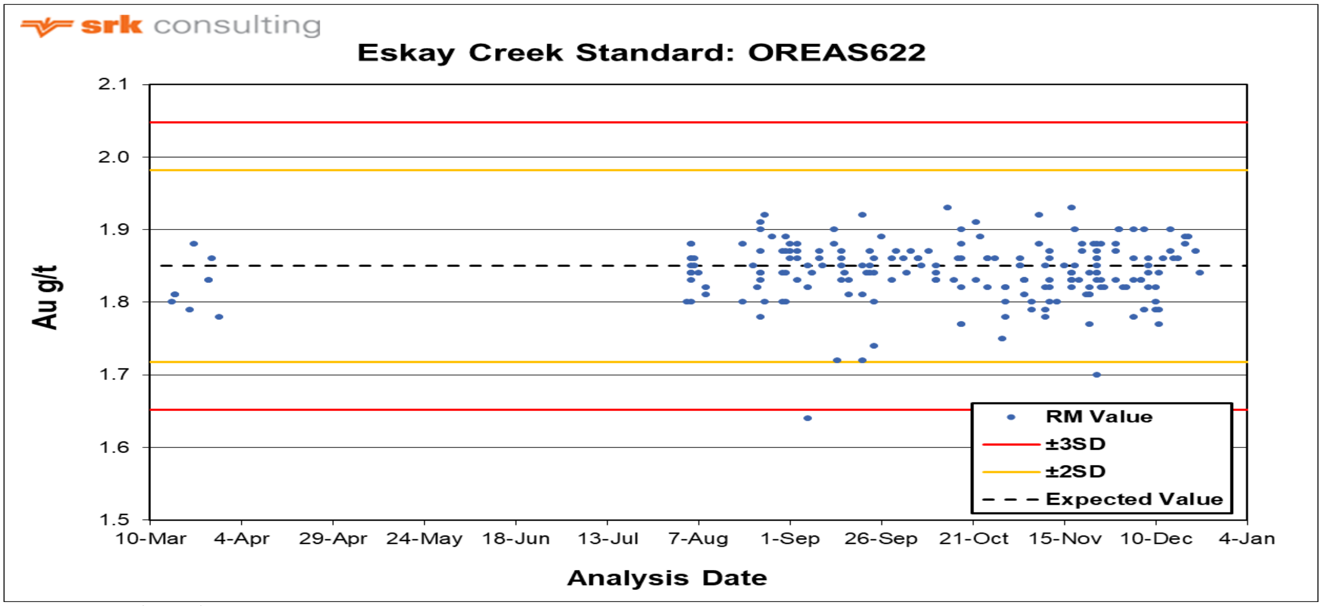
CRM OREAS622 demonstrated acceptable results for gold with all, excepting one sample, falling within the three standard deviation limits (Figure 12-20). Similarly, CRM CDN-ME-1312 results are evenly spread about the expected value and occur wholly within the three standard deviation limits (Figure 12-21).

Figure 12-19: Standard CDN-GS-25 From The 2020 Drill Campaign



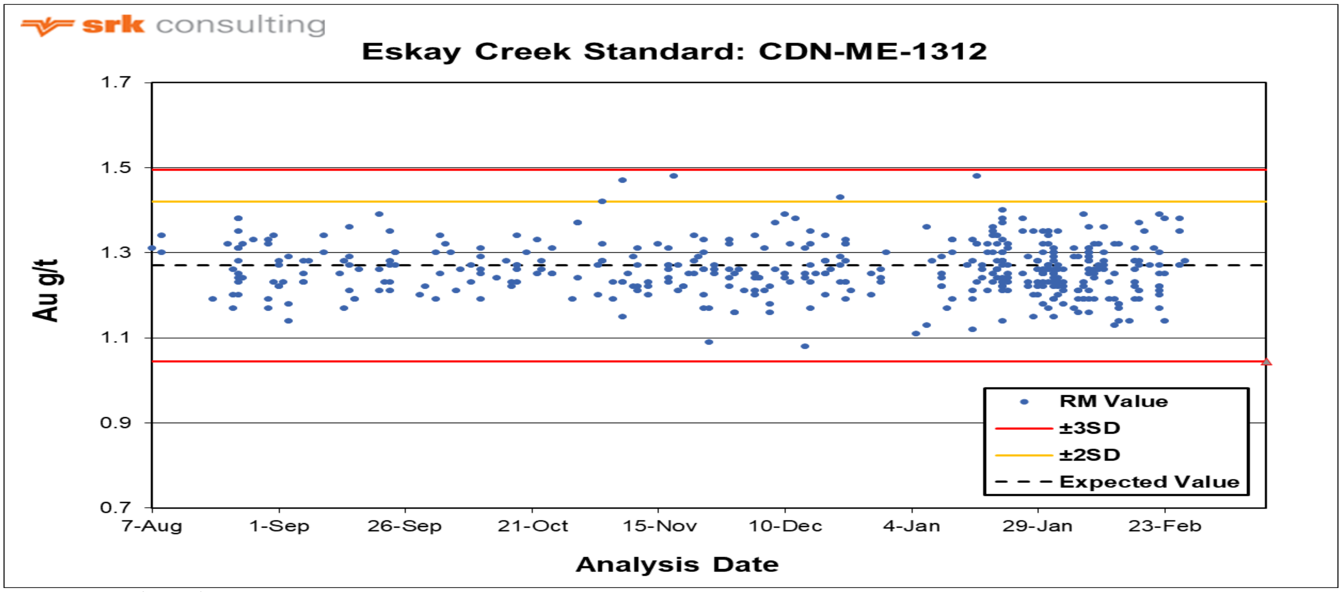
Date: prepared March 23, 2021

Figure 12-20: Standard OREA S622 From the 2020 Drill Campaign



Date: prepared March 23, 2021

Figure 12-21: Standard CDN-ME-1312 From The 2020 Drill Campaign

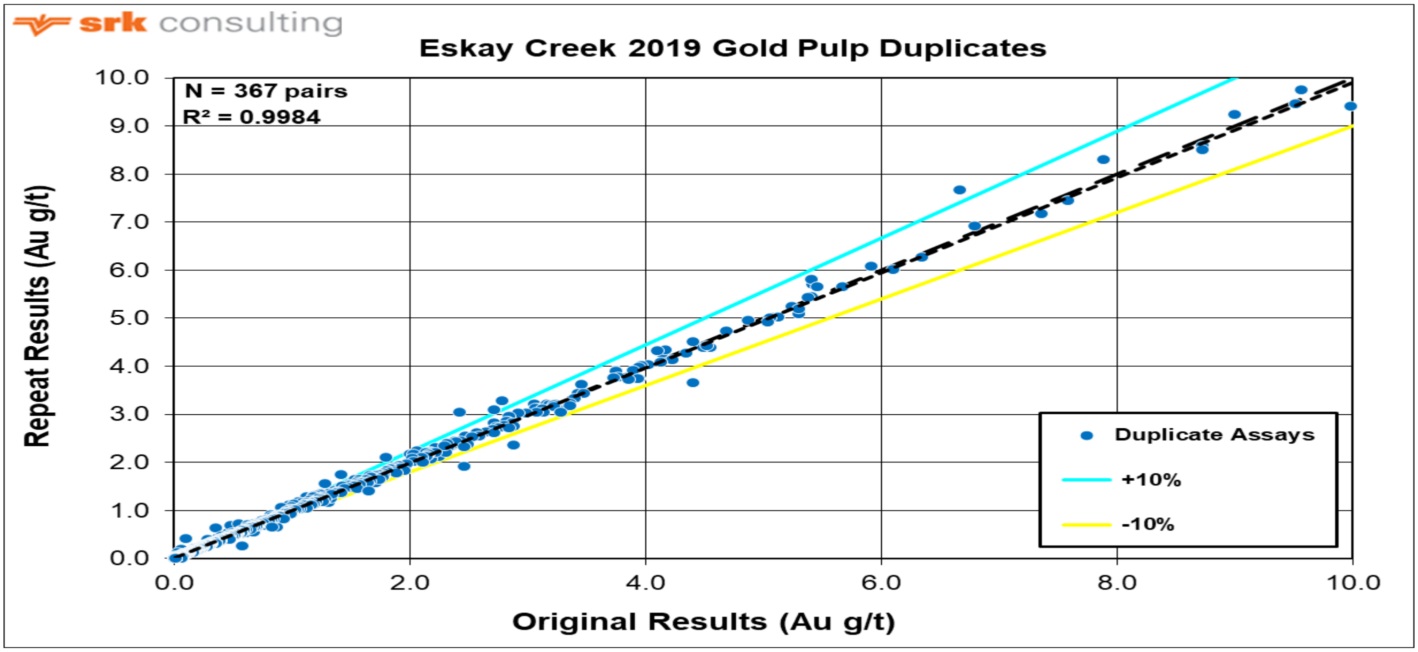


Date: prepared March 23, 2021

Paired preparation and pulp data performed during the 2020 Phase 1 and Phase 2 drilling campaigns occurred within acceptable tolerance criteria at both lower grade and higher-grade values (Figure 12-22 and Figure 12-23).

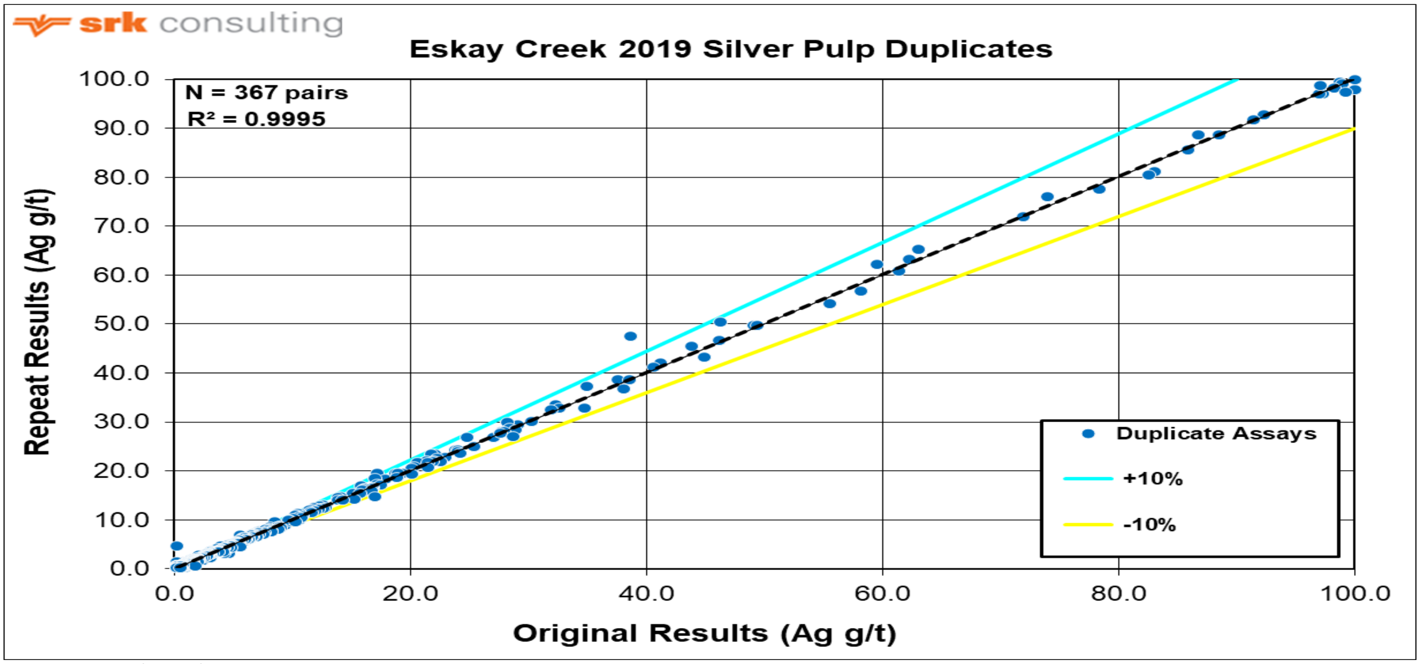
At the end of the 2020 Phase 1 drilling program, a random selection of 2.5% of all assay samples, of which 1.5% occurred within moderate to higher gold grades, were selected and sent SGS Canada (Skeena, 2020c). A total of 22 pulps were checked against pulps originally processed at ALS Vancouver, and one CRM were sent along with the check assay samples. Overall, the check assays performed within acceptable limits. Check assay results for the 2020 Phase 2 drilling program are pending.

Figure 12-22: Gold Pulp Duplicate Samples from the Combined 2020 Drilling Campaigns



Date: prepared March 23, 2021

Figure 12-23: Gold Pulp Duplicate Samples from the 2020 Combined Drilling Campaign



Date: prepared March 23, 2021

12.1.7 Summary – Verifications by SRK

The results of the QA/QC analysis indicate that the historical data are unbiased. A large number of assays in the database were validated against the original digital assay certificates. These assays ranged from the years 1999 to 2004, and less than 1% errors were found. In addition, the data analysed for the 2018, 2019, 2020 Phase 1 and Phase 2 drilling programs were collected and analysed in a systematic and unbiased manner. The data verification of this data did not identify any material issues and the QP is satisfied that the assay data are of suitable quality to be used as the basis for the resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Background

In 1991 and 1992, metallurgical testwork for the original feasibility study had defined a complex hydrometallurgical flowsheet for the recovery of gold and silver, as well as copper and zinc. This process required a large capital outlay with high unit operating costs. The original operating plan was to construct the mining infrastructure at the mine site and transport ore to a processing facility located close to Placer Dome's Equity Silver mine, near Houston, B.C.

In late 1994, mining operations commenced at Eskay Creek. In 1996, a testwork program was initiated at Process Research Associates with follow up locked-cycle testing at International Metallurgical and Environmental Inc. to evaluate the potential of a gravity/flotation process for upgrading ore from the NEX and 109 Zones into marketable concentrates.

Testwork performed by SGS Lakefield for International Corona in 1991 and 1992 examined the amenability of whole ore samples to pressure oxidation (POX) and carbon-in-leach (CIL) and cyanide leaching for the recovery of copper, zinc, gold, and silver. Results summarized in a series of reports submitted to International Corona in 1991 and 1992 covering POX operating conditions, cyanide detoxification, and environmental impact assessment of the tailings products. The final flowsheet was trialled in a series of pilot plant runs at SGS Lakefield.

The work indicated that the mineralized material could be economically upgraded to a saleable concentrate.

In 1997, Prime completed the engineering and construction of a 150 t/d mill to concentrate the gold and silver values for the NEX and 109 Zones. Over the next several years, the mill was steadily upgraded and expanded to its final production capacity of 350 t/d. Since 2008, the mine area has been under a state of reclamation, care, and maintenance.

As part of the 2019 PEA, testwork was completed by Blue Coast Research (Blue Coast) in Parksville BC, including comminution, whole ore leaching, gravity and flotation recovery methods. The process plant flowsheet assumed for the 2019 PEA included flotation recovery of a precious metal concentrate, for transport and shipment overseas. To further investigate to generate doré as a saleable product, several concentrate treatment alternatives were evaluated. Concentrate treatment is an opportunity to transform deleterious minerals into a safe and stable form and avoid high treatment charges and penalties from the sale of concentrate to a smelter or trader.

Several issues were identified during the 2019 PEA testwork program associated with high or variable content of non-sulphide gangue (NSG) minerals such as muscovite, illite, chlorite, and silica. This resulted in extended flotation times due to slow kinetics as well as poor filtration properties of some of the final concentrate samples.

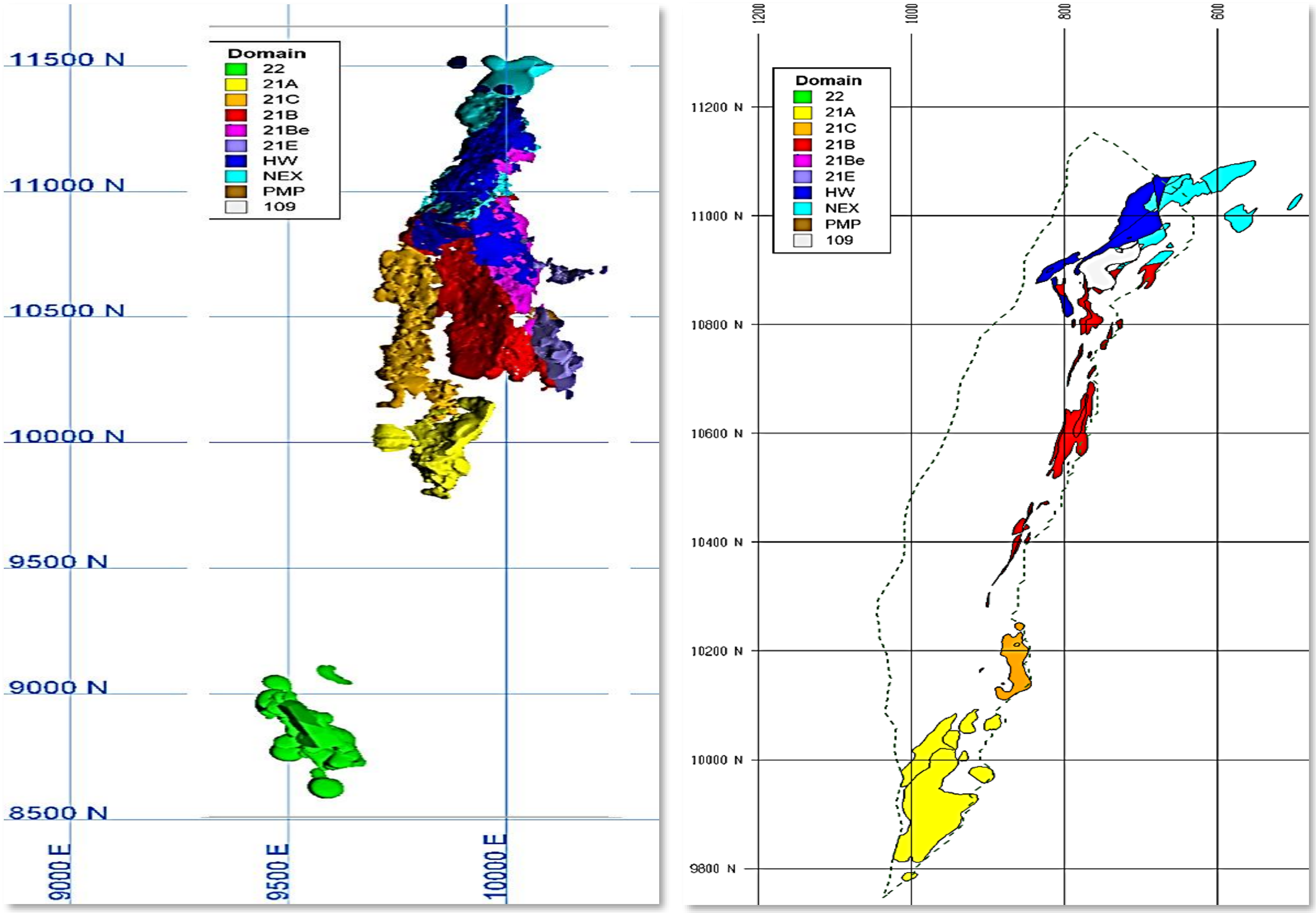
In 2020, a comprehensive testwork program was completed by Base Metallurgical Laboratories Ltd. of Kamloops, B.C. (Base Met), initially focused on issues identified in the 2019 PEA and resulted in a modified process flowsheet. The Base Met program was completed on remaining 2019 PEA test sample material as well as several new drill core samples from the 2018 to 2020 drill campaigns. More detailed mineralogical analysis revealed the effect of NSG on the flotation kinetics.

13.2 Mineralized Zones

The Eskay Creek mineralization is divided in a number of zones or domains as shown in Figure 13-1. Within each zone, the main rock types are mudstone, rhyolite, and hanging wall andesite.

Extensive underground workings are present below 1,000m RL, mainly in the 21C, HW and NEX Zones. A significant part of the open pit mining area will be in the 21A, 21B and 22 Zones over the proposed ten-year mine life. The updated resource will include more material from the NEX and 109 Zones than in the 2019 PEA.

Figure 13-1: Mineralized Zones (left: plan view, right: rotated vertical section)

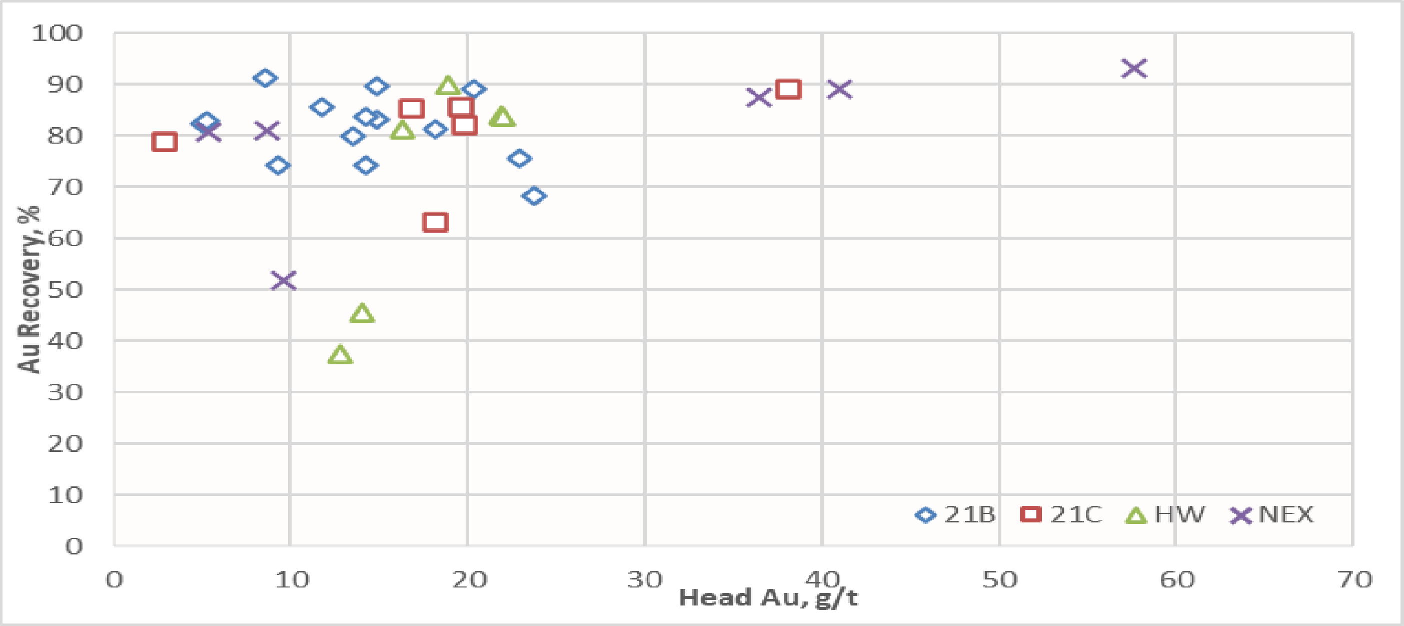


Note: Figure prepared by SRK, 2019.

13.3 Historical Testwork

A 2004 technical report was issued by Barrick on the Eskay Creek project which included testwork results on a number of samples conducted over the period 1996 to 2004 (Mahoney et al., 2004). These samples covered a range of head grades from 3 g/t to almost 58 g/t Au. Figure 13-2 shows the overall gold recovery versus head grade for the combined gravity and flotation concentrate products. The HW and NEX zones showed some lower recoveries despite the >10 g/t head grades, while the 21B and 21C zones showed consistently good performance.

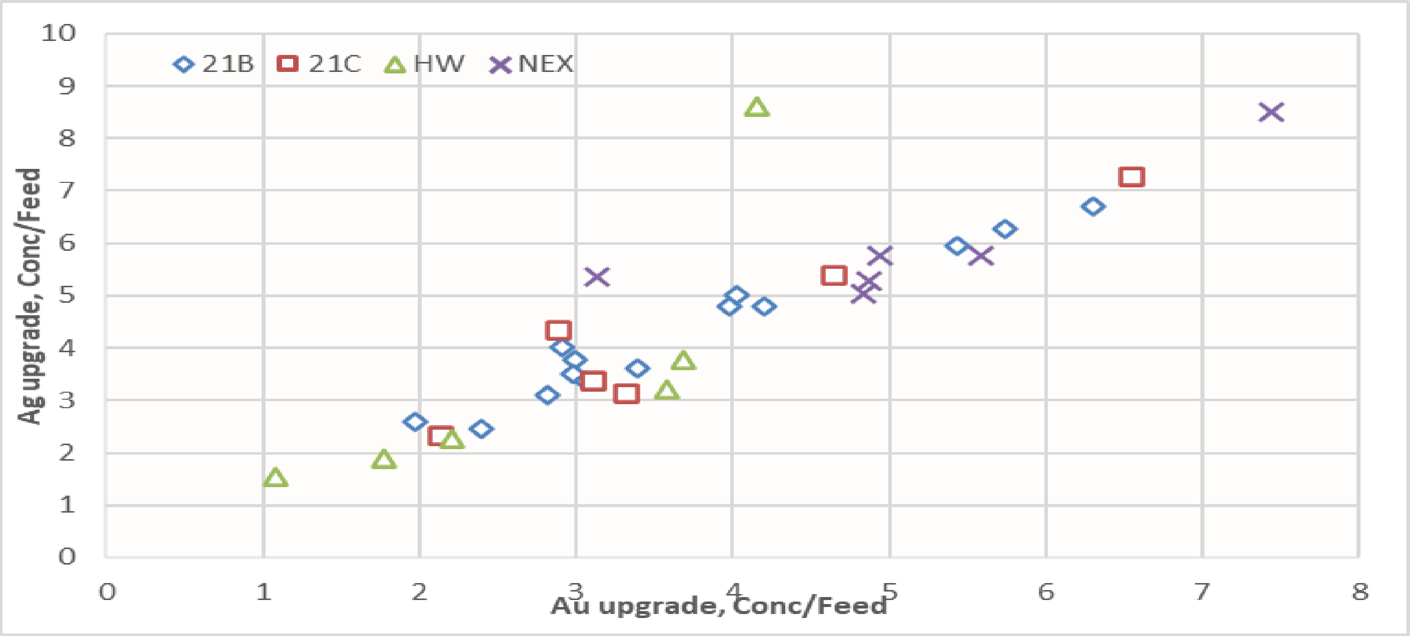
Figure 13-2: 1996 to 2004 Testwork Results by Zone – Gold Recovery vs. Head Grade



Note: Figure prepared by SRK, 2021.

Figure 13-3 shows the upgrading of silver versus gold relationship, which was evident in both the 2019 PEA and 2021 PFS testwork results, conducted on lower grade samples.

Figure 13-3: 1996 to 2004 Testwork Results by Zone – Silver vs. Gold Upgrade



Note: Figure prepared by SRK, 2021.

13.4 Blue Coast Testwork

As part of the 2019 PEA, metallurgical samples were obtained from the 2019 drilling program and submitted to Blue Coast for testing and evaluation (Blue Coast Research, 2019).

13.4.1 Sample Details

The drilling program in 2019 focused primarily on the 21A mineralized zone with auxiliary drill holes added in Zones 21C and 22. Table 13-1 summarizes the samples included in the 2019 PEA testwork program.

Table 13-1: 2019 PEA Metallurgical Sample Grades

Composite	Au (g/t)	Ag (g/t)	As (ppm)	Sb (ppm)	Hg (ppm)	S_{tot} (%)	S_{2-} (%)	C_{tot} (%)	C_{org} (%)
Hot	32.6	690	43,350	100,200	3,024	8.08	7.54	0.86	0.48
21A Low As	1.9	53	315	205	49	1.33	1.37	0.31	0.03
21A High As	8.3	54	4,005	4,240	127	2.59	2.25	0.62	0.43
21C	3.4	207	187	409	12	1.93	1.74	0.16	0.06
22 Low As	1.3	107	205	166	4	0.42	0.43	0.02	0.02
22 High As	2.8	10	1,180	330	9	0.77	0.77	0.02	0.03

The 21A and 22 Zones were divided into High and Low arsenic samples, with the samples covering a range in grades from 1.3 to 32.6 g/t Au and 10 to 690 g/t Ag. The "Hot" mudstone sample was extremely high in gold, arsenic, antimony, and mercury with very high levels of total sulphur, sulphide (S^{-2}) and organic carbon.

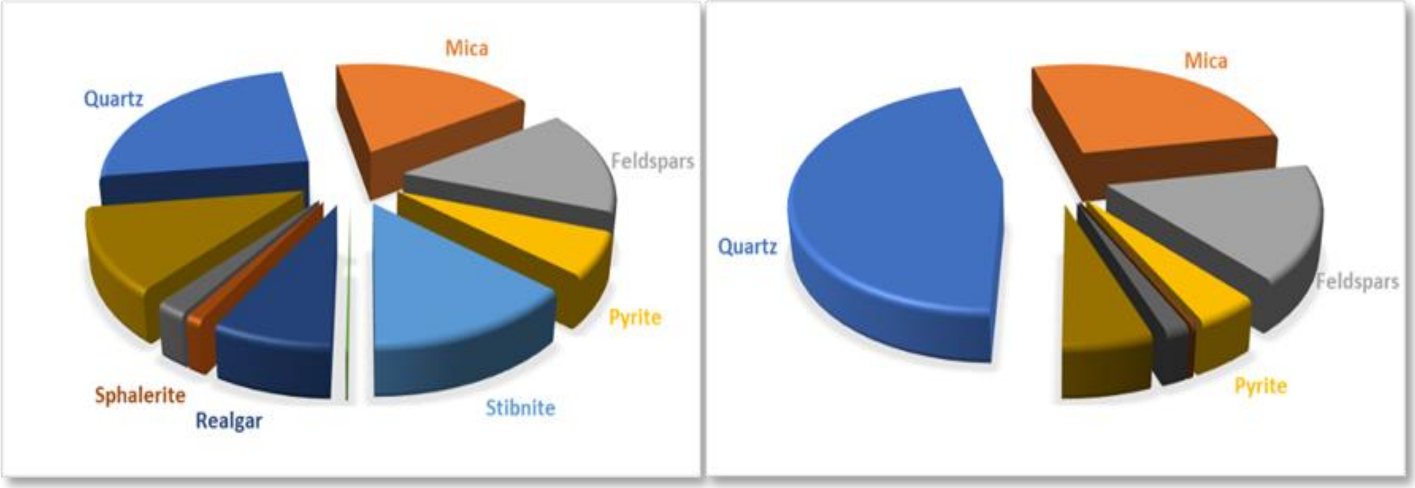
Two composite samples were generated to estimate the expected gold grade for the first three years and the life-of-mine (LOM). The composites were a blend of Hot (mudstone) and 21A Low As (rhyolite) samples:

- LOM sample: 91% 21A Low As + 9% Hot;
- Y1–3 sample: 83% 21A Low As + 17% Hot.

The consequence of blending with the Hot sample was elevated arsenic, antimony, and mercury levels for the two composites, which resulted in flotation concentrates being produced with exaggerated impurity levels. Separate testing on lower-grade samples produced concentrates with lower impurities so performance estimates provided in the 2019 PEA were not biased by the Hot sample blending.

Figure 13-4 shows the modal mineralogy of the mudstone and rhyolite material with the principal minerals labelled in the breakdown. The mudstone has appreciable amounts of sphalerite, realgar, and stibnite while the rhyolite sample had almost none detected using automated mineralogy. In contrast, the rhyolite sample had very high levels of quartz, mica, and feldspars.

Figure 13-4: Model Mineralogy of Mudstone vs. Rhyolite (Hot & 21A Low As samples)



Note: Figure prepared by SRK, 2019

13.4.2 Comminution

Comminution or hardness testing on each sample consisted of semi-autogenous grind (SAG) mill comminution (DWi), Bond rod mill work index (RWi) and Bond ball mill work index (BWi) tests at a closing screen size of 150 µm.

The test results indicated a range of material hardness with mudstone being moderately soft (DWi of 3.2 kWh/m^3, BWi of 13.0 kWh/t), while the 22 Zone exhibited BWi values of as much as 26.4 kWh/t. The hardness values for each sample and test type are summarized in Table 13-2.

Table 13-2: Summary of 2019 PEA Comminution Testwork

Composite	Particle SG	DWi (kWh/m^3)	RWi (kWh/t)	BWi (kWh/t)
Hot	3.06	3.18	N/A	13.0
21A Low As	2.69	4.84	14.0	16.1
21A High As	2.69	4.73	15.2	16.2
21C	3.00	5.80	16.4	16.6
22 High As	2.62	7.34	21.0	23.5
22 Low As	2.59	5.91	21.8	26.4

Note: BWi measured at a closing screen size of 150 µm.

Previous testwork by SGS Lakefield on samples from the 21B, 21C, HW, 109 and NEX Zones reported SAG power index (SPI) values between 49 and 171 minutes with BWi results of 17 to 20 kWh/t, to an unreported closing screen size.

13.4.3 Gravity Recovery

The LOM composite sample (91% 21A Low As + 9% Hot) was tested using an extended gravity recoverable gold (E-GRG) procedure with a three-pass grind and recovery sequence. A total of 45% of the gold was recovered to an 80% passing (P_{80}) grind size of 82 µm with only 13% of the silver recovered. More importantly, the combined gravity + flotation recovery was not higher than flotation alone. Based on this test result, gravity recovery was not recommended in the process flowsheet as part of the 2019 PEA. In addition, it was believed that blending in the Hot sample produced a much higher E-GRG value than would be expected for lower-grade and/or rhyolite material.

13.4.4 Bulk Flotation

A considerable number of open-circuit, rougher and rougher/cleaner float tests were conducted on all samples included in Table 13-3. The 21A Low As sample was initially tested under a wide range of conditions and later applied to the other samples, as part of variability testing of the 21A, 21C and 22 Zones.

The testwork objective was maintaining high precious metal recoveries at a lower mass pull to concentrate, which was evident in early testing as well the historical work done by SGS in 1991.

A range of primary P_{80} grind sizes were tested (from 338 µm down to 39 µm) with ~60 µm used as the target P_{80} grind size for further float work. Rougher concentrate was also reground prior to cleaning, with a target P_{80} size of ~25 µm used as the base case.

It was noted that the grind and regrind times were quite long (up to 40 minutes being required for the 25 µm regrind size); however, an investigation into possible overgrinding of phyllosilicate minerals did not reveal anything significant. Blue Coast noted that the flotation concentrate was (at times) very slow to pressure filter and was a concern to be investigated in the PFS testwork program.

The use of dispersants (sodium silicate and carboxymethyl cellulose, or CMC) was investigated as well as collector dosage. Much lower mass pulls were obtained without affecting recovery using stainless steel grinding liners/media and lower pulp densities (with larger volume laboratory float cells). The mechanism of how the stainless steel liners/media affected mineral surfaces remained unclear.

Samples exhibited relatively slow float kinetics with 80% Au recovery after 20 minutes of rougher flotation and 90% recovery after 40 minutes. An investigation into possible sliming did not reveal any explanation for the slow-floating nature of the samples.

Up to three stages of cleaning were done, with concentrates generated after 25 min, 15 min, and 10 min of float time. Copper sulphate was added at 100 g/t to the primary grind as an activator with potassium amyl xanthate (PAX) used throughout as the collector, with a total of 200 g/t added.

The base case flowsheet used to evaluate the range of samples is shown in Figure 13-5.

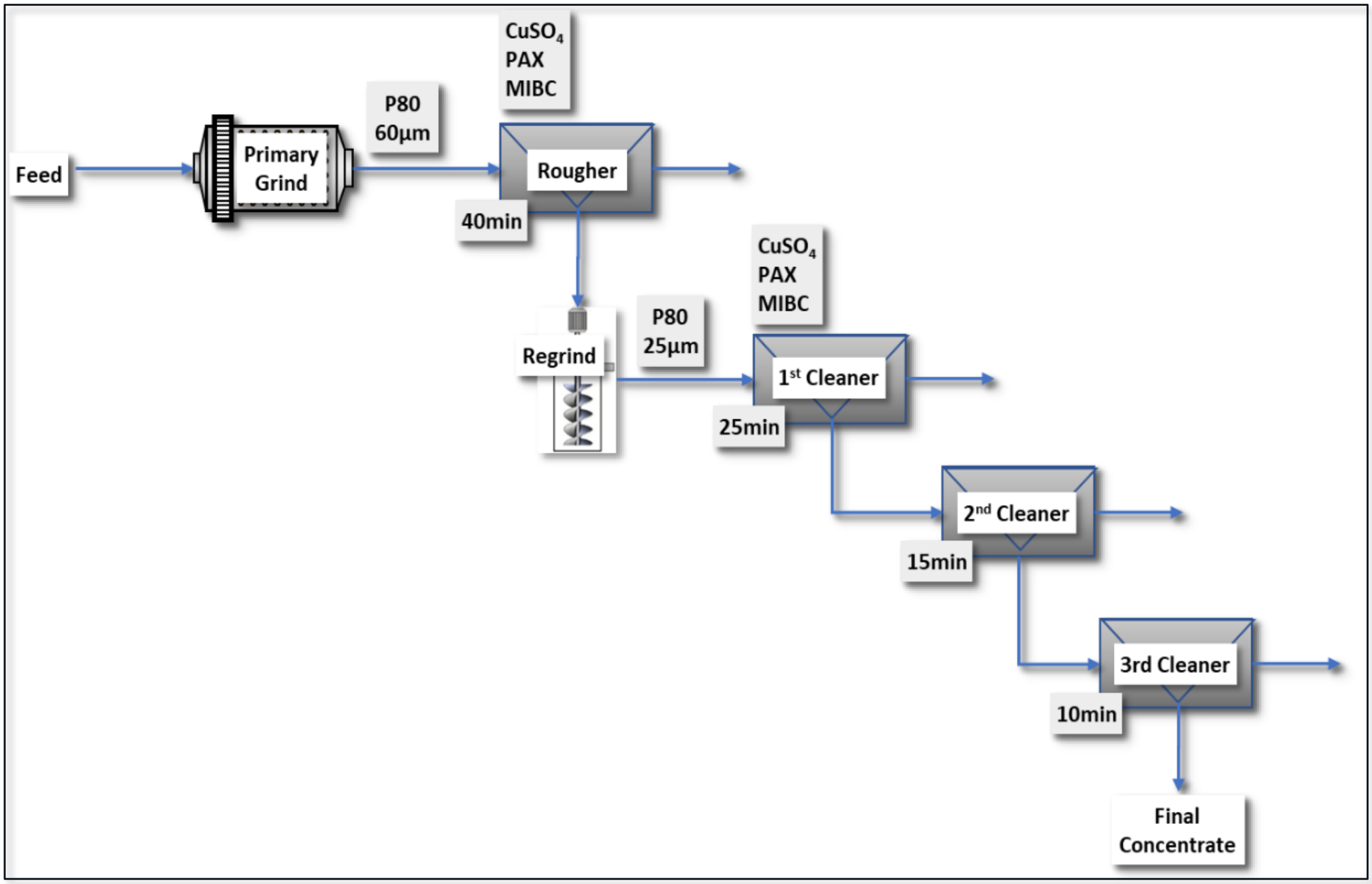
Overall, flotation testwork was able to produce a bulk concentrate with gold recoveries of 80 to 95% at grades of 40 to 50 g/t Au. Silver recoveries were in the range of 84 to 97% with grades from 1,000 to 1,300 g/t Ag.

The base case float conditions were tested on lower-grade samples (21A Low As, 21C, 22 Low As, 22 High As) to generate a metal recovery versus head grade relationship. The results showed a consistent behaviour across all samples, reflecting the relatively low amount of sulphides being recovered from rougher and cleaner flotation.

The range of final concentrate precious metal grades and impurity levels are shown in Table 13-3, sorted in order of gold head grade. As will be discussed in Section 13.5.14, the sulphide minerals containing arsenic, antimony, and mercury closely follow the gold-silver bearing minerals to final concentrate.

For the <3.5 g/t Au head grade samples, the final concentrate contained around 1% As and Sb with ~200 ppm Hg. The LOM composite generated concentrates with much higher impurity levels due to the blend of Hot sample. As the expected mine plan calls for material at 4 g/t Au and below, the lower-grade sample results were used to generate the forecast concentrate quality and quantity for the 2019 PEA.

Figure 13-5: 2019 PEA Open Circuit Float Test Flowsheet



Note: Figure prepared by SRK, 2019.

13.4.5 Concentrate & Tailings Mineralogy

Automated mineralogical analysis was performed on both the final concentrate and tailings from the LOM sample float testing.

Table 13-3 summarizes the main minerals in the two streams.

The LOM sample concentrate contained 19% pyrite with ~7% stibnite and realgar, together with 25% silica and 35% phyllosilicate minerals. In contrast, the tailings contained minimal sulphides (after an extended rougher flotation period) with 54% silica and 31% phyllosilicate minerals.

Table 13-3: Concentrate Grades for 2019 PEA Samples (sorted by gold head grade)

Sample	Head		Final Concentrate				
	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	As (%)	Sb (%)	Hg (ppm)
22 Low As	1.3	107	53.8	7,565	1.0	1.2	239
	1.3	107	49.1	5,395	0.7	0.7	138
21A Low As	1.9	53	40.2	1,208	0.6	0.7	1,091
22 High As	2.8	10	48.8	219	3.5	0.9	
	2.8	10	38.1	154	2.1	0.6	140
21C	3.4	207	52.5	4,150	0.3	1.2	230
	3.4	207	55.9	4,779	0.3	1.2	249
LOM Comp	3.9	96	40.6	1,036	3.5	9.2	
	3.9	96	52.3	1,115	4.8	9.8	3,817
	3.9	96	41.3	1,042	4.2	9.1	3,501
Yr 1-3 Comp	7.7	164	54.8	1,182	5.4	12.3	
	7.7	164	50.0	1,244	5.9	12.3	4,464
21A High As	8.3	54	51.3	382	2.9	3.5	

Table 13-4: LOM Composite Flotation Product Modal Mineralogy (2019 PEA update)

Mineral	Concentrate	Tailings
Pyrite	19	0.5
Chalcopyrite	0.2	–
Sphalerite	1.1	–
Stibnite	7.5	0.1
Realgar	6.5	–
Quartz	25	54
Phyllosilicates	35	31
Calcite	2.1	2.6
Barite	0.1	0.26
Other	3.3	11.6

13.4.6 2019 PEA Recovery Estimates

Based on the 2019 testwork results from samples with a range of head grades, a flotation concentrate of saleable precious metal content was produced at high recoveries of both gold and silver. This concentrate contained impurities of arsenic, antimony and mercury that would be subject to penalties. Depending on the concentrate customer, the antimony content may be included as a payable metal, provided the level is above threshold (e.g., 3% Sb).

The open-circuit rougher and cleaner float test results were used to generate relationships between the gold and silver recovery versus head grade as well as the expected mass pull to concentrate. The concentrate impurity levels were well established from the testwork results. These relationships were done for 50 g/t, 40 g/t and 25 g/t Au concentrate to assist the marketing review completed as part of the 2019 PEA. Achieving a lower-grade concentrate required few stages of cleaner flotation.

13.5 Base Met Testwork

For the 2021 PFS, additional testwork was conducted by Base Met, with all results summarized in two reports (Base Met, 2021a, Base Met, 2021b).

The objectives of this program were to better understand the issues raised in the 2019 PEA testwork along with improving confidence in the process plant performance over a wider range of sample grades and mineral compositions from zones other than 21A, 21C and 22). As a consequence of this early-stage testwork, the 2019 PEA process flowsheet was modified to isolate the variable amount of soft, phyllosilicate minerals found predominantly in the rhyolite material.

Table 13-5 summarizes the list of separation tests conducted by Base Met (ref BL594 Base Met, 2021a) to assist with tables and figures shown in this sub-section that reference test numbers. Table 13-6 summarizes the tests done to investigate rejecting NSG minerals from the final concentrate (ref BL777 Base Met, 2021b).

Table 13-5: List of Separation Tests Conducted (ref Base Met BL594)

Test #	Sample ID	Type
1 to 3	21A Low As	Rougher
4	21A Low As	Gravity
5 to 17	21A Low As	Rougher
18 to 20	Annual Comps	Rougher
21 to 22	Rougher Tails	Diagnostic Leach
23 to 28	Annual Comps	Rougher
29 to 36	21A Low As	Rougher
37	21A High As	Cleaner
38 to 58	21A Low As	Cleaner
59 to 62	Annual Comps	Gravity/Cleaner
63 to 65	Annual Comps	Locked Cycle
66 to 68	Annual Comps	Cleaner
69	New 21A	E-GRG
70 to 86	Variability Comps	Cleaner

Test #	Sample ID	Type
87	New 21A	Cleaner
88	21A 2020	Cleaner
89-92	Annual Comps	Cleaner

Table 13-6: List of Additional Depressant Tests Conducted (ref Base Met BL777)

Test #	Sample ID	Type
1 to 9	Variability Comps	Cleaner

Testwork included: open circuit rougher and rougher/cleaner float tests, locked cycle float tests, diagnostic leach, and E-GRG as well as gravity recovery followed by cleaner flotation.

Tests 1 to 17 were conducted on the main 2019 PEA test sample (21A Low As) to reproduce the results reported by Blue Coast as well as investigate alternate float circuit configurations.

Tests 18 to 20, 23 to 28 and 59 to 62, 66 to 68 were performed on Annual composite samples for Y1, Y2 and Y3. Locked cycle tests on these three composites were done in tests 63 to 65.

Tests 70 to 86, 89 to 92 were done on the original drill core samples referred to as Variability composites.

Test 88 was a 'bulk test' to generate larger masses of concentrate and tailings for physical testing. This test was done on a separate sample referred to as "21A 2020".

As the testwork program progressed, the primary grind P_{80} size started at 60 µm (following the 2019 PEA) and coarsened to 250 µm for several tests including the locked cycle work. Following a process review, the primary grind was reduced to avoid high reduction ratios across the regrind mills and the final flowsheet primary P_{80} size was set to 100 µm.

13.5.1 Sampling Origin & Composition

Head grades for all tested samples are included in Table 13-7, Table 13-8, and Table 13-9. The main 2019 PEA test sample (21A Low As) was investigated further by Base Met to refine the flotation conditions and address the slow flotation kinetics observed by Blue Coast in the 2019 PEA update.

Three Annual composites were prepared from Variability composites, or fresh half core samples from 2018 to 2020 drilling. These Annual composites were blends of the different mineralized zones to match the first three years of the 2019 PEA mine plan. Variability composites were also blended to match the expected gold, arsenic, antimony, and mercury head grades within the mine plan.

Table 13-7: 21A Low As Composite Assays

Composite	Cu (%)	Fe (%)	Ag (g/t)	Au (g/t)	As (g/t)	Sb (g/t)	S (%)	S2- (%)
21A Low As	0.01	1.3	53	1.82	344	227	1.42	1.35

Table 13-8: Annual Composite Assays

Composite	Cu (%)	Fe (%)	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)	SO4 (%)	S2- (%)
Year 1	0.04	1.7	3.68	170	6361	2064	2.99	0.94	2.05
Year 2	0.03	1.6	2.15	127	4348	1481	1.97	0.21	1.76
Year 3	0.02	1.7	2.01	99	1979	699	1.92	0.25	1.67

A list of the Variability composite grades is shown in Table 13-9 along with another 2019 PEA test sample (22A High As) and "New 21A" sample used to replace 21A Low As when it became depleted. Except for the two 2019 PEA samples, all Variability composites were from recent drilling by Skeena.

Head grades for the Variability samples ranged from 0.63–5.7g/t Au, 4–512 g/t Au and a wide range of arsenic and antimony values. Sulphur grades ranged from 0.35– 4.2%.

Table 13-9: Variability Composite Assays

Composite	Cu (%)	Fe (%)	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)
VC1	0.090	2.1	5.67	411	31730	6604	3.92
VC2	0.014	0.8	1.30	20	299	120	0.90
VC3	0.017	1.1	4.76	89	583	1551	3.91
VC4 (21A Low As)	0.011	1.3	1.82	53	344	227	1.42
VC5	0.009	3.6	1.93	41	238	708	2.37
VC6	0.013	4.2	1.89	128	564	663	2.36
VC7	0.103	0.9	1.50	328	201	1312	1.56
VC8	0.008	4.1	2.50	19	353	233	3.10
VC9	0.033	4.2	1.55	151	183	592	3.33
VC10	0.013	2.3	2.22	49	165	218	1.97
VC11	0.016	0.9	2.66	7	188	41	1.22
VC12	0.008	1.0	1.52	4	151	21	1.37
VC13	0.077	2.0	3.21	76	270	373	4.19
VC14	0.008	6.4	1.11	28	781	122	4.14
VC15	0.040	7.7	1.12	43	408	325	2.62
VC16	0.012	1.5	1.18	182	95	267	0.60

Composite	Cu (%)	Fe (%)	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)
VC17	0.002	1.0	0.63	41	102	119	0.35
VC18	0.005	1.2	1.02	76	195	130	0.42
22A High As	-	-	2.80	10	1180	330	0.77
New 21A	0.078	1.6	5.35	512	716	1738	2.14

A final composite labelled “21A 2020” was prepared when an additional 100 kg was required for bulk float testing. This composite was 47% from the “New 21A” sample shown in the last row in Table 13-9 along with a combination of the three annual composites. The 21A 2020” sample for bulk testing was 4.3 g/t Au with 395 g/t Ag, 2,321 ppm As, 1,397 ppm Sb and 2.12% S.

13.5.2 Mineralogy

Quantitative mineralogy was obtained on the 21A Low As and three Annual composites using quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), with the modal compositions shown in Table 13-10. All four samples reported <3% pyrite with 36% to 47% quartz and varying amounts of NSG minerals (muscovite/illite, chlorite, biotite/phlogopite).

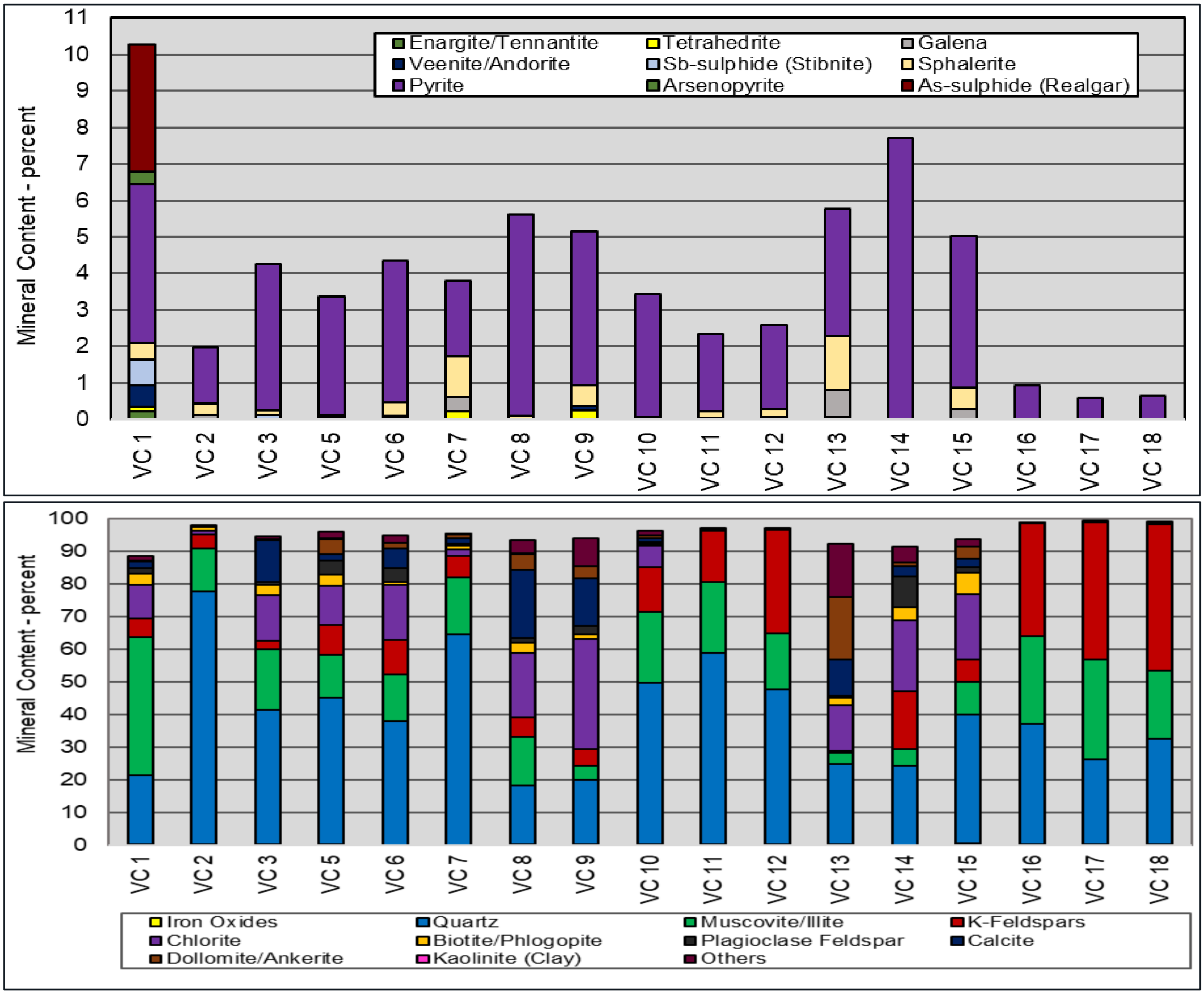
Table 13-10: Main Composite Modal Mineralogy

Mineral	Mineral Compositions (Weight %)			
	21A Low As	Year 1	Year 2	Year 3
Tetrahedrite	<0.1	0.1	0.1	<0.1
Galena	0.1	0.1	0.1	0.1
Veenite/Andorite	<0.1	0.1	0.1	0.1
Sb-sulphide (Stibnite)	<0.1	0.2	0.1	0.1
Sphalerite	0.1	0.3	0.4	0.2
Pyrite	2.3	2.9	2.7	2.6
Arsenopyrite	<0.1	<0.1	0.1	<0.1
As-sulphide (Realgar)	0.0	0.8	0.3	0.2
Iron Oxides	0.5	0.4	0.5	0.4
Quartz	46.9	35.7	37.1	36.2
Muscovite/Illite	36.2	26.4	26.2	22.8
K-Feldspars	4.5	8.3	15.0	17.8

Mineral	Mineral Compositions (Weight %)			
	21A Low As	Year 1	Year 2	Year 3
Chlorite	4.7	11.8	7.3	7.4
Biotite/Phlogopite	0.9	3.3	1.9	2.3
Plagioclase Feldspar	0.5	1.6	1.2	1.1
Calcite	1.7	5.5	3.4	5.1
Others	1.4	2.5	3.4	3.7
Total	100	100	100	100

The mineralogy results for the Variability composites are shown graphically in Figure 13-6. The upper chart shows sulphide minerals while the lower chart summarizes non-sulphides. Pyrite content varied more in these samples compared with the Annual composites, with up to 5.5% reported for VC8 (from the HW zone). VC1 from the 21A zone also reported 3.5% realgar, indicated by the 3% As head assay while the other samples showed very low amounts of arsenical sulphides. Non-sulphide mineralogy varied widely across the Variability composites with significant chlorite, calcite and dolomite occurring in some samples.

Figure 13-6: **Variability Composite Modal Mineralogy (upper: Sulphides; lower: Non-sulphides)**



Note: Figure prepared by Base Met, 2021.

13.5.3 Diagnostic Leach

As an investigation of gold occurrence, diagnostic leach tests were performed on rougher/scavenger float tailings samples from Tests 13 and 14, conducted on sample 21A Low As (Table 13-11). Rougher tailings samples were reground to 42 µm (T13) and 15 µm (T14) to compare performance of the scavenger float circuit.

Diagnostic leach results on the scavenger tailings indicate that, at the coarser regrind size, the majority of gold present in the sample was associated with pyrite and arsenic-bearing sulphide minerals. At the finer grind size, gold losses were associated with arsenic sulphides.

Table 13-11: **21A Low As Tailings Diagnostic Leach Test Results**

Stage	Au g/t - per stage		Au Distribution - percent	
	T13 Ro/Sc Tails	T14 Ro/Sc Tails	T13 Ro/Sc Tails	T14 Ro/Sc Tails
Cyanide-leachable gold	0.33	0.14	19.8	9.8
Carbonate locked gold	0.07	0.06	4.1	4.4
Arsenical mineral (arsenopyrite)	0.39	1.09	23.5	77.7
Pyritic sulphide mineral	0.82	0.07	49.4	4.8
Silicate (gangue) encapsulated	0.05	0.05	3.2	3.3
Total (recalculated) Au grade	1.66	1.40	100	100

For this 21A Low As sample, minimal gold losses were associated with silicates or carbonate-locked particles.

13.5.4 Comminution

Comminution testwork was performed on blended composites, representing the main mineralized zones (Table 13-12). HW as well as 21A material was considerably softer in terms of impact breakage (A*b value) while the 22 Zone sample was again highest in BWi at a closing screen size of 106 µm. The 21A sample was Bond ball mill work index tested at a number of closing screens to observe the impact of grind size on grindability.

Table 13-12: **Comminution Test Summary on Zone Composites**

Zone	SMC (A*b)	Bond Ball Mill Work Index				
		CSS (µm)	F_{80} (µm)	P_{80} (µm)	Gpr	BWi (kWh/t)
21A	-	75	1906	53	0.9	17.3
	-	106	1906	77	1.2	16.0
	62.9	150	1906	125	1.6	15.7
21E	39.4	106	2003	78	1.0	18.5
HW	73.4	106	1821	77	1.4	14.0
21B	32.0	106	2015	77	1.0	17.7
21C	47.9	106	2046	78	1.0	18.7
22	32.0	106	2276	81	0.6	27.6

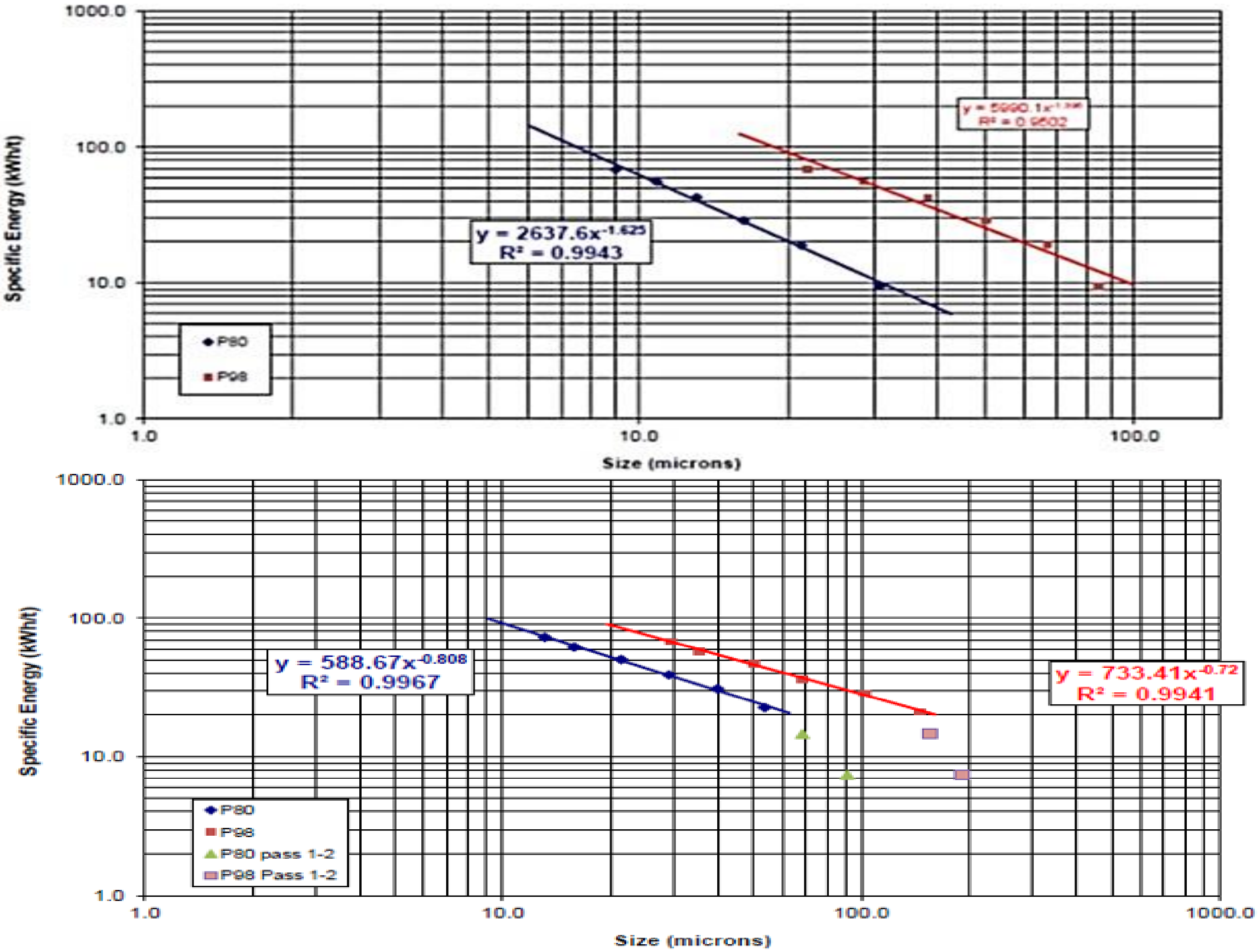
Abrasion index testing was done on the three Annual composites, with the results in Table 13-13 showing a limited range and moderate abrasivity.

Table 13-13: **Abrasion Index Test Results on Annual Composites**

Annual Composite	Abrasion Index (g)
Year 1	0.183
Year 2	0.262
Year 3	0.211

For regrind mill power requirements, the bulk float test on sample “21A 2020” generated sufficient mass of regrind circuit feed samples for IsaMill signature plot testing. In the modified flowsheet for the PFS, two streams require regrinding: rougher concentrate and rougher tailing after desliming. These two samples were signature plot tested as shown in Figure 13-7 with a target primary grind P_{80} size of 100 µm.

Figure 13-7: **IsaMill Signature Plots (Upper: Rougher Concentrate; Lower: Rougher Tail after Deslime)**



Note: Figure prepared by ALS Metallurgy for Base Met, 2021.

The ALS laboratory report results for this work (Base Met, 2021a) included some commentary on the sample characteristics. The rougher concentrate sample had a feed P_{80} size of 66 µm (measured by sieve and cyclosizing methods) but P_{80} size of 85 µm from laser sizing. The signature plot test was conducted using six passes; however, the target product size was achieved after pass three. A low pulp density was required through the test to maintain an acceptable pulp viscosity as measured with a Marsh funnel. The final pulp density was about 27% solids and averaged about 31% solids through the test. A specific energy requirement of 32.4 kWh/t was required to grind the material to 80% passing 15 µm under these conditions.

For the rougher tailings sample, sieve analysis reported a feed P_{80} size of 122 µm while laser sizing reported a P_{80} size of 145 µm. The Signature plot test was conducted using eight passes to achieve the target product size. Dilution water was required through the test to maintain an acceptable pulp viscosity as measured with a Marsh funnel. The final pulp density was about 38% solids and averaged about 42% solids through the test. A specific energy requirement of 66 kWh/t was required to grind the material to 80% passing 15 µm under these conditions.

13.5.5 Gravity Recovery

Following the work done for the PEA, a fewer gravity concentration tests were conducted to observe the effect of primary grind size on the gravity recoverable portion of gold in Eskay Creek samples. It was well recognized that the gold occurred principally as fine particles, with fine regrind sizes required to achieve good flotation concentrate grades.

As the precious metal flotation concentrate is subject to penalties for As, Sb, and Hg, the net gold payability is considerably less than if produced as doré from a gravity concentrate. This was the motivation for continuing to look at options for gravity recovery. As shown in Table 13-14, the 21A Low As sample was evaluated at the 60 µm P_{80} grind size and recovered only 5% of the gold to a 0.2% mass after panning a Knelson concentrate.

Table 13-14: 21A Low As Composite Gravity Recovery (P_{80} of 60 µm)

Product	Mass %	Assay (g/t)					Distribution (%)				
		Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
Pan Concentrate	0.2	44.1	1820	5258	7399	33.9	5.2	7.3	3.3	7.1	5.3
Pan Tail	2.6	6.99	340	857	1335	5.17	9.7	16.3	6.3	15.1	9.6
Knelson Tail	97.2	1.62	42	322	181	1.21	85.2	76.4	90.4	77.8	85.1

The three Annual composites were also tested, but the target primary grind size had shifted to 250 µm at this time, with the results shown in Table 13-15. For these three composites, gold recovery was 4% to 12% to a 0.2% to 0.4% mass after panning a Knelson concentrate.

Table 13-15: Annual Composite Gravity Recovery (P_{80} of 250 µm)

Test	Composite	Product	Mass (%)	Assay (% or g/t)					Distribution (%)				
				Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
T60	Year 1	Pan Con	0.43	**124**	7420	49490	76650	29.4	**12**	16	3	14	5
T61	Year 2	Pan Con	0.24	**39.6**	4340	20620	36500	29.8	**4**	8	1	7	4

Test	Composite	Product	Mass (%)	Assay (% or g/t)					Distribution (%)				
				Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
T62	Year 3	Pan Con	0.21	43.2	2870	15460	20290	31.3	4	7	2	8	3

To further investigate the occurrence of gold particles over a range of grind sizes, an extended gravity recoverable gold (E-GRG) test was done on the 21A Low As sample. Successive Knelson concentrates are generated at 1,700 µm, 212 µm and 75 µm after regrinding the tailings stream from the previous run.

This would also provide the data required by gravity concentrator manufacturers to model plant recoveries based on different grinding stream circulating loads and concentrator locations in the circuit. Table 13-16 summarizes the E-GRG results.

Table 13-16: 21A Low As Composite Extended GRG Test

Sample ID	Product	Weight (%)	Au (g/t)	Au Distribution (%)
New 21A	Knelson Con1 (P_{80} 1700 µm)	0.3	70.7	4.3
	Knelson Con 2 (P_{80} 212 µm)	0.3	116	6.3
	Knelson Con 3 (P_{80} 75 µm)	0.3	107	5.4
	Knelson Tail 3	99.2	4.15	84.1
		Gravity Recoverable Gold	**96.6**	**15.9**

Finally, a parallel test was done on the Y1 Annual composite comparing gravity + sulphide flotation with flotation alone at a reasonably coarse primary grind size. No difference in overall gold recovery was observed by including gravity concentration.

For the 2021 PFS final flowsheet, gravity recovery is not included but remains an option to be considered based on additional testwork to be performed as part of a future feasibility study.

13.5.6 Rougher Flotation

A considerable number of rougher flotation tests were conducted by Base Met as part of the flowsheet development and optimization work. The primary test sample was 21A Low As with follow-up testing done on the three Annual composites.

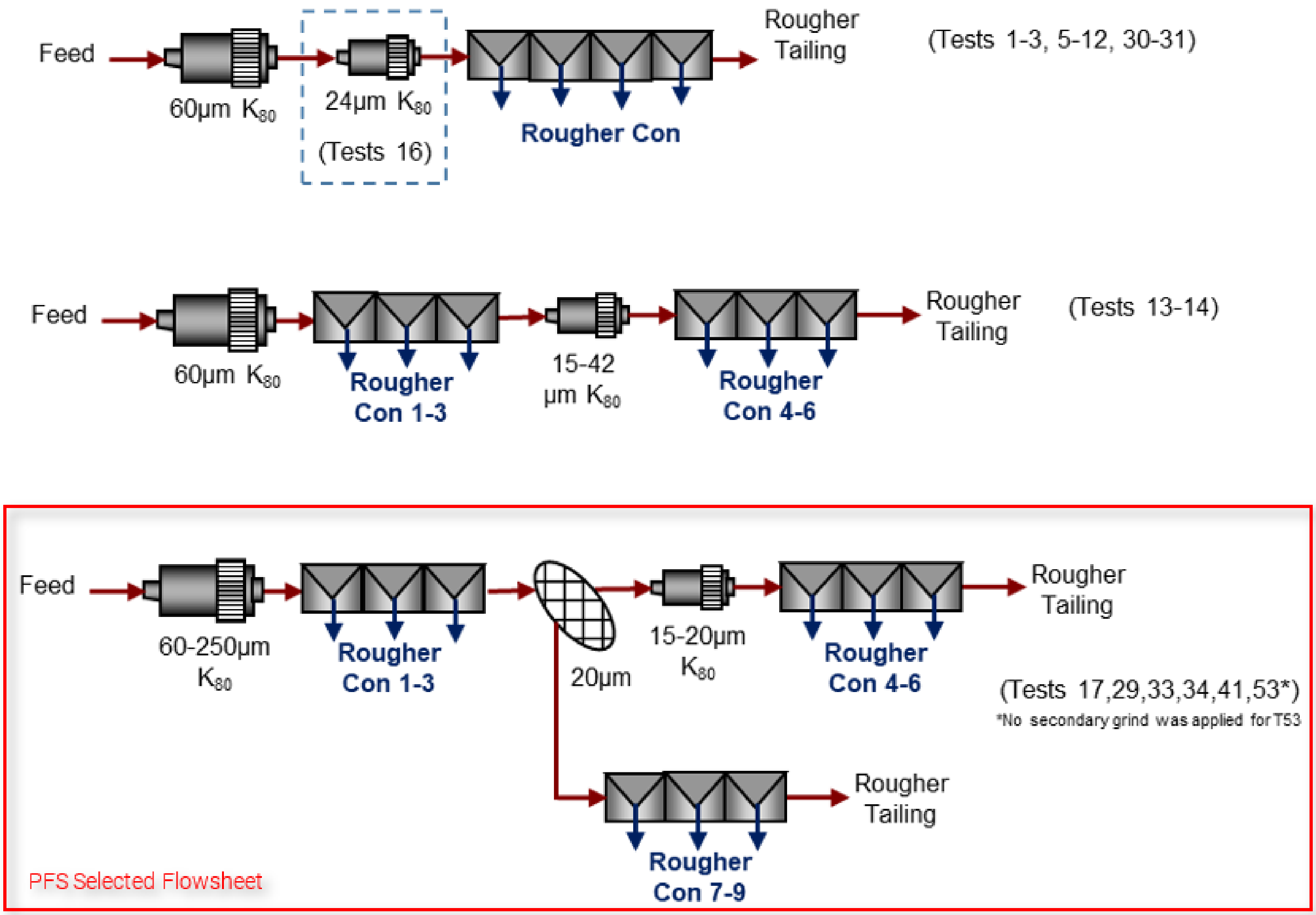
A range of rougher float conditions were investigated including low % solids and stainless steel grinding conditions, as was done by Blue Coast in the 2019 PEA work. In addition, a range of grind sizes, collector additions and the use of dispersants for NSG minerals was evaluated.

As shown in Figure 13-8, a range of different rougher flotation flowsheets were evaluated. The third flowsheet outlined in the figure was selected for the 2021 PFS and continued to be used for both locked cycle and Variability composite testing.

The PFS flowsheet involves rougher flotation (at a range of primary grind sizes), followed by a desliming stage to isolate the fine fraction (high % softer minerals such as muscovite/illite) which is separately floated in a 'fines' rougher circuit. The deslimed coarse fraction is 'secondary' reground before another stage of rougher flotation. Initially, this circuit targeted a primary grind P_{80} size of 60 μm which was coarsened to 250 μm before being finalized at 100 μm to avoid too high a size reduction in the regrinding stages.

The selected circuit is similar to the "Mill-Float-Mill-Float," or MF2 flowsheets used in platinum process plants.

Figure 13-8: Rougher Flotation Flowsheets Evaluated



Note: Figure prepared by Base Met, 2021.

The three Annual composites were tested using the rougher flotation highlighted in Figure 13-8 with Base Met including some comments in their testwork report (Base Met, 2021a). For Year 1, primary grind size did not impact gold recovery with a 100 μm P_{80} reporting 26% mass pull and 79.5% recovery. For Year 2, primary grind size was significant with a coarser grind improving Au recovery up to 74.1% with a 18% mass pull. For Year 3, the 100 μm P_{80} primary grind produced 80.7% Au recovery with a 25% mass pull to concentrate.

These results encourage an investigation into coarser primary grind sizes which was not included in the final process design criteria due to high regrind specific energy requirements.

13.5.7 Cleaner Flotation

Open circuit cleaner testing followed the rougher float tests, all done on sample 21A Low As with the results summarized in Table 13-17. A range of primary grind (PG) sizes, deslime screen size (20 µm to 53 µm), secondary grind (SG) and regrind (RG) sizes were investigated.

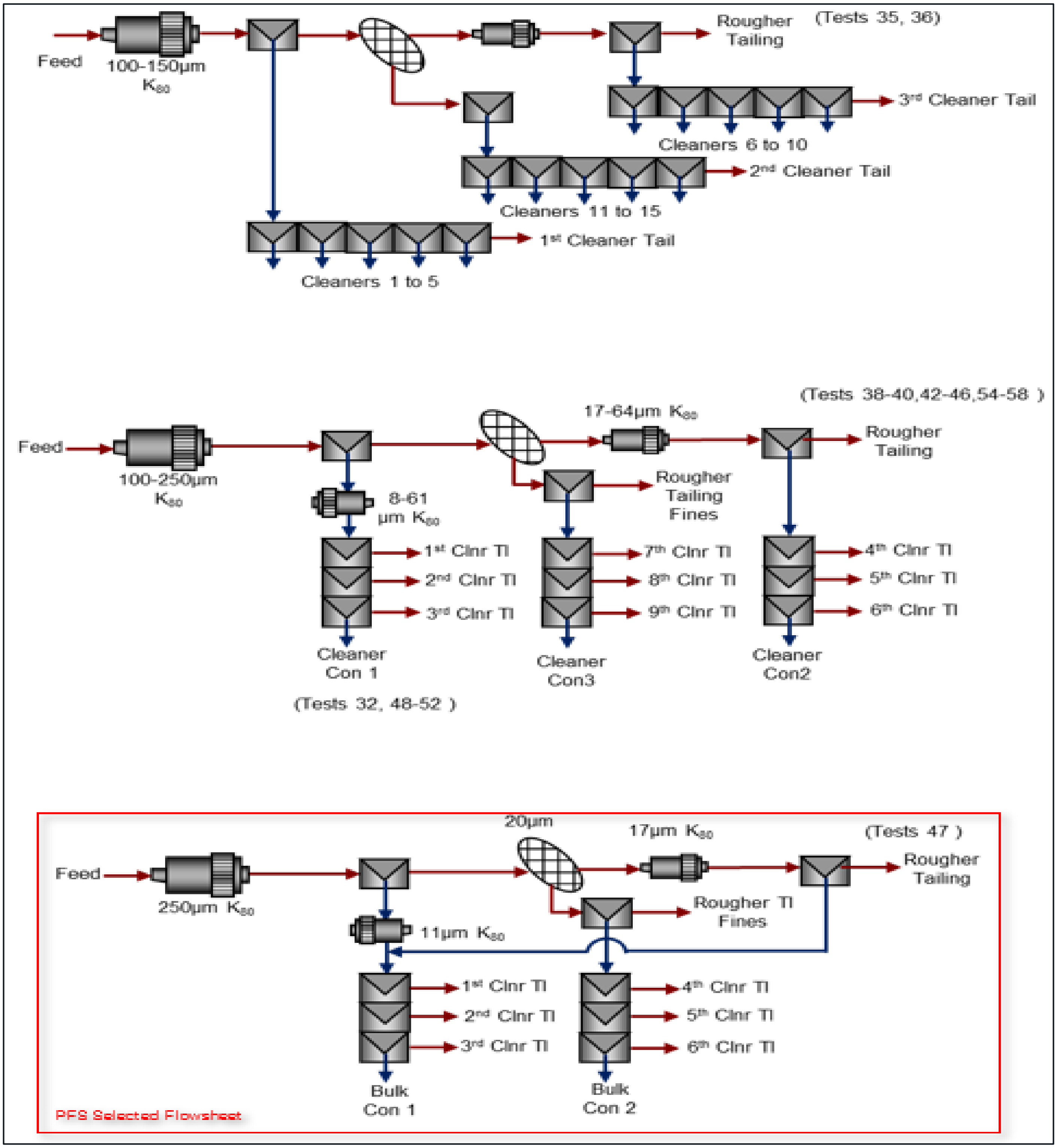
Figure 13-9 summarizes the different cleaner circuit configurations investigated, with the third one selected for the 2021 PFS. This flowsheet included rougher flotation followed by regrinding of the concentrate before three stages of cleaning. The rougher tails was deslimed at 20 µm with the coarse fraction secondary ground to 17–20 µm before being refloated and concentrate combined with the reground product stream. The fines circuit consisted of roughing and three cleaner stages. The final tailings is a combination of secondary ground rougher tails and fines rougher tails—both very fine at less than 20 µm in size.

Table 13-17: 21A Low As Composite Open Circuit Cleaner Tests

Test	Sizing (µm P_{80})		Screen (µm)	Product	Mass (%)	Assay (% or g/t)					Distribution (%)				
	PG	RG				Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
T35	100/19	-	20	Combined Cons	9.0	15.3	547	2272	2225	11.7	73	94	67	90	77
T36	150/19	-	20	Combined Cons	10.4	13.6	465	2111	2117	10.8	78	93	71	89	81
T38	100/19	12	20	Combined Cons	2.9	49.5	1544	6419	6926	33.3	74	94	61	90	75
T39	150/19	13	20	Combined Cons	3.4	42.0	1466	6148	6455	30.9	74	93	65	89	76
T40	200/22	13	20	Combined Cons	3.4	44.7	1479	6548	7577	32.3	74	93	65	89	76
T42	250/18	14	20	Combined Cons	3.3	47.6	1589	6918	7446	34.1	76	93	71	88	80
T43	250/28	10	20	Combined Cons	3.3	50.5	1491	6335	6669	31.1	76	93	70	89	78
T44	250/24	12	20	Combined Cons	3.2	51.7	1644	6436	7306	31.1	72	92	67	88	73
T45	250/21	8	38	Combined Cons	2.6	49.0	1829	6819	7251	34.8	64	89	56	81	66
T46	250/21	8	53	Combined Cons	2.4	45.5	1833	6560	7039	34.7	55	89	52	82	62
T47	250/17	11	20	Combined Cons	2.9	47.1	1641	6836	6523	36.3	69	93	66	86	76
T48	250	10	-	Bulk Con	1.7	48.2	2392	6452	10080	36.0	44	79	37	72	46
T49	250	16	-	Bulk Con	1.7	49.1	2412	5900	10424	34.2	44	79	33	73	43
T50	250	18	-	Bulk Con	2.1	47.5	2084	5332	8996	29.0	49	77	37	74	44
T51	250	24	-	Bulk Con	1.7	61.5	2270	5298	9112	31.0	43	66	32	73	39
T52	250	52	-	Bulk Con	2.9	32.0	1540	3456	6223	20.5	46	78	41	79	45

Test	Sizing (µm P_{80})		Screen (µm)	Product	Mass (%)	Assay (% or g/t)					Distribution (%)				
	PG	RG				Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
T54	250/64	61	20	Combined Cons	7.7	16.1	579	2464	2635	12.9	70	89	68	89	74
T55	250/50	46	20	Combined Cons	7.0	20.0	717	2799	3108	15.1	72	91	73	85	74
T56	250/75	60	20	Combined Cons	7.4	17.9	657	2622	3264	12.6	68	89	64	89	69
T57	250/52	49	20	Combined Cons	5.2	23.4	822	3602	4490	18.2	65	90	61	88	70
T58	250/32	41	20	Combined Cons	3.9	27.2	1287	4092	5882	22.9	60	88	54	83	66

Figure 13-9: Cleaner Flotation Flowsheets Evaluated



Note: Figure prepared by Base Met, 2021.

The third flowsheet shown in Figure 13-9 was tested on the three Annual composites. Base Met included some comments on these tests in their final report (Base Met, 2021a). For Year 1, a primary grind P_{80} size ranging from 100–250 µm was evaluated along with a secondary grind P_{80} size of 15 µm. The finer primary grind resulted in higher intermediate cleaning grade and recovery but did not impact the final gold concentrate grade and recovery. In the combined concentrates, gold recovery was 60% at 35g/t Au.

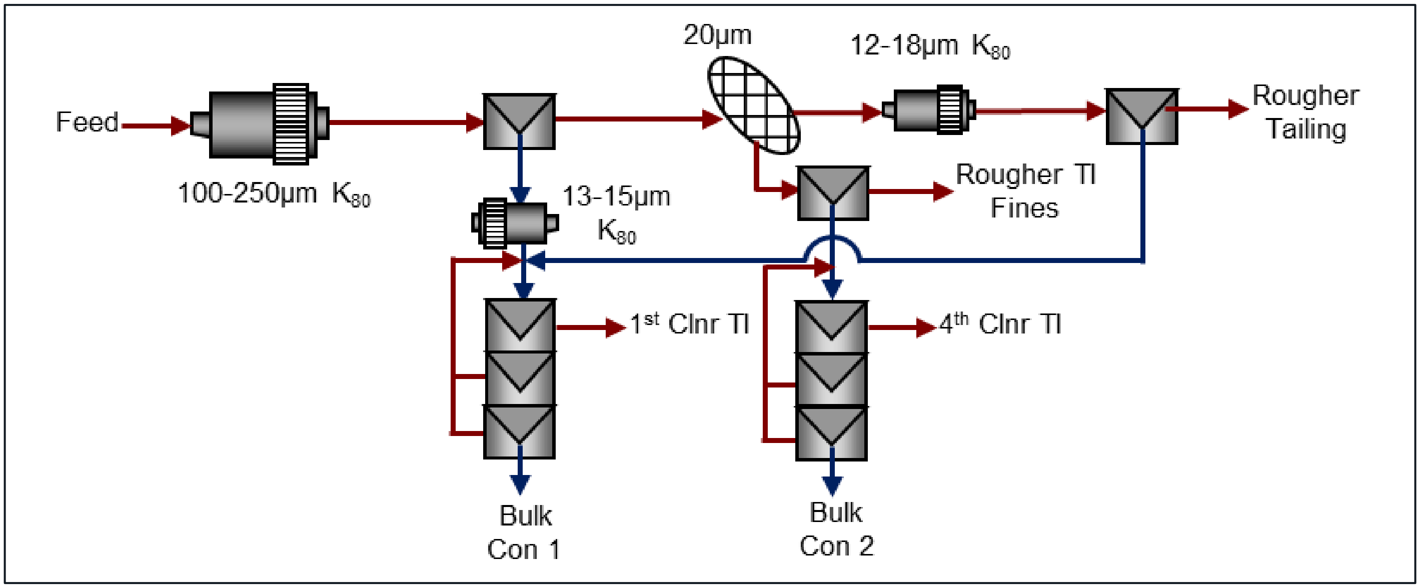
For Year 2, a coarser primary grind size was successful with a combined concentrate gold recovery of 69.5% to a 34 g/t Au grade. Year 3 produced similar results to Year 2 with a combined concentrate grade of 24 g/t Au and 64.9% recovery.

13.5.8 Locked Cycle Tests

Locked cycle testing of the three Annual composites was conducted at a primary grind P_{80} size of 250 µm, which was the target grind at that time in the testwork program. After the design criteria was shifted to a primary grind P_{80} size of 100 µm, follow-up open circuit rougher/cleaner tests were done at this size for recovery estimation at the final design conditions.

The flowsheet used for the three locked cycle tests (T63 to T65) is shown in Figure 13-10 with the results summarized in Table 13-18.

Figure 13-10: Annual Composite Locked Cycle Test Flowsheet



Note: Figure prepared by Base Met, 2012.

The locked cycle work targeted a primary grind P_{80} size of 250 µm, with a secondary and regrind sizes of 15 µm. Circuit stability was established at four and five cycles except for Year 3. Base Met recommended that future locked testing include an additional cycle for stability. To a combined final concentrate, the three Annual composites reported 67.1%, 77.2% and 78.6% gold recovery to a 30.8 g/t, 23.7 g/t and 25.2 g/t Au grade.

It should be noted that Variability testing produced a wide range of final concentrate grades (up to 90 g/t Au) depending on pyrite and NSG content of the samples. The Annual composites all had relatively high levels of muscovite/ illite (see Table

13-10) as well as potassium feldspars which impacted final concentrate recovery and grade. The results of the locked cycle work showed circuit stability, but Variability test results were relied upon to estimate metallurgical performance.

Table 13-18: Annual Composite Locked Cycle Test Results (Primary Grind P_{80} of 250µm)

Product	Weight (%)	Assay (% or g/t)					Distribution (%)				
		Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
Year 1 Composite, T63, Cycles D+E											
Feed	100.0	3.92	152	6667	2372	2.48	100	100	100	100	100
Bulk Con 1	8.1	31.4	1740	77313	27715	19.3	65.0	92.7	93.9	94.6	63.0
Cleaner Tl 1	13.9	4.39	35	1442	409	2.27	15.6	3.2	3.0	2.4	12.7
Bulk Con 2	0.4	19.0	209	6767	2811	9.43	2.1	0.6	0.4	0.5	1.7
Cleaner Tl 4	3.2	2.57	32.0	856	318	1.21	2.1	0.7	0.4	0.4	1.6
Ro Fines Tl	13.3	0.75	6.5	282	77	0.40	2.5	0.6	0.6	0.4	2.2
Ro Tl	61.1	0.81	5.7	183	62	0.77	12.7	2.3	1.7	1.6	18.9
Combined Cons	8.5	30.8	1661	73671	26430	18.8	67.1	93.3	94.3	95.1	64.7
Year 2 Composite, T64, Cycles D+E											
Feed	100.0	2.50	132	4694	1382	1.98	100	100	100	100	100
Bulk Con 1	7.6	24.3	1660	58758	17572	20.4	74.3	95.7	95.6	97.0	79.0
Cleaner Tl 1	11.5	1.64	16	606	116	1.03	7.5	1.4	1.5	1.0	6.0
Bulk Con 2	0.5	14.7	154	5506	1026	8.85	2.9	0.6	0.6	0.4	2.2
Cleaner Tl 4	3.7	1.86	19.5	762	130	1.36	2.7	0.5	0.6	0.3	2.5
Ro Fines Tl	14.9	0.61	4.3	216	43	0.47	3.6	0.5	0.7	0.5	3.6
Ro Tl	61.8	0.36	2.8	84	18	0.21	8.9	1.3	1.1	0.8	6.7
Combined Cons	8.1	23.7	1567	55477	16553	19.7	77.2	96.3	96.1	97.4	81.2
Year 3 Composite, T65, Cycles D+E											
Feed	100.0	2.56	86	2257	656	1.85	100	100	100	100	100
Bulk Con 1	7.4	26.1	1080	28741	8296	18.1	75.1	92.8	94.0	93.4	71.9
Cleaner Tl 1	9.9	1.40	15	365	80	1.00	5.4	1.7	1.6	1.2	5.4
Bulk Con 2	0.6	14.7	138	2758	684	10.3	3.5	1.0	0.8	0.6	3.4
Cleaner Tl 4	3.2	2.38	30.8	600	157	1.67	3.0	1.2	0.9	0.8	2.9
Ro Fines Tl	15.6	0.64	6.0	165	58	0.68	3.9	1.1	1.1	1.4	5.7

Product	Weight (%)	Assay (% or g/t)					Distribution (%)				
		Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
Ro Tl	63.3	0.37	3.0	59	27	0.31	9.0	2.2	1.7	2.6	10.6
Combined Cons	8.0	25.2	1008	26743	7711	17.5	78.6	93.8	94.7	94.0	75.4

13.5.9 Variability Testing

Following locked cycle testing, each of the Variability composites were tested using the open circuit flowsheet outline in Figure 13-9. These drill core samples came from a range of Zones, including 21B, 21E and HW not included in the 2019 PEA testwork program. These tests were all conducted at a primary grind P_{80} size of 100 µm, or the design criteria target size (see Table 13-19).

Table 13-19: Variability Composite (VC) Open Circuit Cleaner Tests (Primary Grind P_{80} of 100 µm)

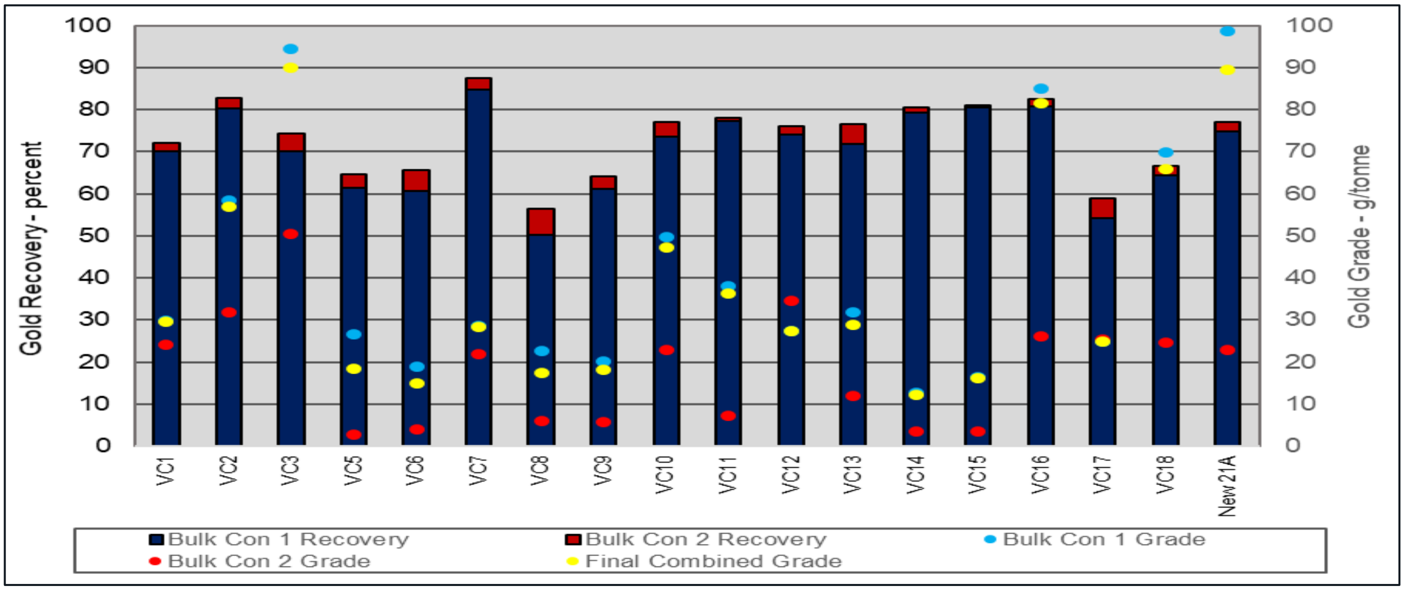
Zone	Variability Comp ID	Test	Combined Bulk Cons									
			Grade					Recovery (%)				
			Au g/t	Ag g/t	As g/t	Sb g/t	S %	Au	Ag	As	Sb	S
21A	VC1	70	29.5	2439	192735	43647	21.2	72.0	95.8	96.5	93.0	82.9
21A	VC2	71	57.0	953	6580	4458	34.7	82.7	95.3	89.5	91.2	89.7
21A	VC3	72	90.0	1801	8955	33905	27.7	74.3	90.7	72.3	93.0	37.2
21E	VC5	73	18.4	326	1173	8285	13.7	64.6	67.5	38.0	80.7	43.9
21E	VC6	74	14.8	1135	2637	6506	14.8	65.7	72.1	42.9	73.6	51.1
21E	VC7	75	28.2	5212	3120	22445	29.7	87.6	93.1	83.1	94.2	89.0
HW	VC8	76	17.3	191	1873	1220	20.1	56.4	62.3	47.9	55.0	52.6
HW	VC9	77	18.1	1619	1115	7085	22.4	64.2	86.3	47.1	87.4	46.9
21B	VC10	78	47.3	50	2609	461	23.3	77.1	68.8	79.4	71.8	77.3
21B	VC11	79	36.3	125	2784	640	18.1	78.2	85.7	79.5	75.8	86.3
21B	VC12	80	27.4	69	2478	310	23.3	76.0	82.3	78.7	65.0	84.9
21C	VC13	81	28.8	676	1419	3651	17.3	76.7	89.7	55.2	90.7	41.2
21C	VC14	82	12.2	161	5870	1039	39.8	80.6	75.9	68.6	77.8	79.3
21C	VC15	83	16.1	627	4106	4180	32.5	80.9	91.7	73.9	93.7	78.9
22	VC16	84	81.6	12905	3889	15978	32.2	82.5	93.5	69.2	84.5	51.2
22	VC17	85	24.8	1987	3544	3968	12.8	59.0	82.3	56.5	59.8	63.9
22	VC18	86	65.9	7348	8544	9278	22.8	66.5	90.7	59.2	72.0	66.3

Zone	Variability Comp ID	Test	Combined Bulk Cons									
			Grade					Recovery (%)				
			Au g/t	Ag g/t	As g/t	Sb g/t	S %	Au	Ag	As	Sb	S
21A	New 21A	87	89.6	10147	10961	33596	31.6	77.2	97.8	73.0	92.7	76.7

Figure 13-11 summarizes the results showing the individual and combined concentrate grades and gold recovery. These results produced a range of recovery versus final concentrate grade curves (see Results of variability composite testing produced a range of combined concentrate grades and impurities in Table 13-19 and Figure 13-11). Base Met observed a trend between the final concentrate grade and Au/(S+Fe) ratio and commented on this in their report (Base Met, 2021a). This ratio reflects the balance between gold content and either pyrite or NSG minerals reporting to final concentrate (see Figure 13-13). Further investigation of the head sample and final concentrate mineralogy showed wide variations in NSG mineral content. Samples were then broadly categorized as high in quartz, feldspar, chlorite or muscovite/ illite.

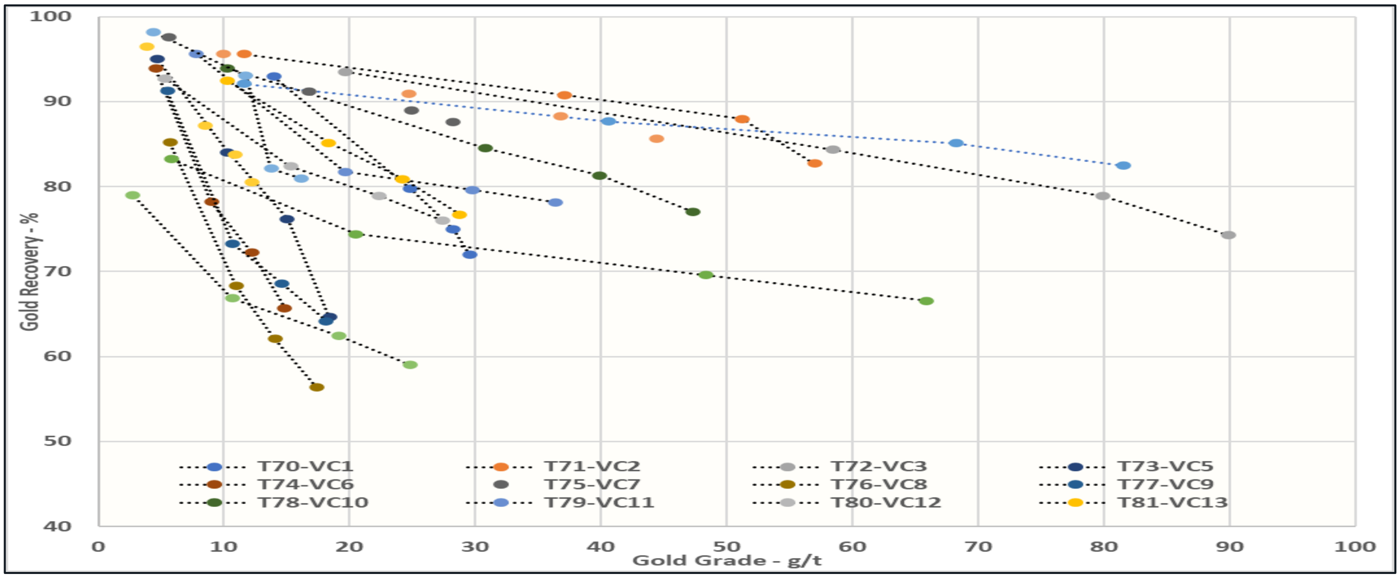
Additional investigation was done by Base Met to determine if some of these NSG minerals could be depressed. In addition, a review of the different mineralized zones was done to compare the Variability composite samples with the Eskay Creek orebody. This resulted in a relative ranking of the zones in terms of their potential to produce a 25 g/t to 45 g/t Au concentrate as well as the impact on final gold recovery.

Figure 13-11: Variability Composite Recovery and Combined Concentrate Grades



Note: Figure prepared by Base Met, 2021.

Figure 13-12: Variability Composite Recovery versus Concentrate Grade

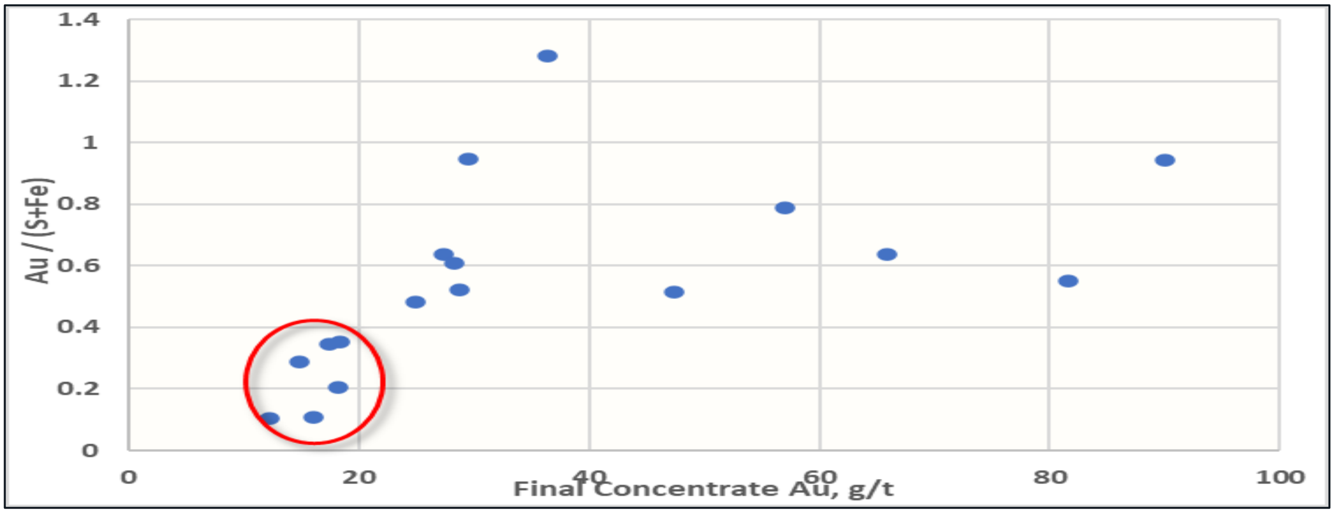


Note: Figure prepared by SRK, 2021.

13.5.10 Final Concentrate

Results of the Variability composite testing produced a range of combined concentrate grades and impurities (see Table 13-19 and Figure 13-11). Base Met observed a trend between the final concentrate grade and Au/(S+Fe) ratio and commented on this in their report (Base Met, 2021a). This ratio reflects the balance between gold content and either pyrite or NSG minerals reporting to final concentrate (Figure 13-13).

Figure 13-13: Variability Composites Au/ (S+Fe) vs. Combined Concentrate Grade



Note: Figure prepared by SRK, 2021.

For samples with ratios >0.4, 25 g/t up to 90 g/t Au final concentrate could be achieved. For samples with ratios <0.4, the high pyrite and/or NSG content made achieving a high concentrate grade challenging without significant gold losses. While the bulk sulphide flotation conditions are relatively simple, fine grinding of both concentrate and tailings streams to liberate gold particles resulted in similar flotation kinetics for both sulphide and non-sulphide minerals.

It was noted that the samples tested in the 2019 PEA program had relatively high Au/(S+Fe) ratios and, in general, higher gold grades. A comparison between the 2021 PFS test samples and the mineralized zones showed that blocks with ratios below 0.4 were a minority and related to some zones, particularly21E and HW. On average, the mudstone units also had lower Au/(S+Fe) ratios compared to Eskay rhyolite material.

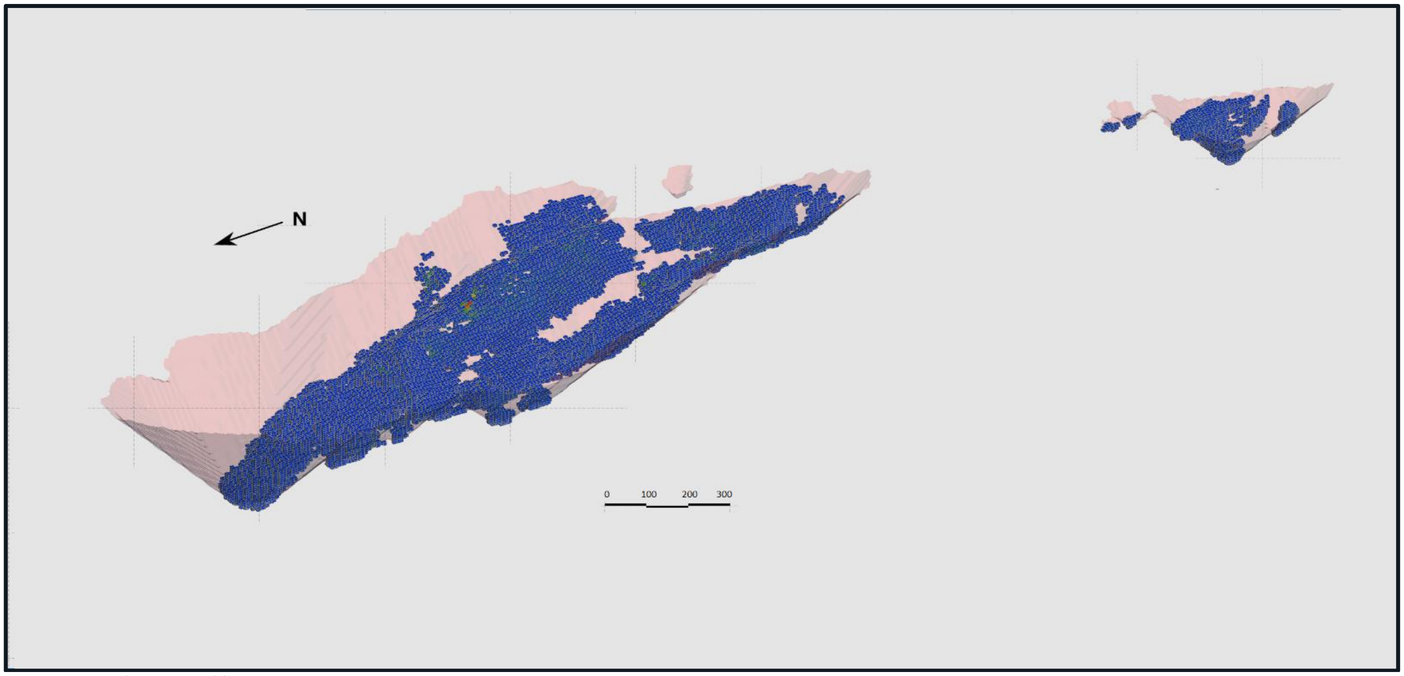
Using the Au/(S+Fe) ratio as an indicator, the zones were grouped into four categories:

•	22, 21A and 21C	generally ratios >0.4;
•	21B, 21Be and 21E	mixed ratios from 0.2 to >0.4;
•	HW	generally ratios <0.4;
•	NEX, 109	untested at this stage, but historical results suggest similar to 21A.

Of these categories, the first represents 40% of the mineralized blocks within the pit shell while the second is another 30%. HW zone material represents only 7% of the mineralized blocks within this pit, while NEX and 109 were not included in either the 2019 PEA or the 2021 PFS testwork program.

The 2019 PEA final pit shell compared with blocks having Au/(S+Fe) ratios of >0.4 is shown in Figure 13-14.

Figure 13-14: Perspective View Showing Ore Blocks with Au/(S+Fe) > 0.4

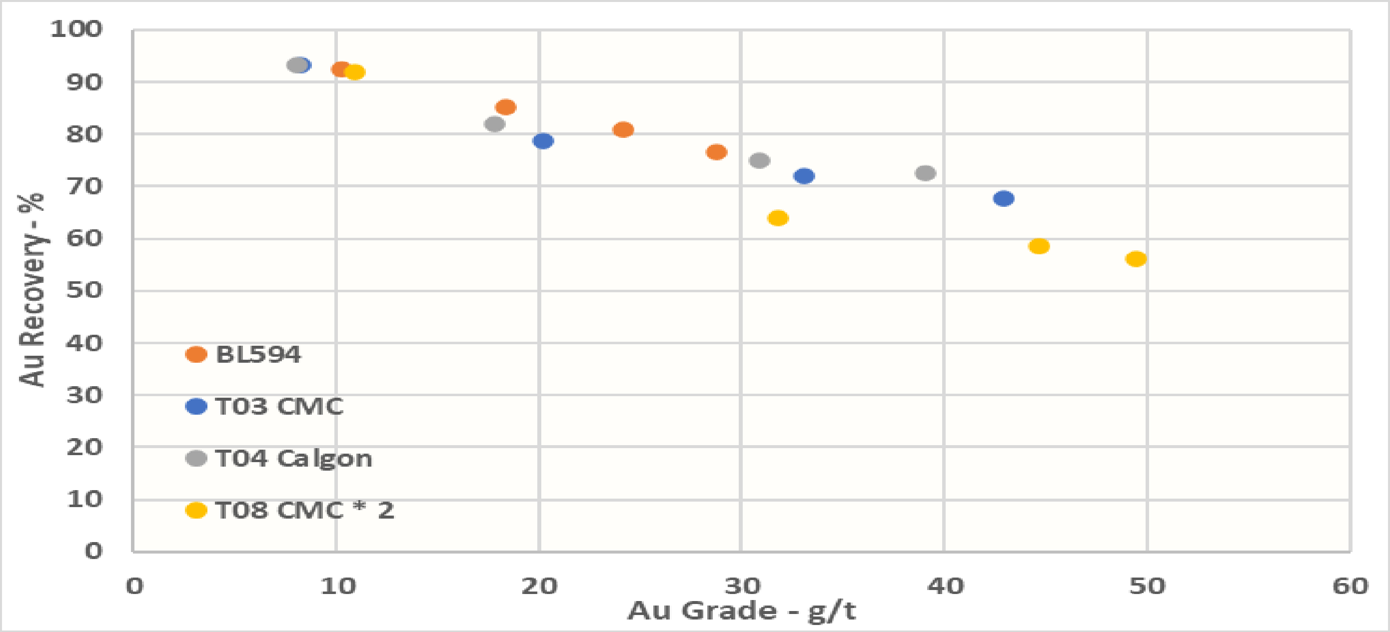


Note: Figured prepared by SRK, 2021.

To investigate options to depress NSG minerals from the final concentrate, Base Met continued their testwork program and reported separately as BL777 (Base Met, 2021b). A number of Variability samples were re-tested with both Calgon and carboxymethyl cellulose (CMC) depressants, to determine if NSG minerals could be depressed without gold losses. The results for VC 13, a sample from the 21C Zone containing barite with Au/(S+Fe) of 0.52, are shown in Figure 13-15.

Compared with base conditions in the BL594 test program, the addition of depressant did not show any selectivity between the NSG minerals and gold. While higher concentrate grades were achieved compared with the base test of 29 g/t Au, gold losses continued to follow the same recovery versus concentrate grade curve.

Figure 13-15: VC13 Cleaner Test with Gangue Depressant



Note: Figure prepared by SRK, 2021.

13.5.11 Solid/Liquid Separation

A number of settling, pressure filtration and vacuum filtration tests were conducted on concentrate samples generated from bulk testing and a number of Variability composites. The current process flowsheet includes thickening and filtration of the final flotation concentrate while both tailings streams are to be discharged directly into the tailings management storage facility (TMSF).

The bulk test (T88) was done at a primary grind P80 size of 100 µm on 100 kg of sample "21A 2020", as described in Section 13.5.1. Flocculant scoping test results are shown in Table 13-20 using five different flocculant types: Magna Floc 351, 380, 336, 10 and 1011. Magnafloc 336 was selected because it provided the best overflow clarity and the fastest free settling rate on the concentrate.

Table 13-20: Bulk Test Final Concentrate Flocculant Scoping Tests

Sample	Flocc	Flocc Type	Final Density (% solids)	Settling Rate (mm/s)	Final Clarity
T88-Final Con	MF351	Non-Ionic	19.6	0.03	Turbid
	MF380	Cationic	17.5	0.02	Turbid
	MF336	Anionic	26.1	0.08	Clear
	MF10	Anionic	19.7	0.03	Turbid
	MF1011	Anionic	19.6	0.03	Turbid

Note: all tests performed at pH 7.4 and 13.5% initial density with 50 g/t flocc dosage

Using Magnafloc 336, a number of static settling tests were conducted on final concentrates from Test 88 and Variability composite cleaner tests (Table 13-21). Final densities ranged from 37–60% solids for the Variability samples, all tested at the same initial density of 9% solids with 50 g/t flocculant dosage.

Table 13-21: Final Concentrate Static Settling Tests

Sample ID	pH	Dosage (g/t)	Initial Density (% solids)	Final Density (% solids)	Settling Rate (mm/s)
T88-Final Con	7.4	50	15.6	33.2	1.51
	11.0	50	15.3	30.1	1.23
	7.4	25	15.3	30.0	0.85
	7.4	75	15.3	34.2	1.59
	7.4	50	10.8	31.0	2.09
	11.0	75	15.3	30.9	1.53
T70-Final Con VC1	6.8	50	9.3	47.7	1.32
T72-Final Con VC3	7.3	50	9.2	37.4	0.69
T73-Final Con VC5	4.6	50	9.3	38.0	0.31
T74-Final Con VC6	6.0	50	9.3	37.1	0.35
T75-Final Con VC7	6.5	50	9.2	45.5	0.77
T76-Final Con VC8	5.1	50	9.2	38.5	0.33
T77-Final Con VC9	6.3	50	9.3	36.5	0.44
T78-Final Con VC10	5.3	50	9.0	42.9	0.45
T79-Final Con VC11	6.1	50	9.3	39.3	0.43
T80-Final Con VC12	5.7	50	9.2	37.9	0.65
T81-Final Con VC13	6.2	50	9.3	38.0	0.46
T82-Final Con VC14	4.4	50	9.1	59.9	0.78
T83-Final Con VC15	5.0	50	9.2	52.4	0.68

Sample ID	pH	Dosage (g/t)	Initial Density (% solids)	Final Density (% solids)	Settling Rate (mm/s)
T87-Final Con New 21A	5.1	50	9.2	47.0	0.68

Dynamic settling tests were conducted by Base Met with a bench-scale thickener using MF336 at 0.5 g/L concentration. In this setup, tailings slurry and flocculant were continually fed into the thickener feed well. Over the range of flocculant dosage tested, underflow densities of 38% to 51% solids were reported (Table 13-22).

Table 13-22: Bulk Test Final Concentrate Dynamic Settling Tests

Test Product	Loading Rate (t/m²/hr)	Flocc Dosage (g/t)	U/F Density (% solids)	TSS (mg/L)
Test 88 Final Concentrate	0.3	50	46.6	1,468
	0.5	50	47.1	692
	0.7	50	43.6	2,956
	0.5	25	38.3	2,542
	0.5	75	51.2	370
	0.5	100	47.9	341

Note: all tests performed at 0.50g/L flocc concentration

Pressure and vacuum filtration tests were completed on the bulk test final concentrate (Table 13-23 and Table 13-24). For pressure filtration tests, blow times of 30, 60, 180, 300 and 600 seconds were assessed. Cake moistures of 15% to 25% were achieved over a range of thicknesses.

Three vacuum filtration tests were conducted on T88 final concentrate at various feed masses, achieving a consistent 30% to 34% final cake moisture.

Table 13-23: Bulk Test Final Concentrate Pressure Filtration Tests

Sample	Sample Mass (grams)	Blow Time - sec		Cake Thickness (mm)	Cake Moisture (%)	Filter Rate (kg/m²/hr)
		Total	Filter Time			
Test 88 Concentrate	30	30	6	9	22.3	1636
		60	9	10	18.7	903
		180	11	12	14.6	320
	60	60	42	20	25.1	2033
		180	49	25	19.3	687
		300	49	23	17.4	391

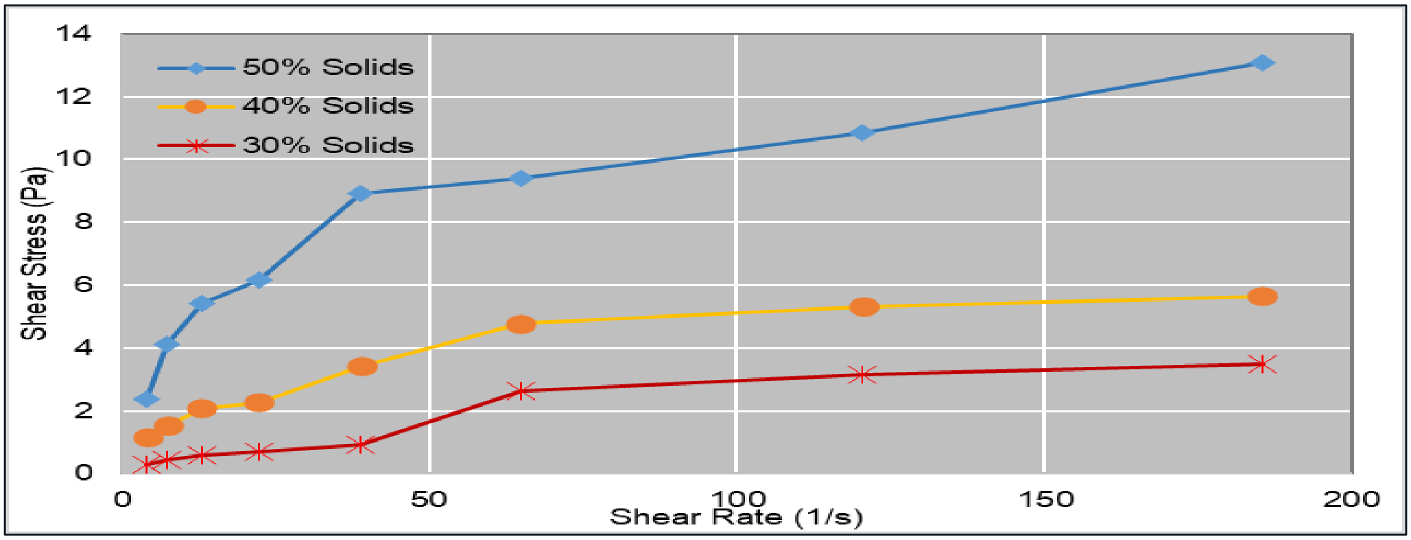
	90	180	150	37	24.9	1020
		300	147	36	21.1	600
		600	153	36	18.4	318

Table 13-24: Bulk Test Final Concentrate Vacuum Filtration Tests

Test	Feed Mass (g)	Final Moisture (%)	Filter Rate (kg/m^2/hr)
VF1	94.2	34.1	1049
		32.6	773
		32.1	612
		31.8	506
VF2	143.3	34.2	705
		33.0	609
		32.1	536
		31.4	479
		31.2	432
		31.0	394
		30.8	335
VF3	194.5	34.4	471
		33.5	437
		32.8	408
		32.1	382
		31.1	339
		30.8	305
		30.7	277

Viscosity measurements were done to identify any mixing/ pumping issues (Figure 13-16). Base Met stated there should not be any issues pumping the concentrate at 50% solids or rougher tailings at 30% solids. However, for tailings at 40% solids, the viscosity was slightly higher and may require viscosity modifying agents or specialty pumping applications. There would be no issues with regards to mixing and screening applications for these streams at the tested densities (Base Met, 2021a).

Figure 13-16: **Bulk Concentrate Viscosity (Shear Stress vs Shear Rate)**

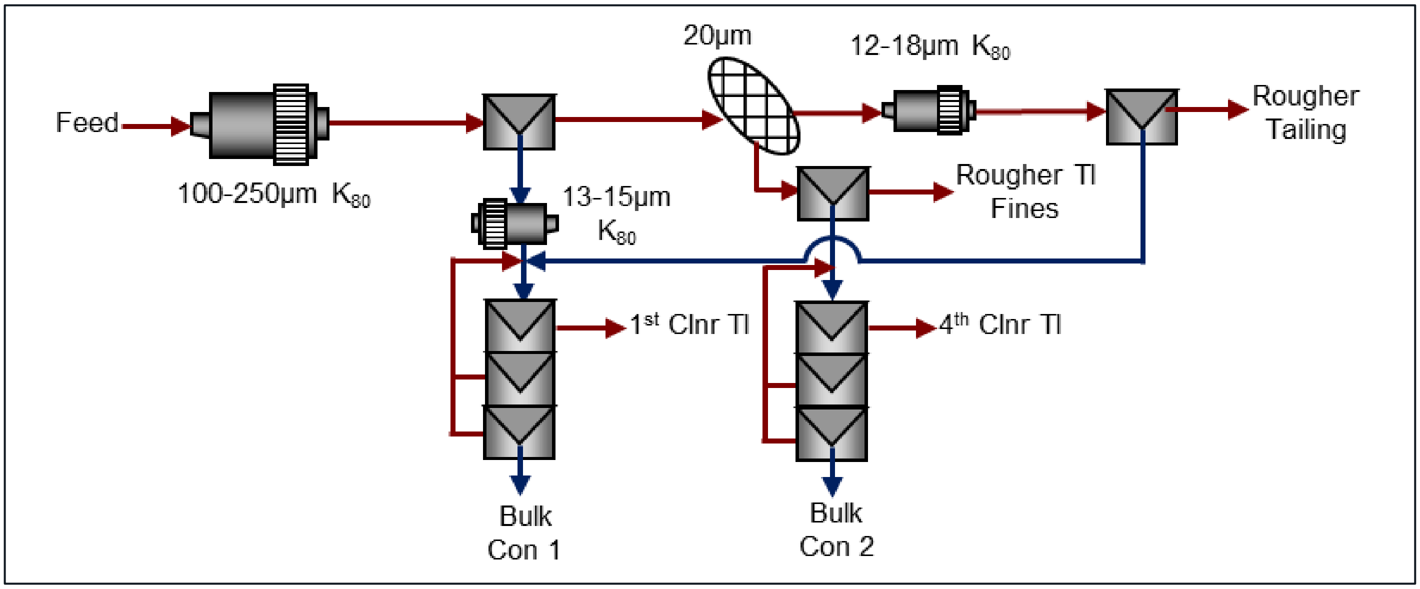


Note: Figure prepared by SRK, 2021.

13.5.12 Final Flowsheet

The final MF2 flowsheet selected for the 2021 PFS is shown in Figure 13-17 and was used for both the locked cycle testing as well as the Variability composite evaluation that was the basis for the recovery estimates.

Figure 13-17: **Final MF2 Flowsheet for Recovery Estimation**



Note: Figure prepared by Base Met, 2021.

A number of Variability composites were tested using both the original 2019 PEA flowsheet (bulk rougher + cleaner flotation) and the MF2 flowsheet shown in Figure 13-17. For all samples, the final MF2 flowsheet produced a higher recovery versus final concentrate grade relationship. The improvement was more noticeable for the high NSG content samples, or those with low Au/(S+Fe) ratios.

13.5.13 Reagent Additions

Based on the locked cycle test sheets for the three Annual composites, the following total reagent usage (in g/t mill feed) were reported by Base Met:

- Collector: potassium amyl xanthate (PAX) 725 g/t;
- Activator: copper sulphate ($CuSO_4$) 600 g/t;
- Frother: methyl isobutyl carbinol (MIBC) 250 g/t;
- Depressant: carboxy methyl cellulose (CMC) 75 g/t (as required).

In addition, the total laboratory flotation time was:

- Rougher 8 min;
- Deslimed Rougher 9 min;
- Coarse Cleaner 24 min;
- Fines Rougher 15 min;
- Fines Cleaner 24 min.

These tests were performed at low pulp densities (˜22% solids) which was found in both the 2019 PEA and 2021 PFS testing to improve flotation kinetics before the desliming stage.

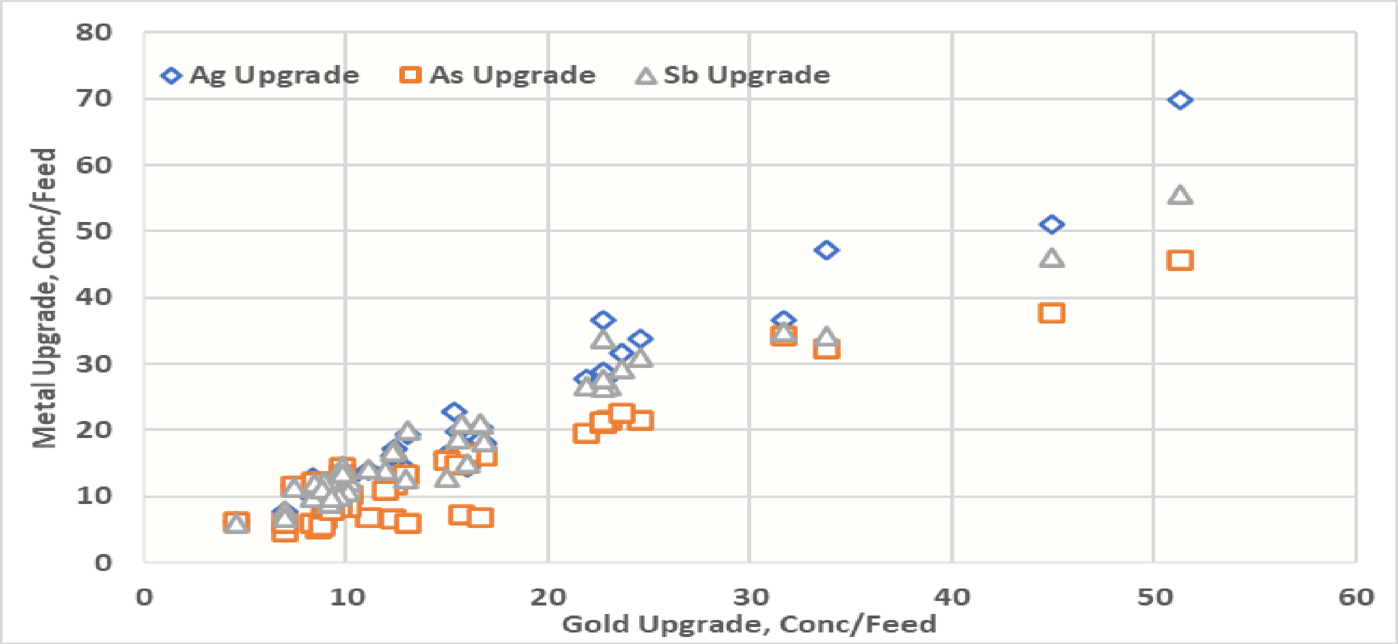
13.5.14 2021 PFS Recovery Estimates

For mine planning purposes, a number of recovery models were developed from the 2021 PFS testwork results. The 2019 PEA recoveries were based on head grade. For a 25 g/t final concentrate grade, mass pull to concentrate = 4 * Au head grade (up to 32%) with:

- <0.5g/t Au = 50% Au recovery;
- <1.0g/t Au = 75% Au recovery;
- <1.5g/t Au = 80% Au recovery;
- <2.0g/t Au = 85% Au recovery;
- <2.5g/t Au = 90% Au recovery;
- >2.5g/t Au = 92% Au recovery;
- <50g/t Ag = 50% Ag recovery;
- <100g/t Ag = 90% Ag recovery;
- <100g/t Ag = 97% Ag recovery.

For other sulphide-occurring metals (including arsenic, antimony, and mercury), both the 2019 PEA and 2021 PFS testwork campaigns showed similar upgrading to concentrate that was reported for gold. Figure 13-18 shows the results for the MF2 flowsheet testwork, with silver, arsenic and antimony all following the constant factor times gold upgrading to concentrate. Figure 13-3 shows this was also observed in the historical testwork on much higher-grade samples.

Figure 13-18: Variability Composites Metal Upgrade vs. Gold Upgrade



Note: Figure prepared by SRK, 2021.

From the 2021 PFS testwork results performed on a much wider range of sample compositions from different zones, a new set of equations were used for metal recoveries.

For concentrate grades:

- Au/(S+Fe) > 0.7 = 45 g/t Au;
- Au/(S+Fe) > 0.6 = 35 g/t Au;
- Au/(S+Fe) > 0.5 = 25 g/t Au;
- Au/(S+Fe) > 0.4 = 20 g/t Au.

For Au/(S+Fe) > 0.4:

- Au recovery = A * (1 – exp(B*Au head grade));

where:

- A = -0.3112 * Au conc grade + 96.753 (valid for 20 g/t to 90 g/t conc grade);
- B = -0.0075 * Au conc grade – 1.0716 (valid for 20 g/t to 90 g/t conc grade).

For Au/(S+Fe) <= 0.4:

- Au recovery = 72% to 20 g/t Au concentrate.

Other metal recoveries:

- Ag upgrade = 0.93 * Au upgrade;
- As upgrade = 0.92 * Au upgrade;
- Sb upgrade = 1.07 * Au upgrade;
- S upgrade = 0.91 * Au upgrade;
- Hg upgrade = 0.80 * Au upgrade.

For any mineralized blocks missing gold, iron, or sulphur assays to categorize the material, the 2019 PEA recoveries were assigned. This occurred mainly in the NEX and 109 Zones that have not yet been tested by Skeena. However, based on historical testwork results (see Figure 13-2), it appears the NEX Zone material should exhibit similar performance to the 21A Zone material.

With the wider range of samples tested in the 2021 PFS program, the different NSG mineral compositions were found to impact the final concentrate recovery vs. grade curves, as shown in Figure 13-12. This resulted in a ranking of the different mineralized zones based on Au/(S+Fe):

22, 21A and 21C	likely to generate 45 g/t Au concentrate;
21B, 21Be and 21E	likely to generate 25 g/t to 45 g/t Au concentrate;
HW	likely to generate 20 g/t to 25 g/t Au concentrate;
NEX, 109	untested at this stage.

The 2021 PFS set of equations used for metal recoveries are acceptable for use in the MRMR estimates and LOM plan used in financial modelling.

13.6 Recommended Future Testwork

For the MF2 flowsheet selected for the 2021 PFS, additional testwork is warranted to improve the confidence in the metallurgical performance estimates. In addition, an expanded variability testwork program to develop geometallurgical models based on mineral composition. Mineral assemblages will need to be related back to ‘proxy’ ICP assays so that block models can be populated with metallurgical performance estimates.

13.7 Concluding Remarks

It is the QP author’s opinion that the test samples selected are representative of the various mineralized zones and rock types and any composites tested represent a reasonable period of operation in the LOM plan.

In addition, there are no processing factors or deleterious elements present in the final product that could have a significant impact on potential economic extraction.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource model was prepared by Skeena and was independently validated and signed off by SRK. The resource model is based on 7,583 historical holes and 751 completed holes drilled by Skeena from 2018 to January 2021. The updated 2021 Mineral Resource estimate has a majority component of Mineral Resources potentially amenable to open pit mining methods. The resource estimation work was completed by Ms. K. Dilworth and was reviewed and accepted by Ms. S. Ulansky, P.Geo (EGBC#36085), Senior Resource Geologist with SRK. The effective date of the Mineral Resource estimate is April 7, 2021.

This section describes the resource estimation methodology and summarizes the key assumptions. The Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and reported using the 2014 CIM Definition Standards. Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The database used to estimate the Mineral Resources was audited by SRK. The QP is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold and silver mineralization and that the assay data are sufficiently reliable to support Mineral Resource estimation.

Leapfrog Geo (version 6.0) was used to update the litho-structural model and mineralization domains that define the 2021 Eskay Creek model. Snowden Supervisor (version 8.13) was used to conduct geostatistical analyses, variography, and a portion of model validation. For block modelling, Maptek Vulcan (version 2021.1) software was used to prepare assay data for geostatistical analysis, modify mineralization domains, construct the block model, estimate metal grades and to tabulate the Mineral Resources.

14.2 Resource Estimation Procedures

The estimation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the litho-structural model;
- Construction of wireframe models for gold–silver mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;

- Resource validation;
- Resource classification;
- Assessment of "reasonable prospects for eventual economic extraction" and selection of appropriate cut-off grades;
- Preparation of the Mineral Resource statement.

14.3 Resource Database

The Eskay Creek database that supports the resource estimate contains 8,334 drill holes totalling 756,073 m. This includes 7,583 historical drill holes within the extents of the resource estimate, for a total of 6,061 underground drill holes and 1,522 surface drill holes (Table 14-1). An additional 751 surface core drill holes were completed by Skeena from 2018 to the end of the Phase 2 program in early 2021 totalling 104,740 m (Table 14-2). The close out date for the database was March 9, 2021.

Table 14-1: Historical Drill Holes

Year	No. of Holes	Length (m)	Assays
Pre-2018	7,583	651,332	427,200

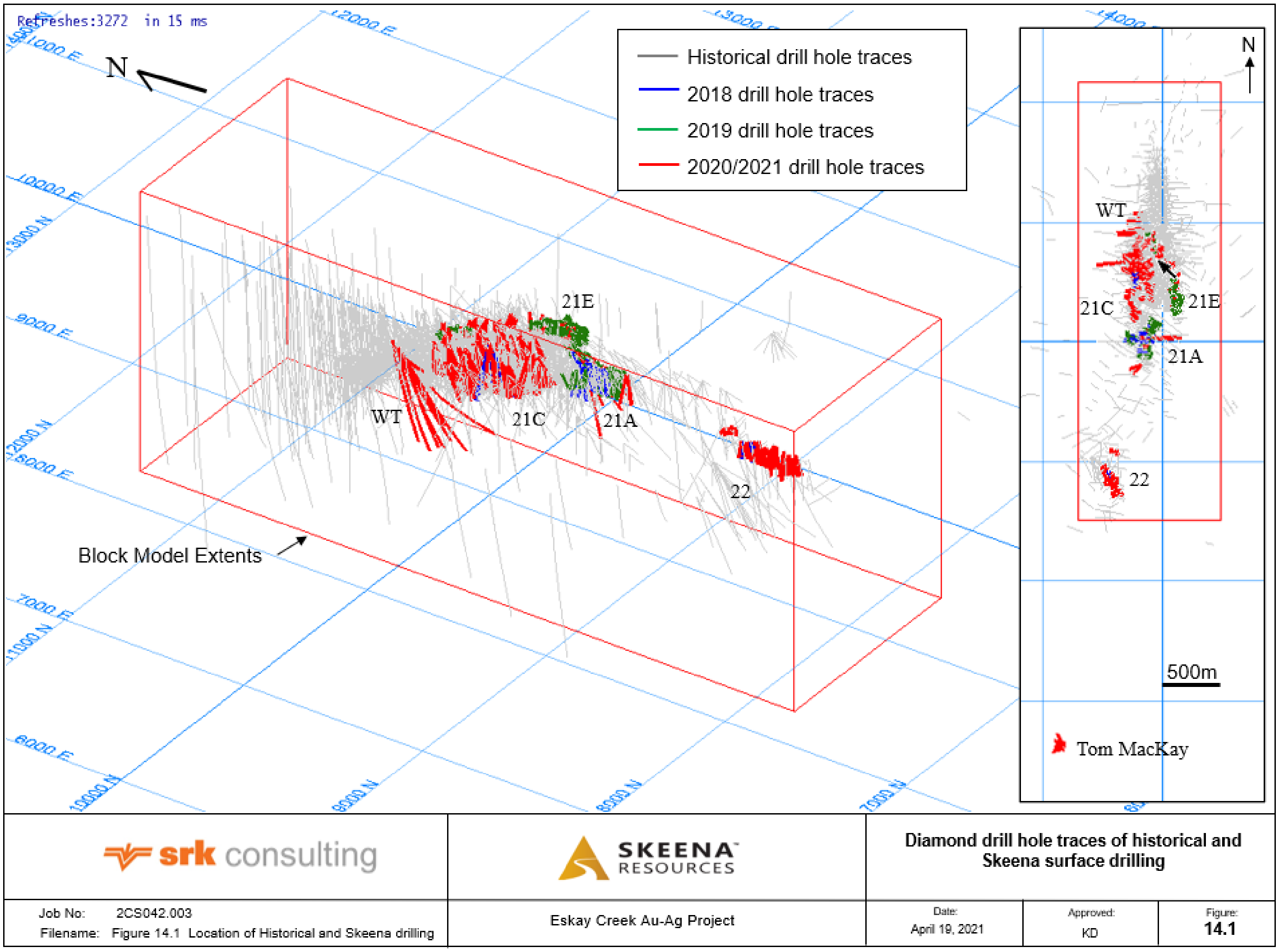
Table 14-2: Skeena Drill Holes

Year	No. of Holes	Length	Assays
2018	46	7,737.45	3,315
2019	203	14,091.87	8,593
2020 Phase 1	197	36,582.45	16,593
2020/21 Phase 2	305	46,328.23	19,184
TOTAL	**751**	**104,740**	**47,685**

Drill hole spacing throughout the orebody varies from 5 m, where underground production drilling encountered complex areas, to 25 m between surface drill holes. The average drill hole spacing is approximately 10–15 m throughout the deposit. Historically, sampling at Eskay Creek was selective and primarily based on visual estimations of sulphide percent. All sample intervals sent to the laboratory were tested for gold and silver, however, lead, copper, zinc, mercury, antimony, and arsenic were inconsistently sampled from one drilling campaign to the next. For underground drilling, lead, copper, zinc, mercury, antimony, and arsenic were assayed when samples exceeded 8 g/t gold equivalent (AuEQ; where AuEQ equalled Au+(Ag/68)) (Barrick, 2005).

Figure 14-1 shows the traces of all drill holes in the historical database as well as the traces of surface drilling completed by Skeena from 2018 to 2021.

Figure 14-1: Oblique View (left) and Plan View (right) of the Historical and Skeena Drill Holes

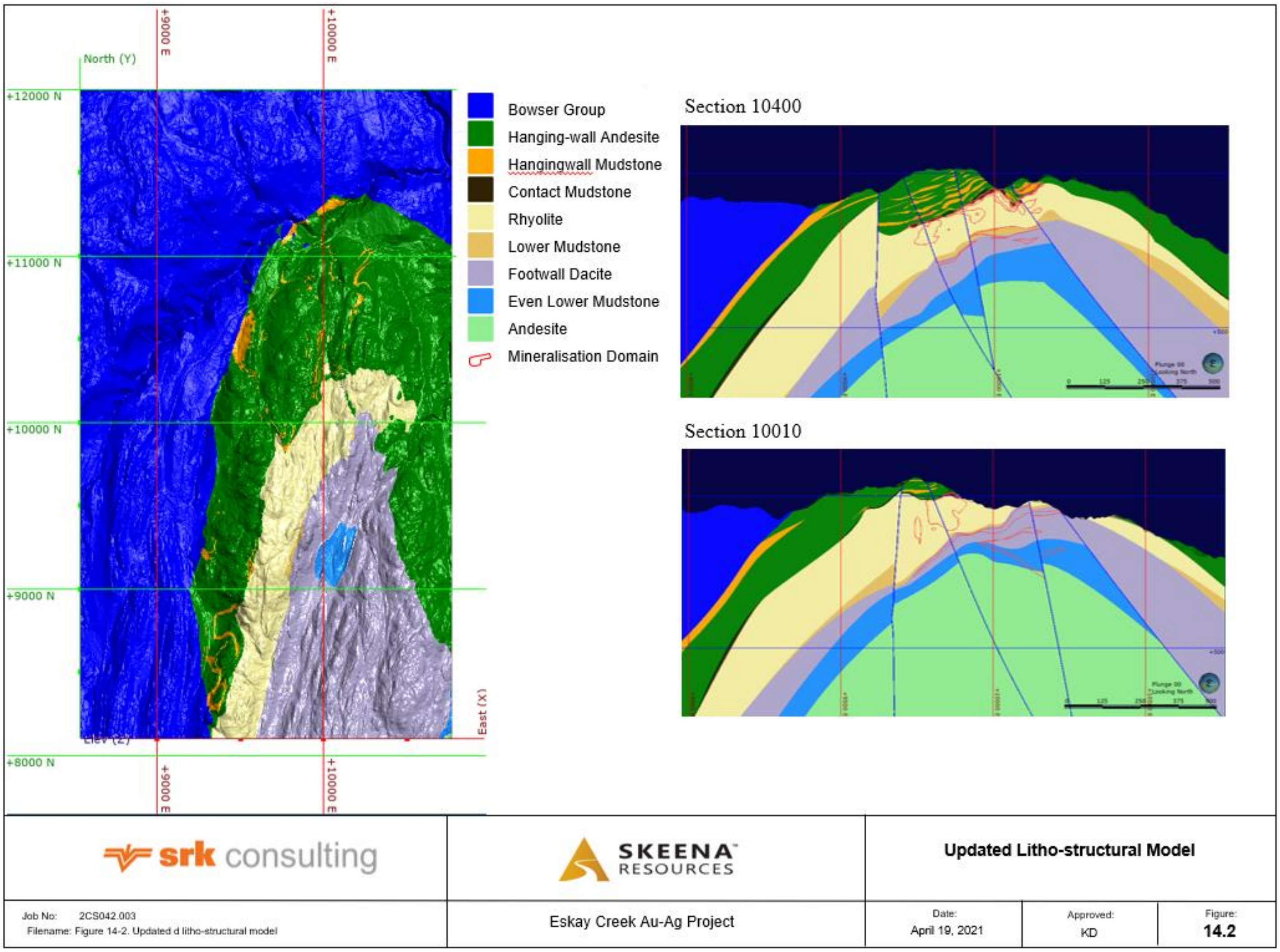


14.4 Solid Body Modelling

14.4.1 3D Litho-Structural Model

During 2020, the litho-structural model was updated to include six additional lithological units that were previously merged within the nearest stratigraphic package, namely, (1) the mudstone in the overlying Hanging Wall Andesite (Hanging Wall Mudstone), (2) two footwall sediment units (Lower Mudstone and Even Lower Mudstone), (3) extrusive units below the Rhyolite (Dacite and Footwall Andesite) and (4) the Bowser Group sediments. The structural model that was created in 2018 by Dr. Ron Uken, a Principal Structural Geologist with SRK, was used. (Figure 14-2).

Figure 14-2: Simplified Litho-Structural Model



14.4.2 Mineralization Domaining

The mineralization domain modelling undertaken for the 2021 estimate was updated and improved from the 2019 estimate. In total, 90 solids were created for the 2021 estimate including 84 mineralization solids, five low-grade envelope solids, and one solid used to restrict the influence of high-grade, mined out material.

14.4.2.1 Mineralization Domains

Eighty-four mineralization solids were created using LeapFrog Geo and Vulcan to constrain the mineralization. The domains were designed by lithology type, structural trends, and AuEQ assay intervals with a nominal cut-off of 0.5 g/t AuEQ or greater (where AuEQ = Au + Ag/74). Occasionally, lower-grade intersections were included to maintain continuity.

Three modelling methods were used:

1. Radial basis function (RFB) indicator interpolants for the Contact Mudstones (the RBF is an estimator that models known data positions and can provide an estimate for any unknown points):
   - Drill holes were composited to 1 m, with left over samples at the end of the holes appended to the previous sample;
   - A 50% probability was applied;
   - A structural trend was used as the search orientation;
2. Interval selection for all other lithologies:
   - A nominal cut-off grade of 0.5 g/t AuEQ was used to select assays intervals directly from the assay database;
   - Domains were created using either the vein or intrusion tool;
3. Manual wireframing created in Vulcan:
   - Two small solids in the Water Tower Zone were manually wireframed in Vulcan software.

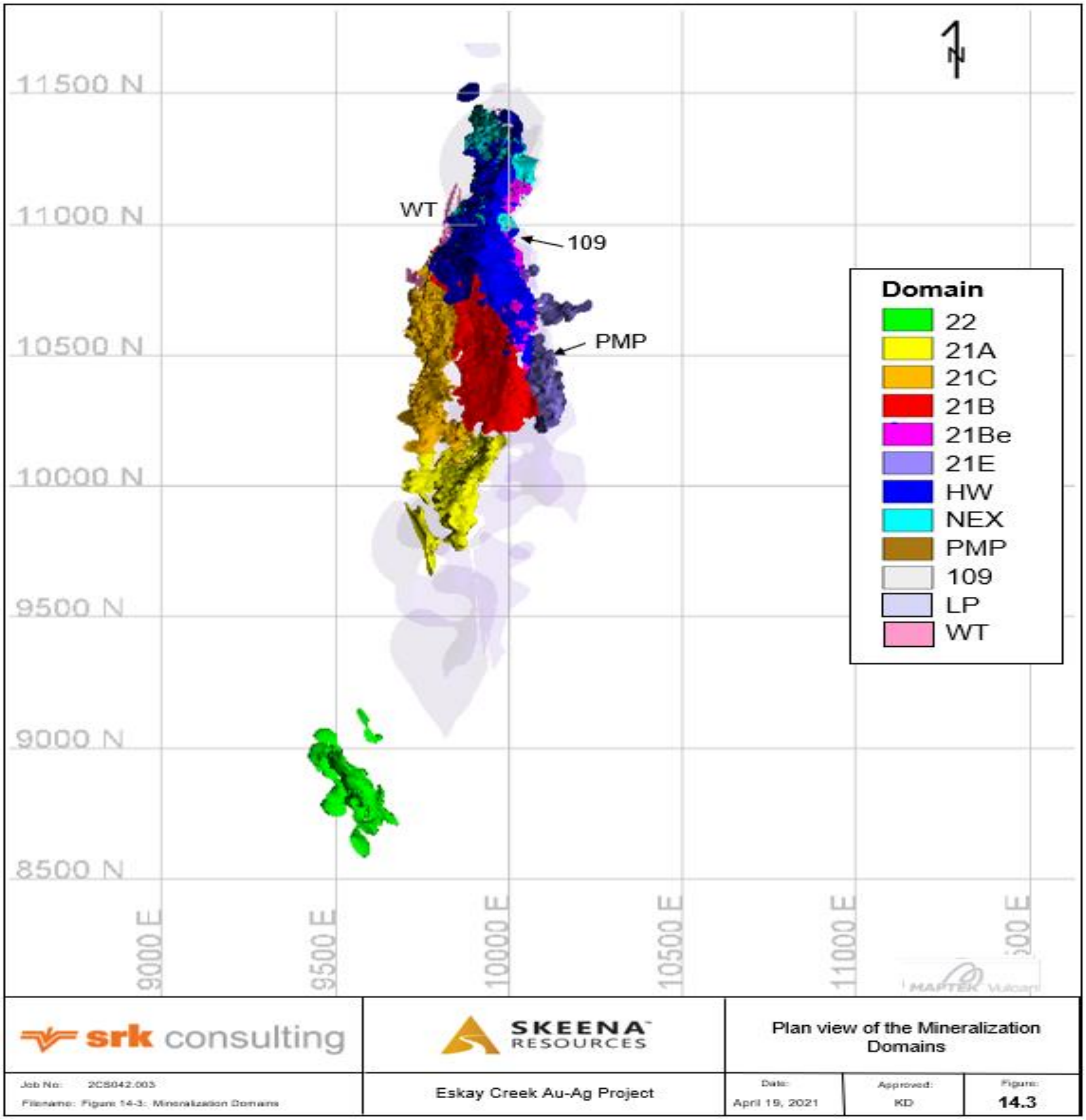
The subsequent wireframes were reviewed in section and level plan view by the QP and considered to be representative of the underlying geology.

The resulting mineralization solids in 2021 differ from the 2019 solids due to the following changes:

- In the 2019 estimate, only three lithologies were modelled: Rhyolite, Contact Mudstone and Hanging Wall Andesite. In the 2021 model, the stratigraphic package was refined to include the Hanging Wall and footwall sediments, the extrusive units below the Rhyolite and the sediments of the Bowser Group;
- In the 2019 estimate, a 0.5 g/t AuEQ indicator interpolant was used for all mineralization domains using Rhyolite and Contact Mudstone/Hanging Wall Andesite combined lithologies, whereas in the 2021 model the indicator interpolant was only used for the Contact Mudstone;
- The Mineralization in the Lower Package was modelled. This domain sits below the previously mined domains, in the Lower Mudstones (equivalent to the Datum Mudstone), Dacite, and less commonly in the Even Lower Mudstones (equivalent to the Spatsizi Mudstone) and Footwall Andesite.

For consistency, the mineralization domain solids were split and/or combined and named according to their location within the previously-established historical mining area zones: 22, 21A, 21C, 21B, 21Be, 21E, HW, NEX, WT, 109 and PMP (as shown in Figure 14-3). For the purposes of this Report, "domain(s)" refer to mineralization solid(s) within the historically-defined mining area zones. Mineralization in the recently-defined Lower Package lithologies cannot be equated with historically-defined mining zones, since they were not defined until 2021.

Figure 14-3: 2021 Mineralization Domains at the Eskay Creek Project



14.4.2.2 Low-Grade Envelope Domain

In addition to the drill hole intervals contained within the mineralized domains, a significant number of drill hole intervals with grades > 0.5 g/t AuEQ were not modelled within a mineralization domain wireframe. A separate low-grade envelope was created around these intervals and subdivided into five domains based on litho-structural fault block groupings.

Figure 14-4 shows the low-grade envelope in relation to the 2 m composite assay grades >0.7 g/t AuEQ outside the mineralization domain boundaries.

14.4.2.3 3 m Restriction Domain

Due to the high-grade nature of the mined-out areas at Eskay Creek, a 3 m solid around the mined-out stopes and lifts was created. All composites within this area were limited in range and were not allowed to influence blocks outside of this 3 m domain. This was done to limit the smearing effect of the high-grade samples into the remaining resource areas.

Figure 14-5 is a representation of the 21B Domain showing the Contact Mudstone, Rhyolite and 3 m restriction domain used for estimation.

14.4.2.4 Solid Model Coding

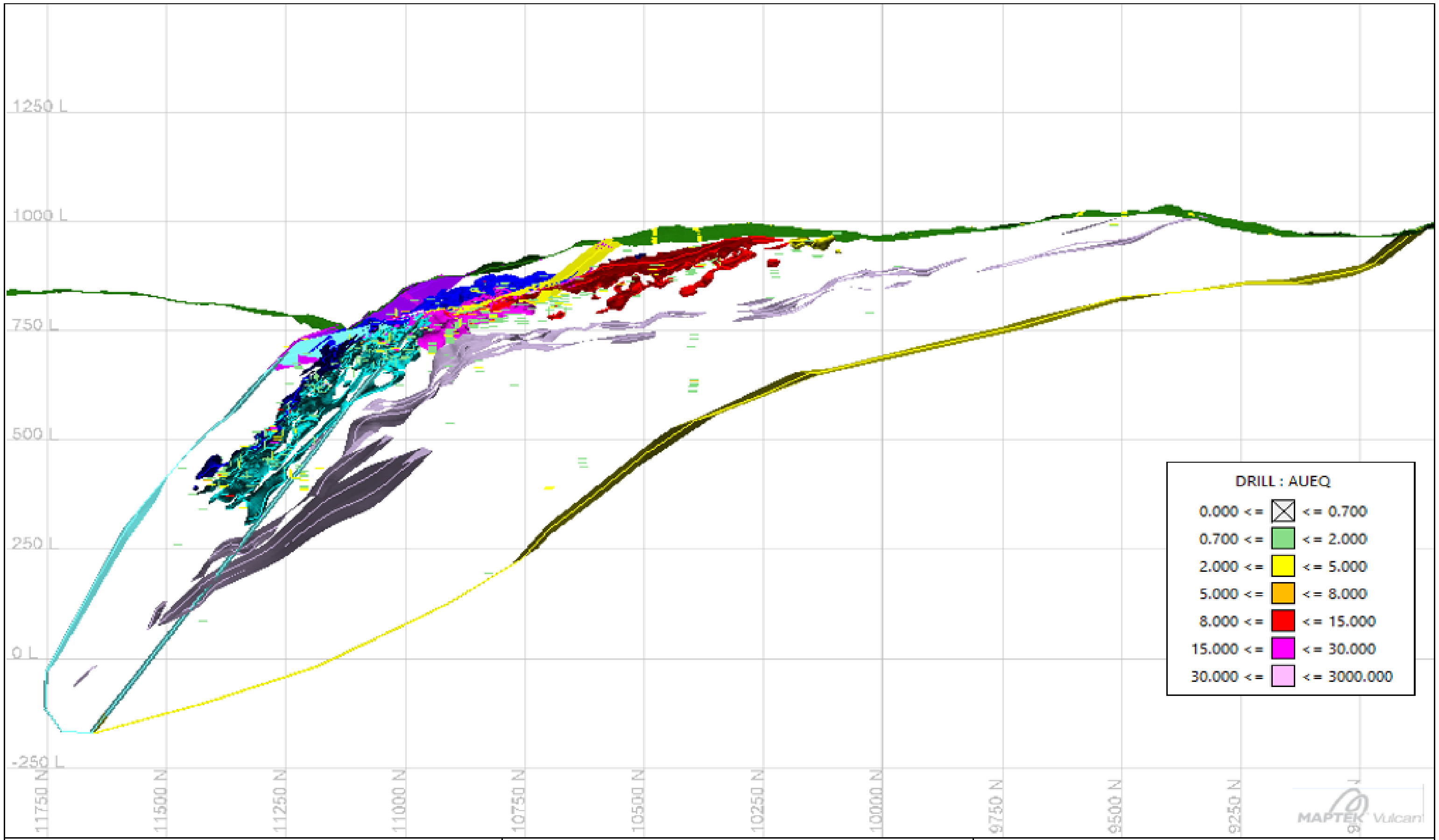
Estimation domains were coded successively based on the following division scheme: (1) location within the historical mining area, (2) dominant lithology type, (3) position within the litho-structural domain and, (4) location within the 3 m high grade restriction zone.

Table 14-3 summarizes the coding scheme used.

14.4.3 Topography

The topography surface was created using a 10 cm resolution from the LiDAR survey.

Figure 14-4: Low-Grade Envelope Domain with 2 m Composites greater than 0. 7 g/t AuEQ Located Outside of the Mineralized Domains



		2 m composites greater than 0.7 AuEQ g/t in the Low-Grade Envelope Section 9975 E looking East +/- 40 m		
Job No: 2CS042.003 Filename: Figure 14.4 2m composites in the LG Envelope	Eskay Creek Au-Ag Project	Date: April 19, 2021	Approved: KD	Figure: **14.4**

Figure 14-5: Three-Metre Restriction Domain Used to Constrain the High-Grade, Minded-out Material in the 21B Domain

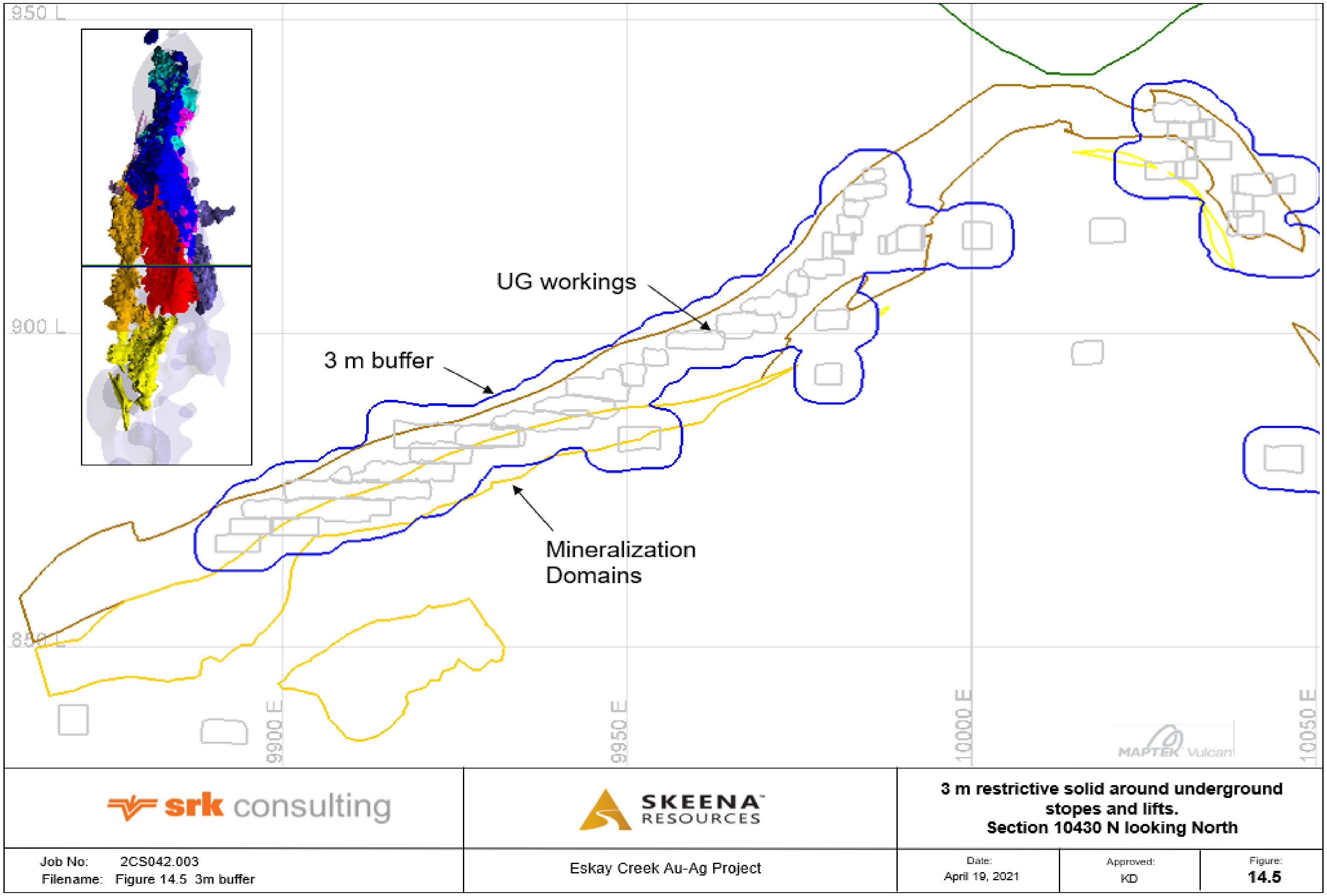


Table 14-3: **Mineralization Coding Summary**

Domain Name	Domain	Rocktype	Zone	Litho-Structural Domain	No. of solids	Est_Zone (outside 3m restriction)	Est_Zone (within 3m restriction)
LG ENV	1	Variable	1		1	1	
			2		1	2	
			3		1	3	
			4		1	4	
			5		1	5	
22	10	Rhyolite	10		3	1011, 1012, 1013	
21A	20	Rhyolite	201		3	2011, 2012, 2013	
		Contact Mudstone	202		1	2021	
		Hanging-wall sediments	203		1	2031	
21C	30	Rhyolite	301		4	3011, 3012, 3013 3014	93011, 93012, 93013, 93014
		Contact Mudstone	302		2	3021, 3022	93021, 93022
		Hanging-wall sediments	303		7	3031, 3032, 3033, 3034, 3035, 3036, 3037	93031, 93032, 93033, 93034, 93035, 93036, 93037
21B	40	Rhyolite	401		7	4012, 4013, 4014, 4015, 4016, 4017	94012, 94013, 94014, 94015, 94016, 94017
		Contact Mudstone	402		2	4021, 4022	94021, 94022
21Be	50	Rhyolite	501		2	5011, 5012	95011, 95012
		Contact Mudstone	502		1	5021	95021
21E	60	Rhyolite	601		4	6011, 6012, 6013, 6010	96011, 96012, 96013, 96010
		Contact Mudstone	602		3	6021, 6022, 6043	96021, 96022, 96043
		Hanging-wall sediments	603		7	6031, 6032, 6033, 6034, 6035, 6041, 6042	96031, 96032, 96033, 96034,96035, 96041, 96042
HW	70	Hanging-wall Sediments	703	3	15	70381, 70382, 70383, 70384, 70386	970381, 970382, 970383, 970384, 970386
				5		70351, 70352, 70353, 70354, 70355, 70356, 70357	970351, 970352, 970353, 970354, 970355, 970356, 970357
				4		70341, 70343, 70342	970341, 970343, 790342

Domain Name	Domain	Rocktype	Zone	Litho-Structural Domain	No. of solids	Est_Zone (outside 3m restriction)	Est_Zone (within 3m restriction)
NEX	80	Rhyolite	801		4	8011, 0812, 8013, 8014	98011, 98012, 98013, 98014
		Contact Mudstone	802		2	8021, 8022	98021, 98022
WTZ	81	Rhyolite	811		6	8111, 8112, 8113, 8114, 8115, 8116	
LP	90	Dacite	90		2	901, 902	
		Even Lower Mudstone	91		1	903	
		Footwall Andesite	92		1	904	
		Lower Mudstone	93		1	905	
		Lower Rhyolite	94		1	906	
PMP	95	Rhyolite	95		3	951, 952, 953	9951
109	99	Rhyolite	99		1	99	999

14.4.4 Underground Workings

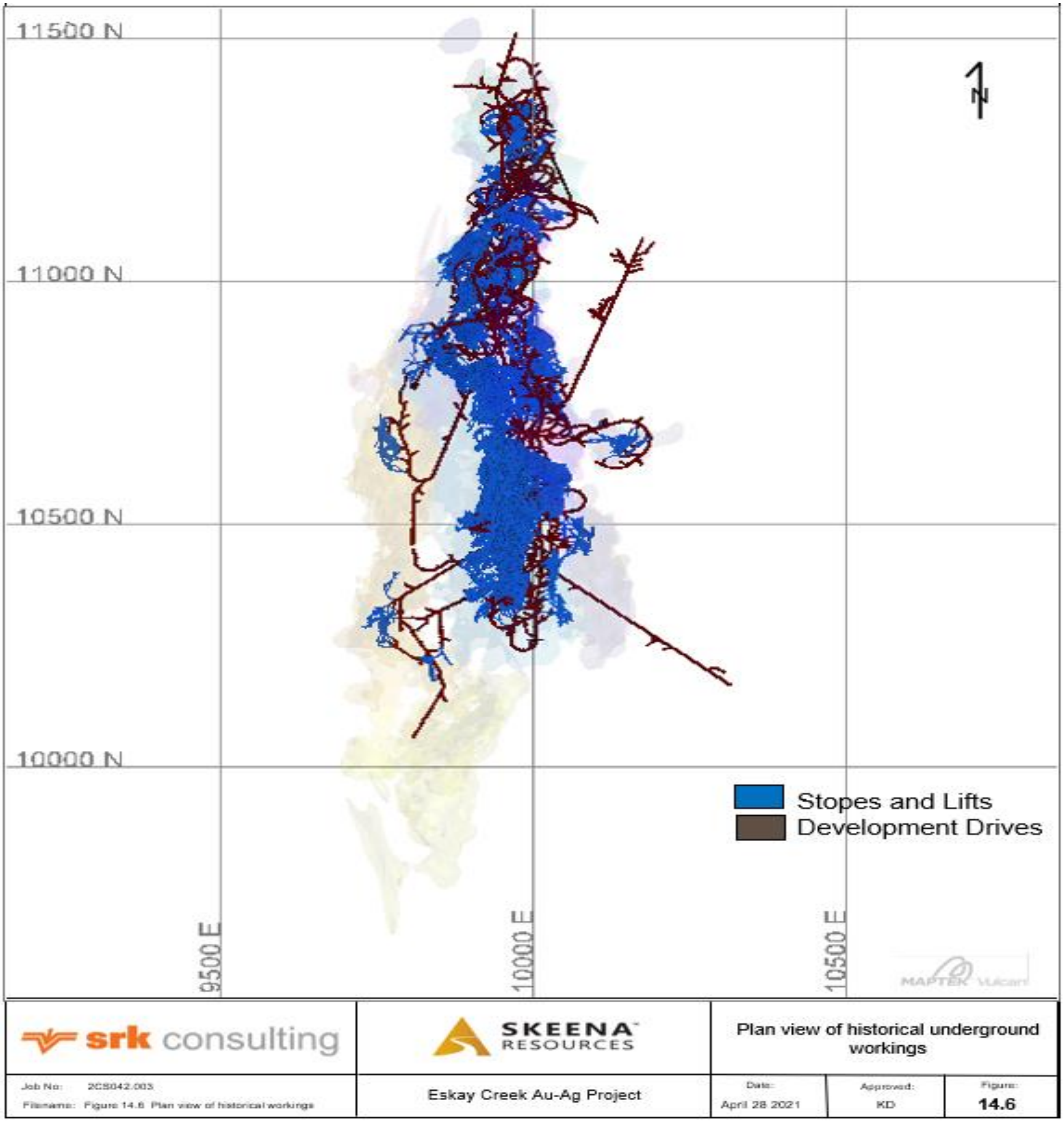
The historical underground workings are a combination of stopes, lifts, and development drives. The previous operator reported that the lifts and the stopes were backfilled with cobble, where cobble was made at the site in a batch cement plant that consisted of screened gravel from the Iskut River supplemented with 4–12% cement (Barrick, 2005).

Skeena checked the location of the underground drill holes in relation to the underground working solids and found no obvious spatial errors. Although the underground workings were routinely surveyed, there is a small measure of uncertainty in the location of the solids due to survey method limitations. Therefore, in addition to the volume within the underground workings, a 0.20 m geotechnical exclusion zone around the underground workings was used to deplete the final resource estimate assuming open pit mining methods. For the underground model, a 1 m geotechnical exclusion zone around all underground workings was used to deplete the resources potentially amenable to underground mining methods. Figure 14-6 and Figure 14-7 show the underground workings used to deplete the current estimate in plan view and long section, respectively.

14.5 Data Analysis

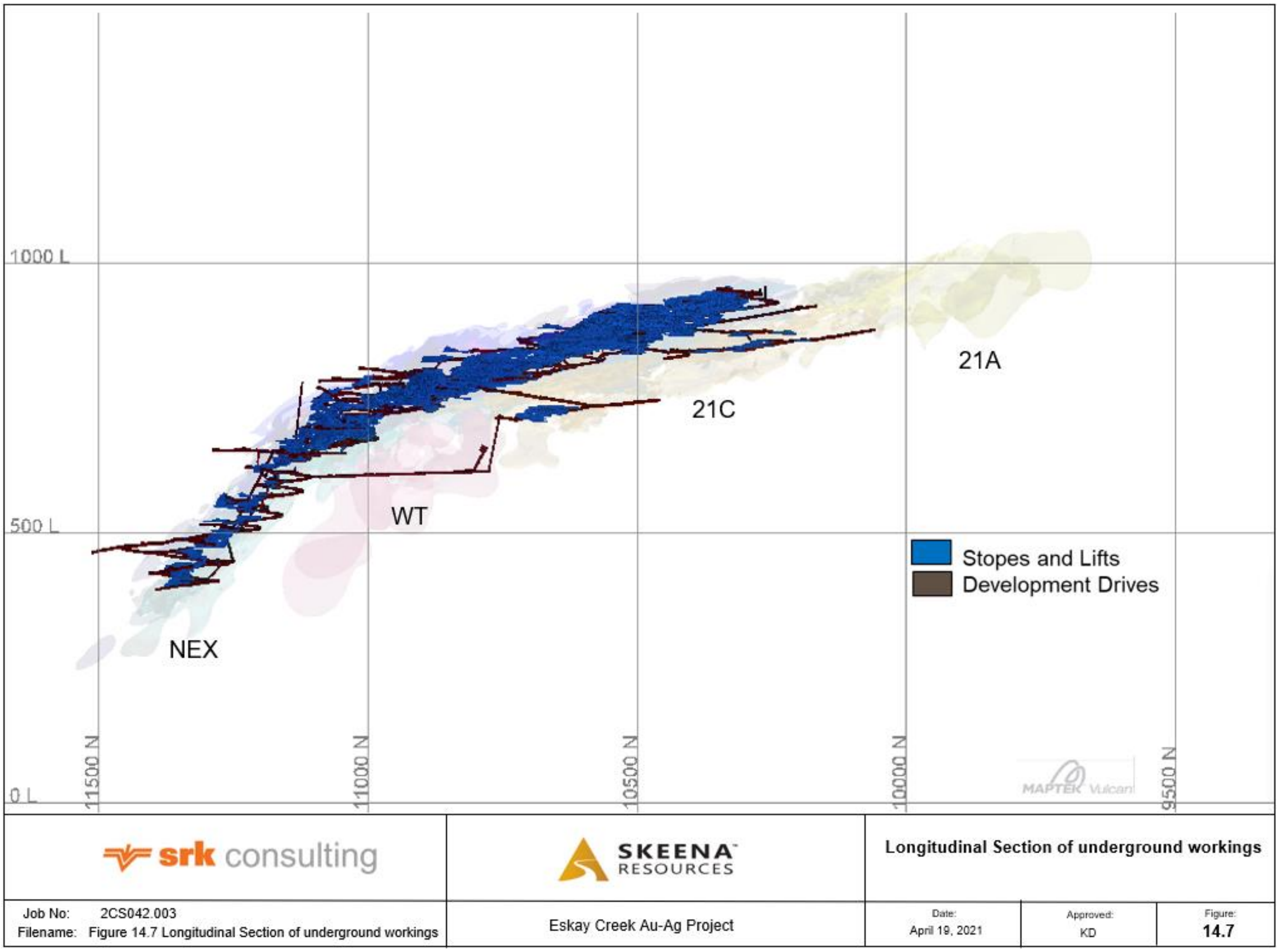
The ZONE item was used to code the assay file in the database for geostatistical analysis, as this split the domain into the main lithology groupings (Table 14-3). These coded intercepts were used to analyse sample length and generate statistics for assays and composites. Table 14-4 summarizes the statistical analysis of original assays for gold and silver. In addition to gold and silver, the contents of the following additional elements were calculated as part of the resource process: lead, copper, zinc, mercury, arsenic, ion and sulphur. The additional elements are important for optimizing ore mining economics, processing options, and saleable product routes, as smelter penalties may apply based on their relative content. Details of these elements are discussed in Section 14.7.

Figure 14-6: Plan View of Historical Underground Mine Workings



Note: for simplicity the Lower Package is not shown.

Figure 14-7: Long Section of the Historical Underground Mine Workings



Note: Looking east with the 1 m buffer applied; domain wireframes are shown for reference; for simplicity, the lower package domain is not shown.

Table 14-4: Summary Statistics Drill Hole Gold and Silver Assays by Zone

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
GOLD g/t								
ENV	1	Rhyolite	10,824	0.1	4.0	0.0	0.0	13.7
ENV	2	-	148,359	0.3	23.3	0.0	0.1	1,193.3
ENV	3	-	43,051	0.3	18.4	0.0	0.1	986.8
ENV	4	-	19,119	0.4	13.4	0.0	0.1	303.8
ENV	5	-	20,322	0.2	19.1	0.0	0.1	352.1
22 Zone	101	Rhyolite	4,351	1.6	3.2	0.0	0.7	225.6
21A	201	Rhyolite	9,756	2.7	2.7	0.0	0.9	216.3
	202	Contact Mudstone	1,103	19.6	2.1	0.0	3.4	677.8
	203	Hanging Wall Sediments	5	0.9	0.6	0.1	0.9	1.4
21C	301	Rhyolite	29,787	4.0	2.8	0.0	1.6	937.0
	302	Contact Mudstone	5,730	4.0	10.6	0.0	1.2	1,774.4
	303	Hanging Wall Sediments	1,533	4.0	2.0	0.0	1.6	122.1
21B	401	Rhyolite	21,723	5.2	5.7	0.0	1.3	1,652.4
	402	Contact Mudstone	16,710	29.6	4.0	0.0	3.4	9,659.0
21Be	501	Rhyolite	19,714	9.2	5.3	0.0	2.2	1,621.9
	502	Contact Mudstone	8,505	18.5	4.2	0.0	2.1	2,072.7
21E	601	Rhyolite	1,762	1.6	1.5	0.0	0.9	41.8
	602	Contact Mudstone	1,110	7.3	2.9	0.0	1.5	450.6
	603	Hanging Wall Sediments	1,633	2.3	2.2	0.0	1.2	111.7
HW	703	Hanging Wall Sediments	16,612	5.2	2.9	0.0	1.6	504.4
NEX	801	Rhyolite	26,883	4.3	6.4	0.0	1.4	1,380.4
	802	Contact Mudstone	24,522	7.9	5.7	0.0	1.8	1,971.1
WT	811	Rhyolite	2,989	2.9	1.9	0.0	1.3	92.8
LP	90	Dacite	4,518	0.9	3.6	0.0	0.5	190.3
	91	Even Lower Mudstone	480	0.9	3.1	0.0	0.4	39.9
	92	Andesite	186	0.9	3.9	0.0	0.3	44.6
	93	Lower Mudstone	553	3.7	16.0	0.0	0.6	1,380.0
	94	Rhyolite	1,182	0.9	2.3	0.0	0.6	55.9
PMP	95	Rhyolite	2,868	6.8	3.7	0.0	2.8	704.8
109	99	Rhyolite	13,419	10.8	4.0	0.0	2.8	1,625.8
SILVER g/t								
ENV	1	Rhyolite	10,824	1.6	4.5	0.01	0.5	347
ENV	2	-	148,354	7.8	32.3	0.01	0.5	47,619

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
ENV	3	-	42,767	12.4	25.8	0.01	0.5	29,222
ENV	4	-	19,118	25.2	25.5	0.05	0.5	44,437
ENV	5	-	20,322	5.8	18.2	0.01	0.5	11,420
22 Zone	101	Rhyolite	4,351	48.1	3.1	0.05	6.0	3,461
21A	201	Rhyolite	9,756	51.4	4.0	0.05	5.0	7,190
	202	Contact Mudstone	1,103	219.7	5.3	0.05	6.0	22,353
	203	Hanging Wall Sediments	5	0.7	0.6	0.05	0.8	1
21C	301	Rhyolite	29,788	46.9	6.9	0.05	0.5	28,419
	302	Contact Mudstone	5,188	107.8	5.9	0.05	11.0	36,696
	303	Hanging-wall Sediments	1,533	251.3	3.0	0.05	17.5	8,174
21B	401	Rhyolite	21,723	260.9	5.8	0.01	3.0	44,767
	402	Contact Mudstone	16,710	1138.7	2.8	0.05	35.0	43,658
21Be	501	Rhyolite	19,713	486.6	6.2	0.05	30.0	155,086
	502	Contact Mudstone	8,505	955.0	3.8	0.05	26.0	54,899
21E	601	Rhyolite	1,762	62.3	3.5	0.05	8.0	4,470
	602	Contact Mudstone	1,110	291.4	4.0	0.05	12.0	17,274
	603	Hanging Wall Sediments	1,633	94.2	5.3	0.05	23.0	10,724
HW	703	Hanging Wall Sediments	16,612	261.9	4.0	0.05	22.0	28,093
NEX	801	Rhyolite	26,876	140.3	9.1	0.05	0.5	55,510
	802	Contact Mudstone	24,522	366.6	6.4	0.05	14.0	59,545
WT	811	Rhyolite	2,989	22.0	4.7	0.05	0.5	2,524
LP	90	Dacite	4,511	8.8	2.0	0.05	4.0	470
	91	Even Lower Mudstone	480	8.8	1.3	0.05	6.0	125
	92	Footwall Andesite	178	3.7	1.2	0.50	2.5	32
	93	Lower Mudstone	553	15.5	2.0	0.05	7.0	365
	94	Rhyolite	1,182	10.9	2.9	0.05	3.0	720
PMP	95	Rhyolite	2,868	178.9	4.8	0.05	22.0	23,117
109	99	Rhyolite	13,418	16.1	6.1	0.05	0.5	4,457

14.6 Compositing

To minimize bias introduced by variable sample lengths, assays were composited from assays honouring the relevant mineralization domain boundaries to 2 m lengths for the open pit model, and 1 m lengths for the underground model. Most samples inside the mineralization domains were collected at approximately 1 m and shorter intervals. All unsampled gold and silver intervals were given a default value of 0.001 g/t during compositing. Missing samples due to lost core, voids or insufficient sample were ignored. Composite lengths that fell short were evenly distributed. The composites were assigned codes on a majority basis corresponding to the mineralized domain, zone, and estimation zone in which they occur. The compositing and coding processes were viewed in 3D to ensure that coding had been applied correctly.

14.6.1 2 m Composites

A total of 109,816 two-metre composites were coded into mineralization domains, not including composites within the low-grade envelope. Summary statistics between the assays and 2 m composites are shown in Table 14-5.

Table 14-5: Comparison of Assay Data to 2 m composites

		Assays				2 m Composites			
Domain Name	Zone	No. of Samples	Maximum	Mean	CV	No. of Samples	Maximum	Mean	CV
				GOLD g/t					
ENV	1	10,824	13.7	0.1	3.948	10,235	8.5	0.1	4.0
	2	148,359	1193.3	0.3	23.288	1,393,934	808.5	0.1	20.7
	3	43,051	986.8	0.3	18.424	40,097	249.3	0.5	12.5
	4	19,119	303.8	0.4	13.415	13,179	218.6	0.3	12.8
	5	20,322	352.1	0.2	19.067	19,865	239.9	0.1	19.6
22 Zone	101	4,351	225.6	1.6	3.192	2,879	75.5	1.5	2.1
21A	201	9,756	216.3	2.7	2.739	5,962	143.9	2.5	2.4
	202	1,103	677.8	19.6	2.065	607	301.2	16.2	1.8
	203	5	1.4	0.9	0.64	3	1.0	0.8	0.2
21C	301	29,787	937.0	4.0	2.796	14,790	261.7	3.7	2.0
	302	5,730	1774.4	4.0	10.583	2,789	808.3	3.4	7.2
	303	1,533	122.1	4.0	2.04	781	105.6	3.3	1.9
21B	401	21,723	1652.4	5.2	5.68	11,237	1048.8	5.0	5.4
	402	16,710	9659.0	29.6	4.04	8,305	2866.4	27.0	2.8
21Be	501	19,714	1621.9	9.2	5.3	9,404	964.3	8.2	4.5
	502	8,505	2072.7	18.5	4.2	4,296	1253.6	16.1	3.6
21E	601	1,762	41.8	1.6	1.49	1,035	18.4	1.5	1.2
	602	1,110	450.6	7.3	2.88	540	227.1	6.3	2.6
	603	1,633	111.7	2.3	2.2	1,051	45.7	2.0	1.5
HW	703	16,612	504.4	5.2	2.92	8,266	326.4	4.5	2.6
NEX	801	26,883	1380.4	4.3	6.43	12,657	571.8	3.7	4.4

		Assays				2 m Composites			
Domain Name	Zone	No. of Samples	Maximum	Mean	CV	No. of Samples	Maximum	Mean	CV
	802	24,522	1971.1	7.9	5.73	11,877	1346.7	7.0	4.7
WT	811	2,989	92.8	2.9	1.86	1,528	45.0	2.6	1.5
LP	90	4,518	190.3	0.9	3.605	2,524	102.0	8.6	2.7
	91	480	39.9	0.9	3.132	268	19.2	0.8	2.2
	92	186	44.6	0.9	3.947	98	21.9	0.9	2.7
	93	553	1380.0	3.7	15.976	297	207.0	2.3	6.5
	94	1,182	55.9	0.9	2.294	629	27.0	0.9	1.6
PMP	95	2,868	704.8	6.8	3.069	1,439	415.9	6.0	2.5
109	99	13,419	1625.8	10.8	3.961	6,554	1063.2	9.9	3.1
SILVER g/t									
ENV	1	10,824	347	1.6	4.495	10,235	195	1.1	4.3
	2	148,354	47,619	7.8	32.299	139,934	25,072	3.9	35.6
	3	42,767	29,222	12.4	25.77	40,097	14,689	5.2	24.4
	4	19,118	44,437	25.2	25.45	13,179	25,127	14.3	24.5
	5	20,322	11,420	5.8	18.15	19,865	3,413	3.0	12.9
22 Zone	101	4,351	3,461	48.1	3.06	2,879	2,105	45.9	2.4
21A	201	9,756	7,190	51.4	3.96	5,962	5,288	48.1	3.2
	202	1,103	22,353	219.7	5.32	607	13,686	188.4	4.6
	203	5	1	0.7	0.6	3	1	0.7	0.4
21C	301	29,788	28,419	46.9	6.95	14,790	12,096	41.5	4.6
	302	5,188	36,696	107.8	5.91	2,789	7,808	86.9	3.3
	303	1,533	8,174	251.3	3.03	781	5,932	193.0	2.6
21B	401	21,723	44,767	260.9	5.79	11,237	26,928	218.8	5.4
	402	16,710	43,658	1138.7	2.84	8,305	33,184	1052.4	2.7
21Be	501	19,713	155,086	486.6	6.24	9,404	115,340	427.2	5.6
	502	8,505	54,899	955.0	3.83	4,296	44,646	804.7	3.5
21E	601	1,762	4,470	62.3	3.48	1,035	1,813	54.3	2.6
	602	1,110	17,274	291.4	3.96	540	5,597	232.9	2.9
	603	1,633	10,724	94.2	5.34	1,051	3,984	70.2	3.4
HW	703	16,612	28,093	261.9	3.99	8,266	19,852	228.1	3.4
NEX	801	26,876	55,510	140.3	9.07	12,657	39,787	109.6	8.2
	802	24,522	59,545	366.6	6.37	11,877	48,057	314.3	5.7
WT	811	2,989	2,524	22.0	4.67	1,528	1,170	19.2	3.6
LP	90	4,511	470	8.8	1.96	2,524	309	8.6	1.8
	91	480	125	8.8	1.33	268	77	8.3	1.2
	92	178	32	3.7	1.15	98	17	3.5	0.8
	93	553	365	15.5	1.98	297	216	14.1	1.5

		Assays				2 m Composites			
Domain Name	Zone	No. of Samples	Maximum	Mean	CV	No. of Samples	Maximum	Mean	CV
	94	1,182	720	10.9	2.85	629	390	10.6	2.3
PMP	95	2,868	23,117	178.9	4.82	1,439	15,837	158.5	4.2
109	99	13,418	4,457	16.1	6.13	6,554	2,746	14.3	5.1

14.6.2 1 m Composites

For the underground model, only domains that occurred below the conceptual open pit were estimated. A total of 79,596 one-metre composites were coded into five mineralization domains. Summary statistics between the assays and 1 m composites are shown in Table 14-6.

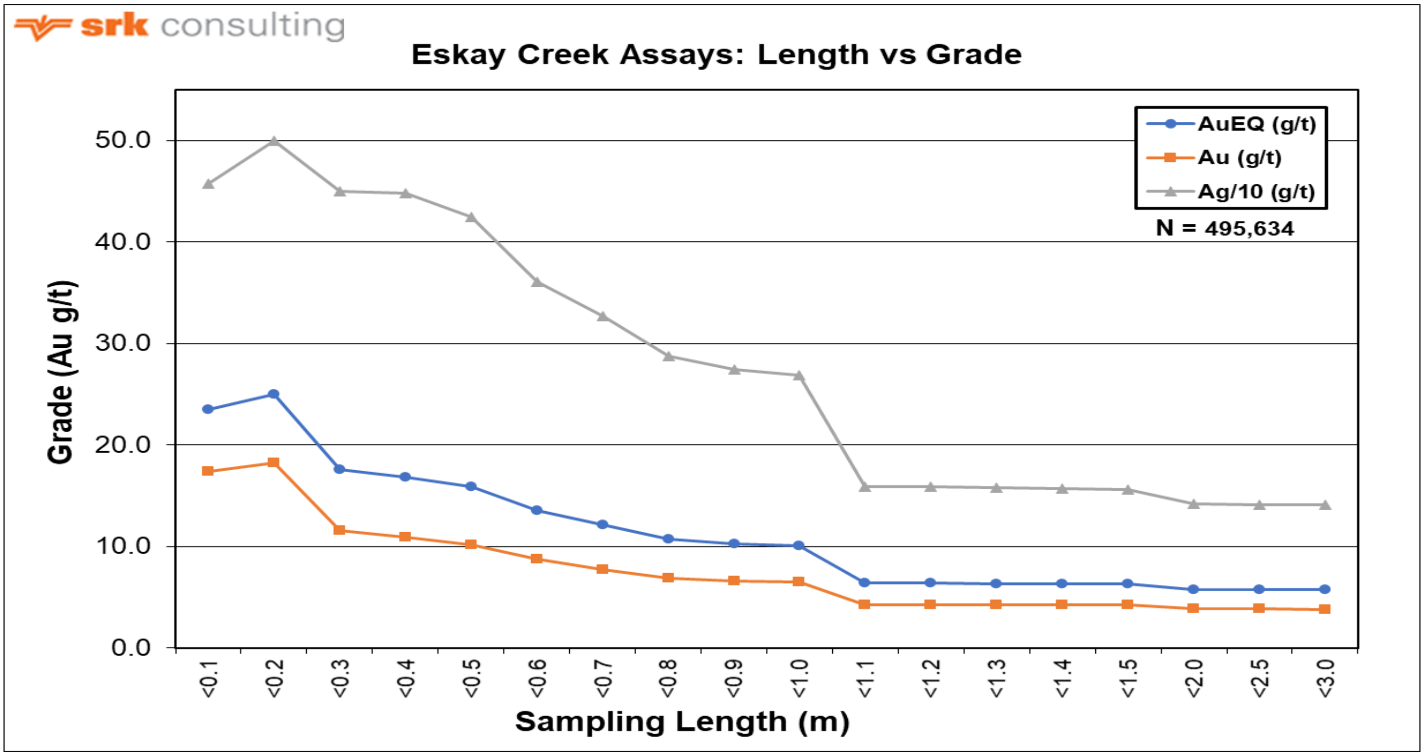
Table 14-6: Comparison of Assay Data to 1 m Composites

		Assays				1 m Composites			
Domain	Zone	No. of Samples	Maximum	Mean	CV	No. of Samples	Maximum	Mean	CV
GOLD g/t									
22 Zone	101	4,351	225.6	1.6	3.2	5,775	90.4	1.48	2.3
HW	703	16,612	504.4	5.2	2.9	15,999	472.2	4.63	2.8
NEX	801	26,883	1380.4	4.3	6.4	24,941	930.7	3.79	5.2
	802	24,522	1971.1	7.9	5.7	22,446	1,621.6	7.14	5.4
WT	811	2,989	92.8	2.9	1.9	2,966	63.5	2.72	1.7
LP	90	4,518	190.3	0.9	3.6	4,953	190.3	0.9	3.5
	91	480	39.9	0.9	3.1	529	30.7	0.91	2.9
	92	186	44.6	0.9	3.9	199	41.5	0.92	3.5
	93	553	1380.0	3.7	16.0	577	413.3	2.45	8.9
	94	1,182	55.9	0.9	2.3	1,201	55.9	0.92	2.3
SILVER g/t									
22 Zone	101	4,351	3,461	48.1	3.1	5,775	2,370	45.82	2.7
HW	703	16,612	28,093	261.9	4.0	15,999	24,960	234.39	3.7
NEX	801	26,876	55,510	140.3	9.1	24,941	41,007	116.28	8.7
	802	24,522	59,545	366.6	6.4	22,446	52,902	318.21	6.3
WTZ	811	2,989	2,524	22.0	4.7	2,966	1,566	19.77	4.2
LP	90	4,511	470	8.8	2.0	4,953	470	8.58	2.1
	91	480	125	8.8	1.3	529	125	8.39	1.3
	92	178	32	3.7	1.2	199	27	3.57	1.0
	93	553	365	15.5	2.0	577	358	14.72	1.8
	94	1,182	720	10.9	2.9	1,201	720	10.63	2.9

14.7 Evaluation of Outliers

Block grade estimates may be overly affected by very high-grade assays; therefore, capping was applied to all domains. An analysis of sample lengths versus gold grade shows that effort was taken to sample intervals based on visible mineralization, since gold grades are highest in the smallest assay lengths (Figure 14-8). For this reason, capping was applied after compositing. Capping values were selected on a zone-by-zone basis using the results from log probability plots, histograms, co-efficient of variation (CV) values, degradation plots, and percent metal loss.

Figure 14-8: Gold Grade versus Sample Length



Note: Figure prepared by SRK, 2021

14.7.1 2 m Composites

Percent metal loss was variable between zones, ranging from as little as 0.5% to as high as 47% for gold, and 0.4–17% for silver, in the main domains excluding the low-grade envelope (Table 14-7). For domains with percent metal loss >10%, the uncapped mean values were sensitive to extremely high-grade samples. On average,<7% gold and 6% silver were lost during the process of capping. Gold grades were capped more aggressively in the low-grade envelope.

Table 14-7: **Gold and Silver Assay Capped Grades per Zone**

Domain	Zone	# Samples	Cap Value	No. Cut	% Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
GOLD g/t										
ENV	1	10,235	2	24	0.2	0.05	4.0	0.05	3.1	6
ENV	2	1,393,934	4	402	0.0	0.14	20.7	0.10	20.7	26
ENV	3	40,097	6	113	0.3	0.16	12.5	0.11	4.2	27
ENV	4	13,179	9	40	0.3	0.26	12.8	0.17	4.0	33
ENV	5	19,865	3	48	0.2	0.10	19.6	0.07	3.4	27
22 Zone	101	2,879	30	5	0.2	1.48	2.1	1.44	1.7	3
21A	201	5,962	80	7	0.1	2.45	2.4	2.42	2.2	1
	202	607	130	4	0.7	16.24	1.8	15.80	1.7	3
	203	3	1	0	0.0	0.78	0.2	0.78	0.2	0
21C	301	14,790	90	10	0.1	3.66	2.0	3.62	1.8	1
	302	2,789	70	6	0.2	3.44	7.2	2.59	2.2	24
	303	781	33	8	1.0	3.27	1.9	3.15	1.6	4
21B	401	11,237	500	6	0.1	4.99	5.4	4.41	4.9	12
	402	8,305	650	8	0.1	26.98	2.8	26.64	2.5	1
21Be	501	9,404	500	12	0.1	8.24	4.5	8.07	4.2	2
	502	4,296	600	8	0.2	16.09	3.6	15.81	3.4	2
21E	601	1,035	10	8	0.8	1.53	1.2	1.51	1.1	2
	602	540	70	5	0.9	6.28	2.6	5.73	1.9	9
	603	1,051	16	8	0.8	1.96	1.5	1.88	1.2	4
HW	703	8,266	100	14	0.2	4.49	2.6	4.35	2.2	3
NEX	801	12,657	250	10	0.1	3.66	4.4	3.52	3.6	4
	802	11,877	450	13	0.1	6.99	4.7	6.77	4.1	3
WT	811	1,528	30	4	0.3	2.62	1.5	2.60	1.5	1
LP	90	2,524	7	10	0.4	0.89	2.7	0.82	1.0	8
	91	268	7	3	1.1	0.84	2.2	0.74	1.5	11
	92	98	7	3	3.1	0.91	2.7	0.74	1.7	19
	93	297	20	3	1.0	2.33	6.5	1.23	2.0	47
	94	629	7	2	0.3	0.89	1.6	0.84	1.0	6
PMP	95	1,439	100	4	0.3	6.00	2.5	5.73	1.7	5
109	99	6,554	500	4	0.1	9.93	3.1	9.80	2.8	1
SILVER g/t										
ENV	1	10,235	60	9	0.1	1.10	4.3	1.02	2.9	8
ENV	2	139,934	150	4	0.0	3.93	35.6	1.95	4.7	50
ENV	3	40,097	200	7	0.0	5.18	24.4	2.45	5.3	53
ENV	4	13,179	200	8	0.1	14.25	24.5	5.55	3.4	61
ENV	5	19,865	150	5	0.0	2.95	12.9	2.29	3.9	23
22 Zone	101	2,879	700	10	0.3	45.87	2.4	44.05	2.1	4
21A	201	5,962	1,900	6	0.1	48.05	3.2	47.30	3.0	2
	202	607	2,500	6	1.0	188.37	4.6	156.18	3.4	17
	203	3	-	0	0.0	0.68	0.4	0.68	0.4	1
21C	301	14,790	2,300	11	0.1	41.51	4.6	40.13	3.8	3
	302	2,789	2,300	5	0.2	86.89	3.3	83.51	2.7	4
	303	781	2,400	8	1.0	192.98	2.6	180.65	2.4	6
21B	401	11,237	15,000	13	0.1	218.78	5.4	213.66	5.1	2
	402	8,305	20,000	11	0.1	1052.43	2.7	1,046.11	2.6	1

Domain	Zone	# Samples	Cap Value	No. Cut	% Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
21Be	501	9,404	25,000	17	0.2	427.19	5.6	403.34	4.6	6
	502	4,296	20,000	16	0.4	804.73	3.5	772.90	3.2	4
21E	601	1,035	720	20	1.9	54.26	2.6	51.10	2.2	6
	602	540	3,000	9	1.7	232.92	2.9	213.54	2.7	8
	603	1,051	900	9	0.9	70.18	3.4	66.11	2.8	6
HW	703	8,266	8,000	14	0.2	228.09	3.4	220.00	3.0	4
NEX	801	12,657	10,000	15	0.1	109.64	8.2	101.88	6.5	7
	802	11,877	22,000	7	0.1	314.26	5.7	301.54	5.1	4
WT	811	1,528	500	7	0.5	19.15	3.6	17.93	3.0	6
LP	90	2,524	90	11	0.4	8.55	1.8	8.20	1.3	4
	91	268	25	14	5.2	8.33	1.2	8.06	1.0	3
	92	98	-	0	0.0	3.54	0.8	3.54	0.8	0
	93	297	60	8	2.7	14.11	1.5	12.89	1.1	9
	94	629	100	5	0.8	10.60	2.3	9.59	1.6	10
PMP	95	1,439	2,000	14	1.0	158.47	4.2	134.36	2.4	15
109	99	6,554	600	9	0.1	14.28	5.1	12.92	3.0	10

14.7.2 1 m Composite

For the underground model, 1 m composites were used. Statistics for the uncapped and capped 1 m composites are shown in Table 14-8. Percent metal loss was variable between zones, ranging from as little as 0.2% to as high as 52% for gold, and 2–16% for silver. For domains with percent metal loss >10%, the uncapped mean values were sensitive to extremely high-grade samples.

Table 14-8: Summary Statistics for 1 m Capped and Uncapped Composites by Zone

Domain	Zone	# Samples	Cap Value	No. Cut	% Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
GOLD g/t										
22 Zone	101	5,775	40	8	0.1	1.48	2.3	1.44	1.9	3
HW	703	15,999	180	13	0.1	4.63	2.8	4.55	2.5	2
NEX	801	24,941	200	46	0.2	3.79	5.2	3.47	3.4	8
	802	22,446	400	45	0.2	7.14	5.4	6.61	4.2	7
WTZ	811	2,966	28	18	0.6	2.72	1.7	2.65	1.5	3
LP	90	4,953	8	20	0.4	0.9	3.5	0.82	1.1	9
	91	529	4	13	2.5	0.91	2.9	0.65	1.1	29
	92	199	4	4	2.0	0.92	3.5	0.63	1.3	32
	93	577	18	6	1.0	2.45	8.9	1.17	1.9	52
	94	1,201	7	8	0.7	0.92	2.3	0.85	1.1	8
SILVER g/t										
22 Zone	101	5,775	1000	14	0.2	45.82	2.7	44.3	2.4	3
HW	703	15,999	15000	12	0.1	234.39	3.7	231.6	3.5	1
NEX	801	24,941	9500	41	0.2	116.28	8.7	97.35	5.8	16
	802	22,446	30000	19	0.1	318.21	6.3	308.6	5.8	3

Domain	Zone	# Samples	Cap Value	No. Cut	% Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
WTZ	811	2,966	550	13	0.4	19.77	4.2	17.61	3.2	11
LP	90	4,953	200	9	0.2	8.58	2.1	8.4	1.8	2
	91	529	40	11	2.1	8.39	1.3	7.96	1.1	5
	92	199	10	8	4.0	3.57	1.0	3.28	0.6	8
	93	577	130	5	0.9	14.72	1.8	13.87	1.4	6
	94	1,201	125	11	0.9	10.63	2.9	9.51	1.8	11

* % metal loss equals (mean – meanCap)/mean*100 where mean is the average grade of the assays before capping and meanCap is the average grade of assays after capping. Composites are not declustered.

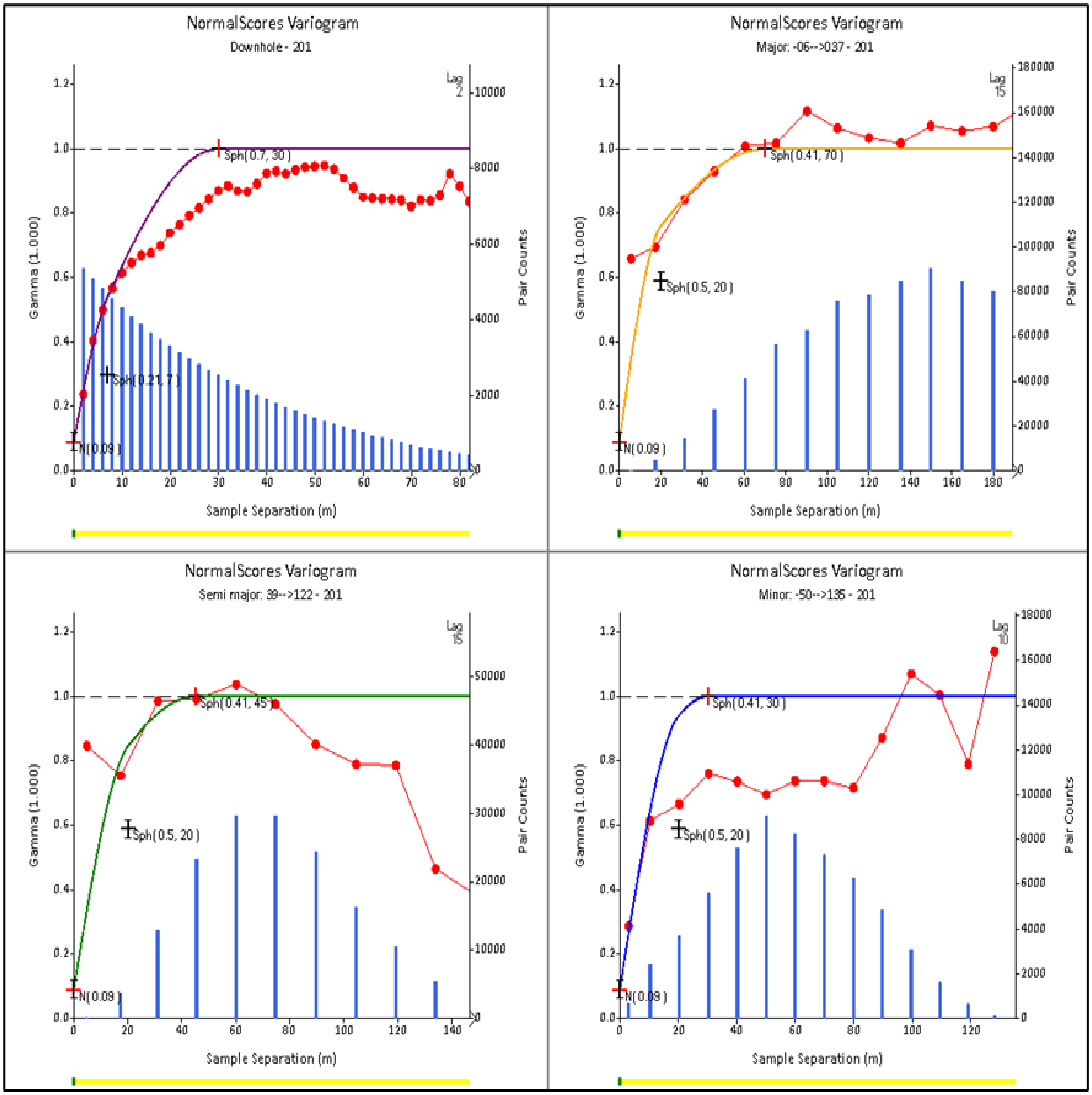
14.8 Variography

Variograms were used to assess for grade continuity, spatial variability in the estimation domains, sample search distances, and kriging parameters. Variograms were prepared using 2 m composites for the open pit model.

14.8.1 2 m Composites

Spatial continuity was assessed using variogram maps and 3D representations of grade continuity. The most suitable orientation was selected based on the general understanding of the attitude of each mineralized zone. Initially, the variograms were produced on normal scores of the composite assay grades. Downhole variograms were calculated to characterize the nugget effect. Final variogram models on original gold and silver composites were designed from the variograms on normal scores and backtransformed. Spherical variogram models used for determining grade continuity are summarized in Table 14-9 (for gold) and Table 14-10 (for silver). Figure 14-9 shows the gold variogram in the 21A Zone rhyolite zone, and Figure 14-10 illustrates gold search ellipsoids and ranges used per domain.

Figure 14-9: **Gold Variograms in the 21A Zone**



Note: Figure prepared by SRK, 2021.

Table 14-9: Variogram Parameters for Gold by Estimation Zone

Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
101	1	0.199	0.581	35	15	15	331.8	17.4	-42.2
	2		0.221	60	40	35			
201	1	0.205	0.599	20	20	20	37.3	-6.4	-140.4
	2		0.197	70	45	30			
202	1	0.0892	0.498	25	25	10	50.0	0.0	-135.0
	2		0.413	45	40	15			
203	1	0.0892	0.498	25	25	10	50.0	0.0	-135.0
	2		0.413	45	40	15			
204	1	0.0879	0.622	20	20	20	336.5	-28.9	72.8
	2		0.290	65	45	30			
3011	1	0.309	0.487	10	10	5	168.2	16.3	-53.3
	2		0.204	40	20	15			
3012	1	0.210	0.359	10	10	10	9.1	-4.2	155.3
	2		0.432	65	30	30			
302	1	0.300	0.555	10	5	5	10.8	-6.3	13.7
	2		0.145	40	30	15			
303	1	0.155	0.506	15	15	10	192.1	10.3	-22.9
	2		0.339	30	20	15			
401	1	0.262	0.663	7	7	5	2.5	-9.9	28.5
	2		0.075	25	20	10			
4011	1	0.0755	0.517	10	10	5	3.0	-33.8	157.2
	2		0.405	30	30	10			
402	1	0.0546	0.694	15	15	5	4.4	-12.7	38.3
	2		0.251	95	60	10			
501	1	0.192	0.695	15	10	5	6.7	-18.9	-47.2
	2		0.113	35	20	10			
502	1	0.0895	0.770	10	5	5	358.2	-21.6	-34.5
	2		0.141	35	30	10			
601	1	0.0828	0.470	15	15	5	354.1	-14.0	33.0
	2		0.447	70	60	20			
602	1	0.0828	0.47	15	15	5	354.1	-14.0	33.0
	2		0.447	70	60	20			
603	1	0.0544	0.403	25	25	15	354.1	-14.0	33.0
	2		0.543	70	30	20			
604	1	0.0768	0.557	10	5	5	100.8	37.8	26.6
	2		0.366	45	30	10			
7034	1	0.197	0.520	10	10	5	125.7	23.9	-26.3
	2		0.370	20	15	10			
7035	1	0.200	0.717	15	10	5	359.2	-26.1	-44.3
	2		0.078	35	20	10			
7038	1	0.125	0.666	20	5	5	3.0	-36.0	37.0
	2		0.209	35	30	10			
801	1	0.292	0.551	10	5	5	1.0	-44.0	-153.0
	2		0.157	75	30	25			
811	1	0.156	0.563	15	10	10	10.0	-44.0	-104.0
	2		0.281	50	30	30			
802	1	0.248	0.624	10	10	10	25.0	-36.0	46.0
	2		0.128	60	60	30			

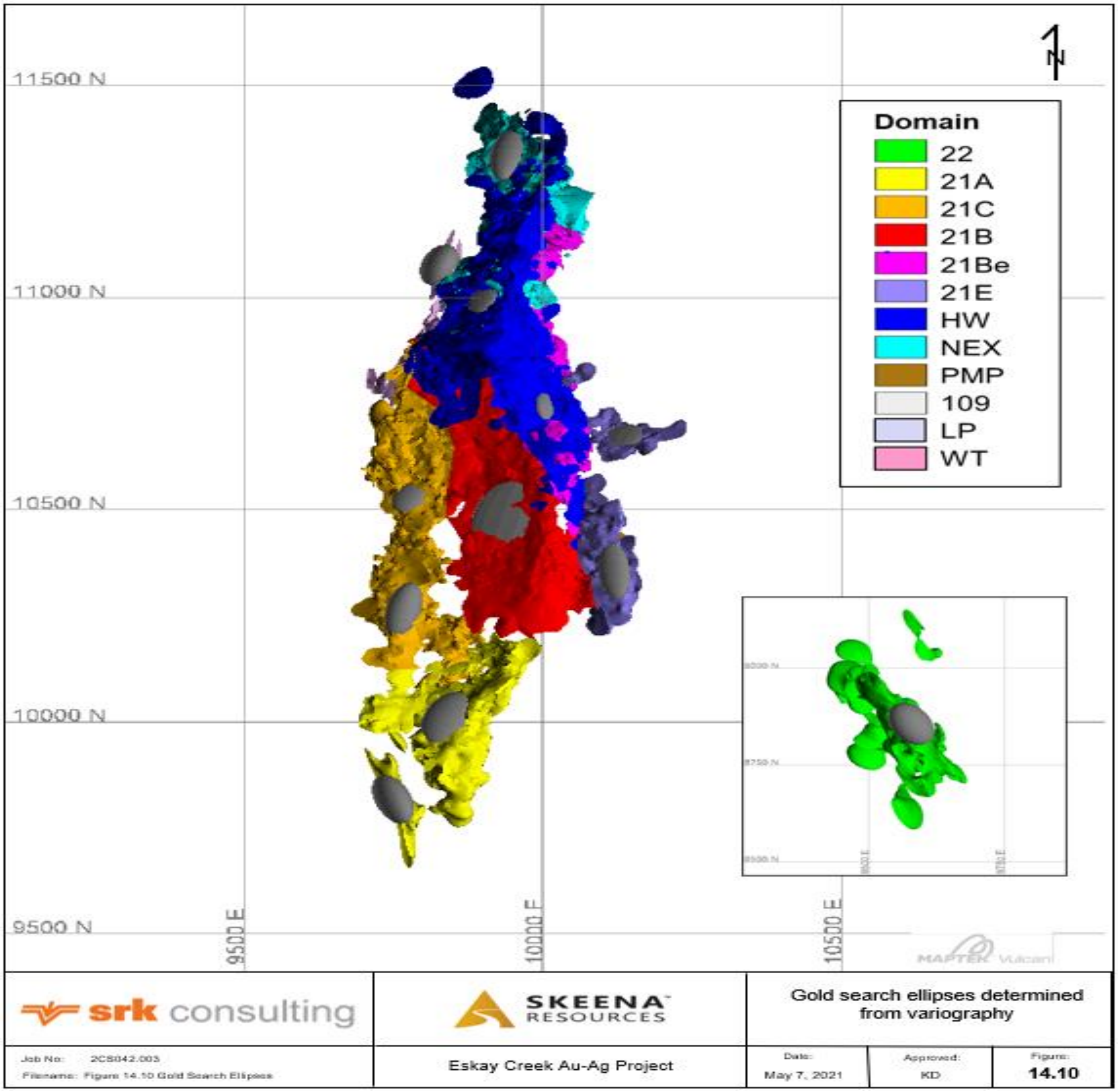
Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
90	1	0.267	0.273	20	10	10	356.3	-8.3	18.3
	2		0.459	60	45	20			
91	1	0.267	0.273	20	10	10	67.0	0.0	-40.0
	2		0.459	60	45	20			
92	1	0.267	0.273	20	10	10	67.0	0.0	-40.0
	2		0.459	60	45	20			
93	1	0.305	0.326	30	28	8	327.1	-10.3	22.9
	2		0.368	60	40	20			
94	1	0.232	0.518	25	15	13	356.7	-8.3	23.7
	2		0.250	55	25	15			
95	1	0.134	0.521	10	10	5	345.4	-19.3	105.9
	2		0.345	45	25	10			
99	1	0.352	0.450	15	10	10	6.3	-27.0	61.7
	2		0.198	60	50	20			

Table 14-10: Variogram Parameters for Silver by Estimation Zone

Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
101	1	0.097	0.459	20	15	10	331.8	17.4	-42.2
	2		0.444	55	25	35			
201	1	0.207	0.594	15	15	15	37.3	-6.4	-140.4
	2		0.199	70	40	40			
202	1	0.208	0.615	30	25	7	50.0	0.0	-135.0
	2		0.177	60	30	10			
203	1	0.208	0.615	30	25	7	50.0	0.0	-135.0
	2		0.177	60	30	10			
204	1	0.185	0.794	20	20	20	336.5	-28.9	72.8
	2		0.022	45	45	30			
3011	1	0.341	0.553	7	7	7	168.2	16.3	-53.3
	2		0.106	45	20	15			
3012	1	0.252	0.476	10	10	5	9.1	-4.2	155.3
	2		0.272	35	40	40			
302	1	0.168	0.662	15	15	10	10.8	-6.3	13.7
	2		0.170	45	30	15			
303	1	0.175	0.752	15	15	10	192.1	10.3	-22.9
	2		0.073	30	20	15			
401	1	0.222	0.655	10	10	5	2.5	-9.9	28.5
	2		0.123	35	30	20			
4011	1	0.092	0.692	10	10	5	3.0	-33.8	157.2
	2		0.216	30	30	10			
402	1	0.073	0.644	10	10	5	4.4	-12.7	38.3
	2		0.283	70	60	10			
501	1	0.128	0.790	10	10	5	6.7	-18.9	-47.2
	2		0.082	50	40	10			
502	1	0.062	0.844	5	5	5	358.2	-21.6	-34.5
	2		0.094	30	20	10			
601	1	0.076	0.711	20	5	5	354.1	-14.0	33.0
	2		0.212	50	40	20			
602	1	0.044	0.665	40	15	10	354.1	-14.0	33.0
	2		0.291	70	45	15			
603	1	0.044	0.665	40	15	10	354.1	-14.0	33.0
	2		0.291	70	45	15			

Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
604	1	0.060	0.765	5	5	5	100.8	37.8	26.6
	2		0.175	25	25	10			
7034	1	0.226	0.647	10	10	5	125.7	23.9	-26.3
	2		0.127	25	15	10			
7035	1	0.107	0.798	5	5	5	359.2	-26.1	-44.3
	2		0.095	35	35	10			
7038	1	0.055	0.758	10	5	5	3.0	-36.0	37.0
	2		0.187	30	30	15			
801	1	0.243	0.705	10	10	5	1.0	-44.0	-153.0
	2		0.052	50	25	15			
811	1	0.197	0.590	15	15	10	10.0	-44.0	-104.0
	2		0.212	30	30	15			
802	1	0.243	0.643	15	10	5	25.0	-36.0	46.0
	2		0.114	65	65	20			
90	1	0.220	0.459	30	15	10	356.3	-8.3	18.3
	2		0.321	60	30	20			
91	1	0.220	0.459	30	15	10	67.0	0.0	-40.0
	2		0.321	60	30	20			
92	1	0.220	0.459	30	15	10	67.0	0.0	-40.0
	2		0.321	60	30	20			
93	1	0.181	0.509	30	28	10	327.1	-10.3	22.9
	2		0.310	50	40	12			
94	1	0.234	0.527	25	18	15	356.7	-8.3	23.7
	2		0.239	55	30	20			
95	1	0.173	0.578	12	12	6	345.4	-19.3	105.9
	2		0.249	54	25	15			
99	1	0.286	0.554	10	6	5	6.3	-27.0	61.7
	2		0.160	20	18	16			

Figure 14-10: Gold Search Ellipses (in grey) Determined by Variography



14.8.2 1 m Composites

For the underground model, variograms for the 1 m composites used the same orientations determined from the 2 m composites, however the nugget, sills and ranges were updated accordingly (Table 14-11 and Table 14-12). The Lower Mudstone and Dacite zones were subdivided into steep and shallow solids to aid with variogram creation and estimation.

Table 14-11: 1 m Variogram Parameters for Gold by Zone

Vario_Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
9011	1	0.314	0.531	30	30	10	10.768	-26.06	44.311
	2		0.156	45	45	20			
9012	1	0.362	0.444	25	25	10	1.656	-11.313	16.6
	2		0.194	105	105	25			
9021	1	0.188	0.621	10	10	10	10.768	-26.06	44.311
	2		0.191	45	40	20			
9022	1	0.161	0.519	10	10	10	1.656	-11.313	16.6
	2		0.320	60	40	25			
9031	1	0.161	0.519	10	10	10	60	0	40
	2		0.320	60	40	25			
9032	1	0.161	0.519	10	10	10	1.656	-11.313	16.6
	2		0.320	60	40	25			
92	1	0.161	0.519	10	10	10	60	0	40
	2		0.320	60	40	25			
93	1	0.149	0.552	22	22	5	327.09	-10.289	22.91
	2		0.299	40	40	10			
94	1	0.220	0.514	25	15	15	356.74	-8.31	23.66
	2		0.266	40	30	20			
101	1	0.135	0.538	10	10	7	331.75	17.39	-42.19
	2		0.327	50	30	20			
7038	1	0.084	0.778	10	5	5	3	-36	37
	2		0.137	30	30	15			
801	1	0.291	0.649	12	10	5	1	-44	-153
	2		0.0596	30	20	15			
811	1	0.131	0.744	15	15	10	10	-44	-104
	2		0.125	30	30	15			
802	1	0.137	0.773	15	10	7	25	-36	46
	2		0.0898	50	40	20			

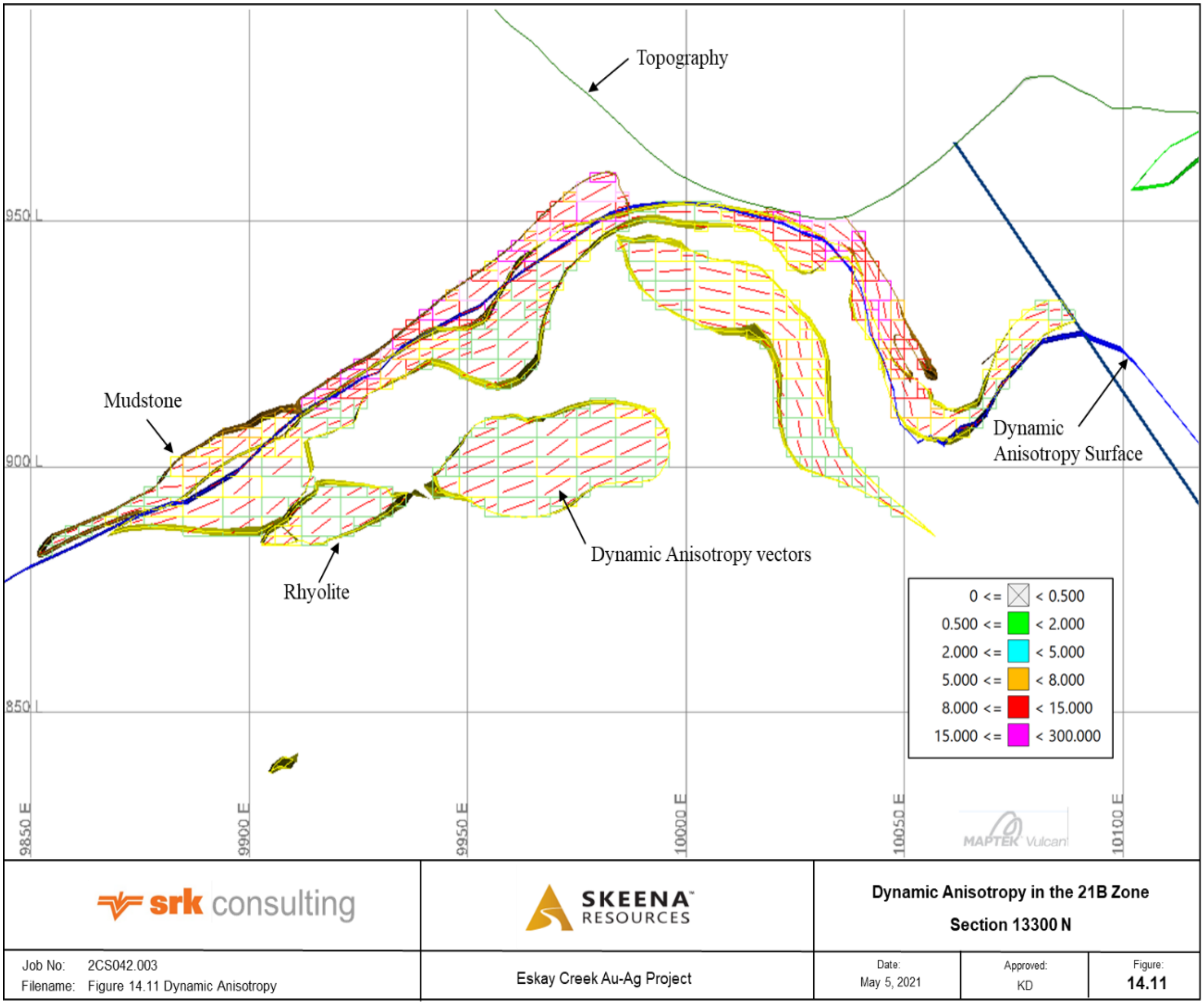
Table 14-12: 1 m Variogram Parameters for Silver by Zone

Est_Zone	Structure	Nugget	Sill	Major (Y)	Semi (X)	Minor (Z)	Final Rotation		
							Y	X	Z
9011	1	0.314	0.531	30	30	10	10.768	-26.06	44.311
	2		0.156	45	45	20			
9012	1	0.362	0.444	25	25	10	1.656	-11.313	16.6
	2		0.194	105	105	25			
9021	1	0.188	0.621	10	10	10	10.768	-26.06	44.311
	2		0.191	45	40	20			
9022	1	0.161	0.519	10	10	10	1.656	-11.313	16.6
	2		0.320	60	40	25			
9031	1	0.161	0.519	10	10	10	60	0	40
	2		0.32	60	40	25			
9032	1	0.161	0.519	10	10	10	1.656	-11.313	16.6
	2		0.320	60	40	25			
92	1	0.161	0.519	10	10	10	60	0	40
	2		0.320	60	40	25			
93	1	0.149	0.552	22	22	5	327.09	-10.289	22.91
	2		0.299	40	40	10			
94	1	0.22	0.514	25	15	15	356.74	-8.31	23.66
	2		0.266	40	30	20			
101	1	0.135	0.538	10	10	7	331.75	17.39	-42.19
	2		0.327	50	30	20			
7038	1	0.084	0.778	10	5	5	3	-36	37
	2		0.137	30	30	15			
801	1	0.291	0.649	12	10	5	1	-44	-153
	2		0.060	30	20	15			
811	1	0.131	0.744	15	15	10	10	-44	-104
	2		0.125	30	30	15			
802	1	0.137	0.773	15	10	7	25	-36	46
	2		0.090	50	40	20			

14.9 Dynamic Anisotropy

Due to the folded nature of the deposit, search ellipsoid orientations were not considered suitable for effectively estimating all the estimation domains. Dynamic anisotropy was selected as the preferred estimation method for the 21A, 21C and 21B because adjustments in each block could be made in relation to the presiding mineralization trend. The anisotropy direction was defined from the base of the Contact Mudstone (see example in Figure 14-11).

Figure 14-11: Dynamic Anisotropy Vectors Used in the Folded 21B Domain (looking north)



14.10 Specific Gravity

During 2018 to the end of the 2020 Phase 2 drilling program, Skeena collected 4,965 specific gravity measurements. The density used for tonnage calculation for the 2021 estimate is based on the lithological model, with the mean value of measurements selected as the density for each lithology considered, with the exception of the barite-rich Mudstone in the 21C Domain (302). Table 14-13 summarizes the bulk density measurement by lithology used in the model.

Table 14-13: Specific Gravity by Rocktype

Rocktype	SG
Bowser Sediments	2.74
Hanging Wall Andesite	2.80
Hanging Wall Mudstone	2.72
Contact Mudstone	2.78
Rhyolite	2.66
Lower Mudstone	2.79
Footwall Dacite	2.78
Even Lower Mudstone	2.75
Footwall Andesite	2.75
Barite-rich 21C mudstone	3.00

14.11 Block Model and Grade Estimation

The grade estimate for the 2021 Mineral Resource estimate was constructed in two stages: (1) open pit modelling and, (2) underground modelling. For the open pit model, grades were estimated into all twelve mineralization domains, and the five low-grade envelope domains. Five estimation domains below the bottom of the optimized resource pit were reported as resources potentially amenable to underground mining methods (22, HW, NEX, WTZ and the LP). Each of the models were optimized based on the defining mining scenario, and the separate methodologies and parameters are described in the following sub-sections.

14.11.1 Open Pit Model

14.11.1.1 Open Pit Model

The block model geometry and extents used for grade estimation in the open pit model are summarized in Table 14-14.

Table 14-14: Details of the Open Pit Block Model Dimensions and Block Size

	Bearing	Plunge	Dip	Start Offset			End Offset			Block Size (metres)		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8508	-50	1188	3654	1500	9	9	4
Sub-block	90	0	0	9300	8508	-50	1188	3654	1500	3	3	2

Ordinary kriging (OK) was used to estimate gold and silver in all domains. Two-metre capped composites were used for the open pit model. Gold and silver grades within the mineralization domains were estimated in three successive passes with increasing search radii based on variogram ranges as outlined in Table 14-15 and Table 14-16. Pass 1 equalled ⅔ of the variogram range, Pass 2 equalled the variogram range and Pass 3 equalled 2.5 times the variogram range. The low-grade envelope domain was estimated using restricted ranges using one pass. A fourth validation pass at five times the variogram range was estimated in the Lower Package Domain to aid with validation.

For Pass 1, a minimum of eight and maximum of 10 composites were used per block. For Pass 2, a minimum of five and maximum of 15 composites were used per block and for Pass 3 a minimum of three and maximum of 15 composites were used per block. A maximum of two composites per drill hole was specified for all passes.

Hard boundary interpolation was honoured for all domains. A hard boundary was applied within a 3-m restriction domain to limit the spread of high-grade values from mined-out intervals into the remaining resources area. A discretization grid of 4 m x 4 m x 3 m was used. A summary of gold and silver parameters used for estimation are shown in Table 14-15 and Table 14-16.

Table 14-15: Gold Estimation Parameters by Estimation Zone

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
1	1	22 Zone	1	331.7/17.4/-42.2	25	25	15	3	10	2
2	2	Main Zone	1	Dynamic Anisotropy	25	25	15	3	10	2
3	3	NEX	1	33.18/-28.0/49.47	25	25	15	3	10	2
4	4	21Be	1	358.2/-21.6/34.5	25	25	15	3	10	2
5	5	21E	1	354.1/-14.0/32.4	25	25	15	3	10	2
101	1011, 1012, 1013	Rhyolite	1	331.7/17.4/-42.2	40	27	23	8	10	2
			2		60	40	35	5	15	2
			3		150	100	87.5	3	15	2
201	2011 (fault)	Rhyolite	1	336.5/-28.9/72.1	44	30	20	8	10	2
			2		65	45	30	5	15	2
			3		162.5	112.5	75	3	15	2
201	2012, 2013	Rhyolite	1	Dynamic Anisotropy	47	30	20	8	10	2
			2		70	45	30	5	15	2
			3		175	112.5	75	3	15	2
202	2021	Contact Mudstone	1	Dynamic Anisotropy	30	27	10	8	10	2
			2		45	40	15	5	15	2
			3		112.5	100	37.5	3	15	2
203	2031	Hang Wall Mudstone	1	Dynamic Anisotropy	30	27	10	8	10	2
			2		40	40	15	5	15	2
			3		100	100	37.5	3	15	2

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
301	3011, 3012, 3013 (North)	Rhyolite	1	Dynamic Anisotropy	27	13	10	8	10	2
			2		40	20	15	5	15	2
			3		100	50	37.5	3	15	2
301	3014 (South)	Rhyolite	1	Dynamic Anisotropy	44	20	10	8	10	2
			2		65	30	30	5	15	2
			3		162.5	75	75	3	15	2
302	3021, 3022	Contact Mudstone	1	Dynamic Anisotropy	27	20	10	8	10	2
			2		40	30	15	5	15	2
			3		100	75	37.5	3	15	2
303	3031 to 3037	Hang Wall Mudstone	1	12.1/-10.3/22.3	20	13	10	8	10	2
			2		30	20	15	5	15	2
			3		75	50	37.5	3	15	2
401	4011, 4013, 4014, 4015, 4017	Rhyolite	1	Dynamic Anisotropy	17	13	7	8	10	2
			2		25	20	10	5	15	2
			3		175	112.5	75	3	15	2
401	4012, 4016	Rhyolite	1	3/-33.826/157.2	20	20	7	8	10	2
			2		30	20	10	5	15	2
			3		75	50	25	3	15	2
402	4021, 4022	Rhyolite	1	Dynamic Anisotropy	64	40	7	8	10	2
			2		95	60	10	5	15	2
			3		237.5	150	25	3	15	2
501	5011, 5012	Rhyolite	1	6.7/-19/--47.2	23	13	7	8	10	2
			2		35	20	10	5	15	2
			3		87.5	50	25	3	15	2
502	5021	Contact Mudstone	1	358.2/-21.6/-34.5	20	20	7	8	10	2
			2		35	30	10	5	15	2
			3		87.5	75	25	3	15	2
601		Rhyolite	1	354.0/-14.0/32.4	47	40	13	8	10	2

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
	6011, 6012, 6013, 6014,		2		70	60	20	5	15	2
			3		175	150	50	3	15	2
602	6021	Contact Mudstone	1	354.0/-14.0/32.4	47	20	13	8	10	2
			2		70	30	20	5	15	2
			3		175	75	50	3	15	2
603	6031, 6032, 6033, 6034	Hanging-Wall Mudstone	1	354.0/-14.0/32.4	47	20	13	8	10	2
			2		70	30	20	5	15	2
			3		175	75	50	3	15	2
604	6041, 6042, 6043	Hanging-Wall Mudstone	1	100.8/37.8/26.6	30	20	7	8	10	2
			2		45	30	10	5	15	2
			3		112.5	75	25	3	15	2
703	70341, 70342, 70343	Hanging-Wall Mudstone	1	Dynamic Anisotropy	13	10	7	8	10	2
			2		20	15	10	5	15	2
			3		50	37.5	25	3	15	2
703	70351 to 70357	Hanging-wall Mudstone	1	359.2/-26.0/-44.3	23	13	7	8	10	2
			2		35	20	10	5	15	2
			3		87.5	50	25	3	15	2
703	70382 to 70386	Hanging-Wall Mudstone	1	2.5/-35.93/37.4	23	20	7	8	10	2
			2		35	30	10	5	15	2
			3		87.5	75	25	3	15	2
801	8011, 8012, 8013, 8014	Rhyolite	1	6.9/-41.6/-149.2	50	20	17	8	10	2
			2		75	30	10	5	15	2
			3		187.5	75	25	3	15	2
802	8021, 8022	Contact Mudstone	1	33.2/-28.0/49.5	40	40	20	8	10	2
			2		60	60	30	5	15	2
			3		150	150	75	3	15	2
811	8111 to 8116	Rhyolite	1	13.1/-34.4/102.1	34	34	20	8	10	2
			2		50	50	30	5	15	2

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
			3		125	125	75	3	15	2
90	901,902	Dacite	1	356.3/-8.3/18.3	40	30	13	8	10	2
			2		60	45	20	5	15	2
			3		150	112.5	50	3	15	2
91	903	Even Lower Mudstone	1	356.3/-8.3/18.3	40	30	13	8	10	2
			2		60	45	20	5	15	2
			3		150	112.5	50	3	15	2
92	904	Footwall Andesite	1	67/0/40	40	30	13	8	10	2
			2		60	45	20	5	15	2
			3		150	112.5	50	3	15	2
93	905	Lower Mudstone	1	160.5/2.1/-24.9	40	27	13	8	10	2
			2		60	45	20	5	15	2
			3		150	112.5	50	3	15	2
94	906	Rhyolite	1	356.7/-8.31/23.7	37	17	10	8	10	2
			2		60	45	15	5	15	2
			3		150	112.5	37.5	3	15	2
95	951, 952,953	Rhyolite	1	345.4/-19.3/105.9	30	17	7	8	10	2
			2		45	25	10	5	15	2
			3		112.5	62.5	25	3	15	2
99	99	Rhyolite	1	6.3/-26.9/61.7	40	34	13	8	10	2
			2		60	50	20	5	15	2
			3		150	125	50	3	15	2

Table 14-16: **Silver Grade Estimation Parameters by Estimation Zone**

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
1	1	22 Zone	1	331.7/17.4/-42.2	25	25	15	3	10	2
2	2	Main Zone	1	Dynamic Anisotropy	25	25	15	3	10	2
3	3	NEX	1	33.18/-28.0/49.47	25	25	15	3	10	2
4	4	21Be	1	358.2/-21.6/34.5	25	25	15	3	10	2

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
5	5	21E	1	354.1/-14.0/32.4	25	25	15	3	10	2
101	1011, 1012, 1013	Rhyolite	1	331.7/17.4/-42.2	82	37	52	8	10	2
			2		55	25	35	5	15	2
			3		138	63	88	3	15	2
201	2011 (fault)	Rhyolite	1	336.5/-28.9/72.1	67	67	45	8	10	2
			2		45	45	30	5	15	2
			3		113	113	75	3	15	2
201	2012, 2013	Rhyolite	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		70	40	40	5	15	2
			3		175	100	100	3	15	2
202	2021	Contact Mudstone	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		60	30	10	5	15	2
			3		150	75	25	3	15	2
203	2031	Hanging Wall Mudstone	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		60	30	10	5	15	2
			3		150	75	25	3	15	2
301	3011, 3012, 3013 (North)	Rhyolite	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		45	20	15	5	15	2
			3		113	50	38	3	15	2
301	3014 (South)	Rhyolite	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		35	40	40	5	15	2
			3		88	100	100	3	15	2
302	3021, 3022	Contact Mudstone	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		45	30	15	5	15	2
			3		113	75	38	3	15	2
303	3031 to 3037	Hanging Wall Mudstone	1	12.1/-10.3/22.3	82	37	52	8	10	2
			2		30	20	15	5	15	2
			3		75	50	38	3	15	2
401	4011, 4013, 4014, 4015, 4017	Rhyolite	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		35	30	20	5	15	2
			3		88	75	50	3	15	2
401	4012, 4016	Rhyolite	1	3/-33.826/157.2	82	37	52	8	10	2
			2		30	30	10	5	15	2
			3		75	75	25	3	15	2
402	4021, 4022	Rhyolite	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		70	60	10	5	15	2
			3		175	150	25	3	15	2
501	5011, 5012	Rhyolite	1	6.7/-19/--47.2	82	37	52	8	10	2
			2		50	40	10	5	15	2
			3		125	100	25	3	15	2
502	5021	Contact Mudstone	1	358.2/-21.6/-34.5	82	37	52	8	10	2
			2		30	20	10	5	15	2
			3		75	50	25	3	15	2
601	6011, 6012, 6013, 6014,	Rhyolite	1	354.0/-14.0/32.4	82	37	52	8	10	2
			2		50	40	20	5	15	2
			3		125	100	50	3	15	2
602	6021	Contact Mudstone	1	354.0/-14.0/32.4	82	37	52	8	10	2
			2		70	45	15	5	15	2
			3		175	113	38	3	15	2
603	6031, 6032, 6033, 6034	Hanging Wall Mudstone	1	354.0/-14.0/32.4	82	37	52	8	10	2
			2		70	45	15	5	15	2
			3		175	113	38	3	15	2

Zone	Est_Zone	Rocktype	Search Pass	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
					X	Y	Z	Minimum	Maximum	
604	6041, 6042, 6043	Contact Mudstone/ Rhyolite	1	100.8/37.8/26.6	82	37	52	8	10	2
			2		25	25	10	5	15	2
			3		63	63	25	3	15	2
703	70341, 70342, 70343	Hanging Wall Mudstone	1	Dynamic Anisotropy	82	37	52	8	10	2
			2		25	15	10	5	15	2
			3		63	38	25	3	15	2
703	70351 to 70357	Hanging Wall Mudstone	1	359.2/-26.0/-44.3	82	37	52	8	10	2
			2		35	35	10	5	15	2
			3		88	88	25	3	15	2
703	70382 to 70386	Hanging Wall Mudstone	1	2.5/-35.93/37.4	82	37	52	8	10	2
			2		30	30	15	5	15	2
			3		75	75	38	3	15	2
801	8011, 8012, 8013, 8014	Rhyolite	1	6.9/-41.6/-149.2	82	37	52	8	10	2
			2		50	25	15	5	15	2
			3		125	63	38	3	15	2
802	8021, 8022	Contact Mudstone	1	33.2/-28.0/49.5	82	37	52	8	10	2
			2		65	65	20	5	15	2
			3		163	163	50	3	15	2
811	8111 to 8116	Rhyolite	1	13.1/-34.4/102.1	82	37	52	8	10	2
			2		30	30	15	5	15	2
			3		75	75	38	3	15	2
90	901,902	Dacite	1	356.3/-8.3/18.3	82	37	52	8	10	2
			2		60	30	20	5	15	2
			3		150	75	50	3	15	2
91	903	Even Lower Mudstone	1	356.3/-8.3/18.3	82	37	52	8	10	2
			2		60	30	20	5	15	2
			3		150	75	50	3	15	2
92	904	Footwall Andesite	1	67/0/40	82	37	52	8	10	2
			2		60	30	20	5	15	2
			3		150	75	50	3	15	2
93	905	Lower Mudstone	1	160.5/2.1/-24.9	82	37	52	8	10	2
			2		50	40	20	5	15	2
			3		125	100	50	3	15	2
94	906	Rhyolite	1	356.7/-8.31/23.7	82	37	52	8	10	2
			2		55	30	20	5	15	2
			3		138	75	50	3	15	2
95	951, 952,953	Rhyolite	1	345.4/-19.3/105.9	82	37	52	8	10	2
			2		55	25	15	5	15	2
			3		138	63	38	3	15	2
99	99	Rhyolite	1	6.3/-26.9/61.7	82	37	52	8	10	2
			2		55	25	35	5	15	2
			3		138	63	88	3	15	2

14.11.1.2 Open Pit Model – Visual Validation

Estimated block grades were assessed in plan and sectional view along with composite assay intervals. This method provides a local visual assessment of interpolated blocks in relation to the nearest composite. Figure 14-12 and Figure 14-13 show estimated AuEQ block grades in relation to 2 m AuEQ composite intervals in the 21B/21E and 21A domains, respectively. Overall, the data show good agreement and no obvious discrepancies between block grades and composites were observed.

Figure 14-12: Visual Comparison of Block Model Gold Grades vs 2 m Composite Gold Grades in the 21B, 21E and PMP Domains (looking north)

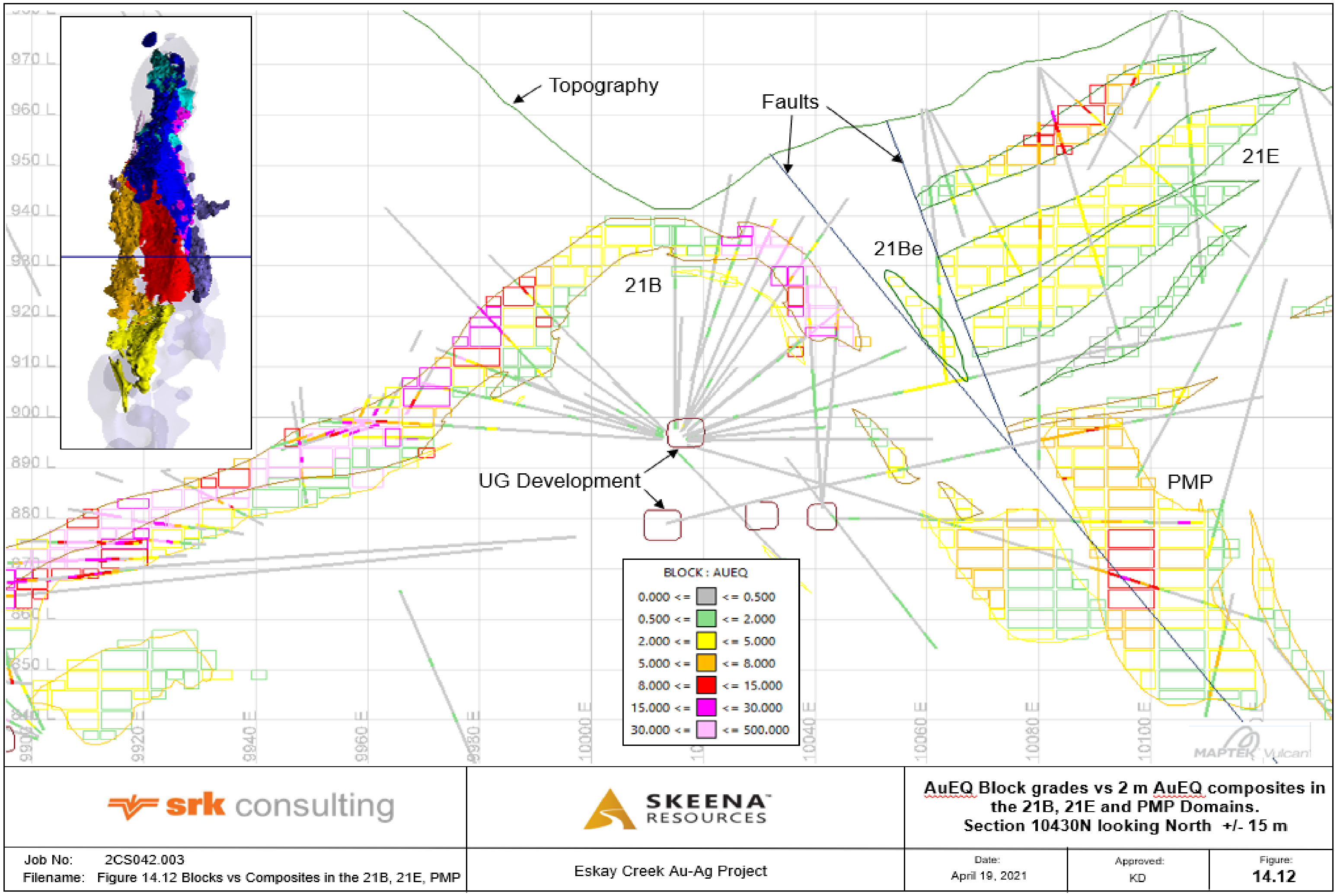
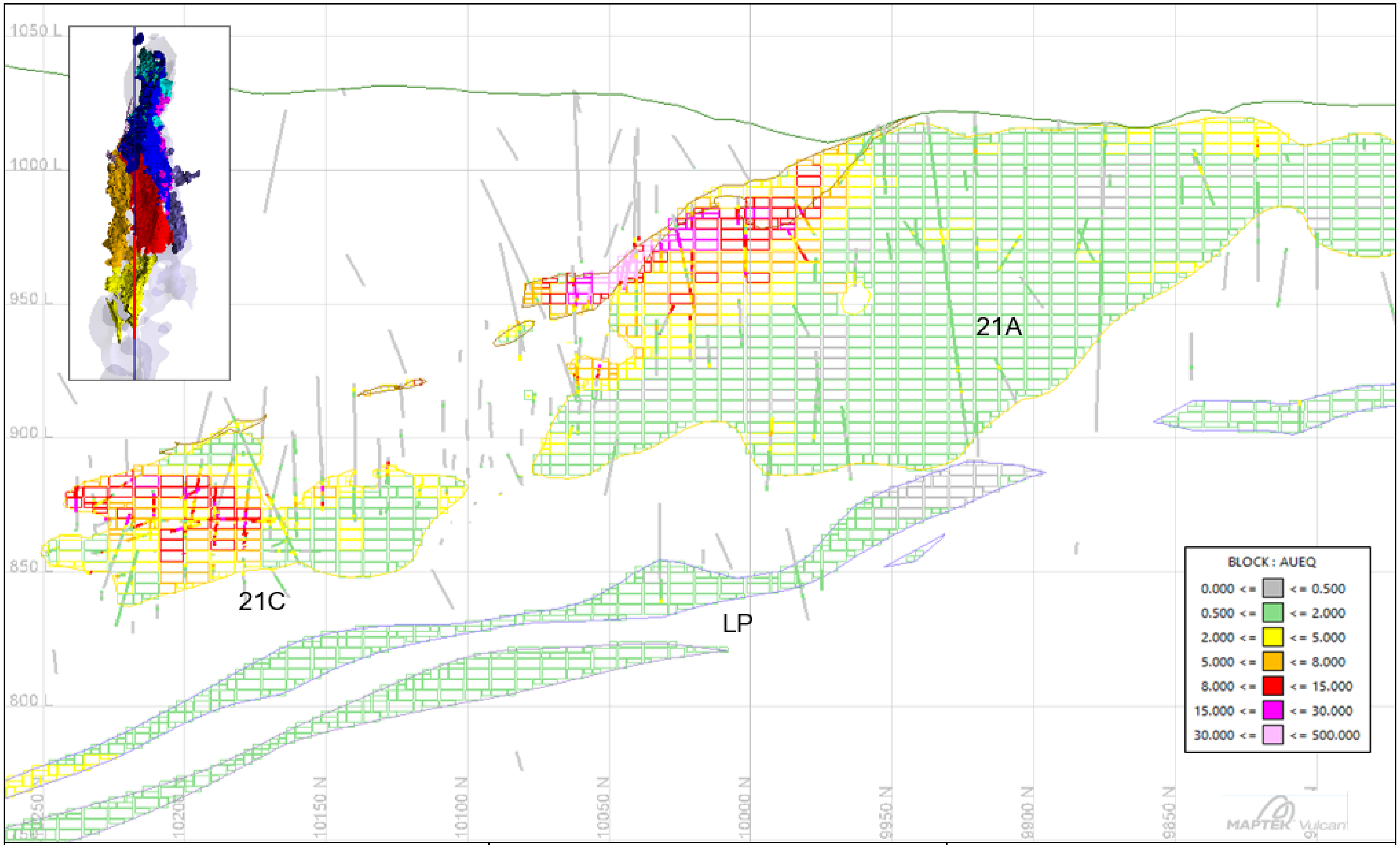


Figure 14-13: Visual Comparison of Block Model AuEQ Grades and 2 m Composite AuEQ Grades in the 21A Domain (looking east)



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AuEQ Block grades vs 2 m AuEQ composites in the 21A Domain Section 9865 E looking East +/- 15 m

Job No: 2CS042.003
Filename: Figure 14.13 Blocks vs Composites in 21A Domain

Eskay Creek Au-Ag Project

Date: April 19, 2021
Approved: KD
Figure: **14.13**

14.11.1.3 Open Pit Model – Comparison of Interpolation Models

To obtain an appropriate declustered mean of the composite grades, true nearest-neighbour (NN declustered) models were created. For the open pit model, parent blocks of 2 x 2 x 2 m were created and the closest 2 m composite up to a maximum distance of 200 m was estimated. For the underground model, parent blocks of 1 x 1 x 1 m were created and the closest 1 m composite up to a maximum distance of 200 m was estimated.

Global bias check models using block sizes equivalent to the OK estimate method were estimated using inverse distance weighting to the second power (ID^2) and NN models.

Although variable between zones, the overall global bias in relation to declustered mean values (NN declustered) were less than 1% for both gold and silver in the open pit model. A summary of global bias between the NN declustered, ID^2, and OK estimation methods for gold and silver by estimation zone are summarized in Table 14-17 and Table 14-18. The differences are within acceptable limits.

Table 14-17: Global Bias Check for Gold by Zone

Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
1	-	0.07	0.07	0	-
2	-	0.099	0.096	3	-
3	-	0.085	0.081	5	-
4	-	0.128	0.123	4	-
5	-	0.058	0.057	2	-
101	1.27	1.29	1.27	2	2
201	1.67	1.69	1.68	1	1
202	11.26	11.22	11.58	-3	0
203	0.83	0.77	0.83	-8	-8
301	2.6	2.66	2.63	1	2
302	2.24	2.23	2.25	-1	0
303	2.85	2.9	2.96	-2	2
401	2.53	2.54	2.53	0	0
402	24.18	23.12	23.64	-2	-5
501	5.05	5.13	5.05	2	2
502	13.57	13.08	12.97	1	-4

Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
601	1.55	1.54	1.54	0	-1
602	4.5	4.65	4.8	-3	3
603	1.8	1.82	1.88	-3	1
703	3.32	3.31	3.34	-1	0
801	2.49	2.53	2.54	0	2
802	5.29	5.3	5.35	-1	0
8011	2.64	3	2.91	3	12
90	0.79	0.80	0.8	0	2
91	0.99	0.96	0.896	6	-3
92	0.95	1.01	1.05	-4	6
93	0.74	0.75	0.765	-2	1
94	0.89	0.88	0.85	3	-1
95	3.54	3.61	3.61	0	2
99	7.08	7.17	7.31	-2	1
				-1	1

Table 14-18: Global Bias Check for Silver by Zone

Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
1	-	1.2	1.23	0	-
2	-	1.6	1.61	2	-
3	-	1.7	1.62	3	-
4	-	4.6	4.4	4	-
5	-	1.8	1.81	2	-
101	46.8	41.8	44.7	-7	-12
201	36.5	36.8	37	-1	1

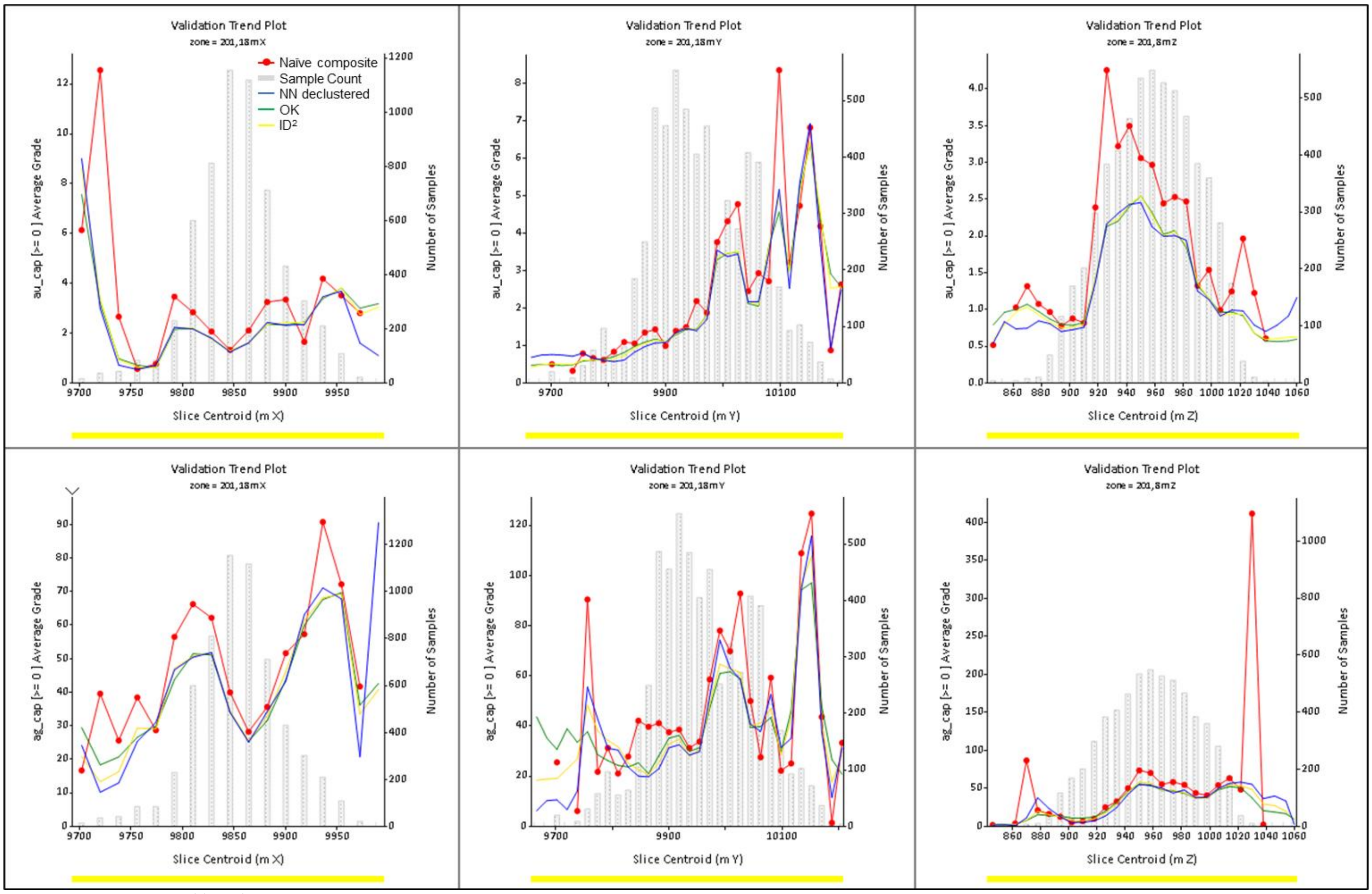
Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
202	115.3	110.6	113.9	-3	-4
203	0.7	0.7	0.7	0	0
301	28.4	29.4	29	1	3
302	75.3	76.4	77.6	-2	1
303	131.6	129.3	140	-8	-2
401	91.4	90.6	88.6	2	-1
402	924.8	929.1	939.1	-1	0
501	209.7	224.9	226.4	-1	7
502	522.4	491.3	510.5	-4	-6
601	48.7	49.1	50	-2	1
602	184.7	187.2	198.7	-6	1
603	59.3	60.8	62.2	-2	2
703	157.6	153.9	164.2	-7	-2
801	59.2	59.2	58.6	1	0
802	233.1	228.9	231.6	-1	-2
8011	14.3	15.02	14.925	1	5
90	7.8	7.6	7.9	-4	-3
91	7.5	6.9	7.2	-4	-8
92	3.9	3.1	3.4	-8	-24
93	12.5	13.2	11.6	12	5
94	10.4	10.7	10.4	2	2
95	79.9	78	76.1	2	-2
99	12.6	13.3	13.1	2	5
				-1	-1

14.11.1.4 Open Pit Model – Swath Plots

The open pit model was checked for local trends in the grade estimate using swath plots within each domain. This was done by plotting the mean values from the naïve, declustered NN, and ID^2 estimates against the OK estimate along north–

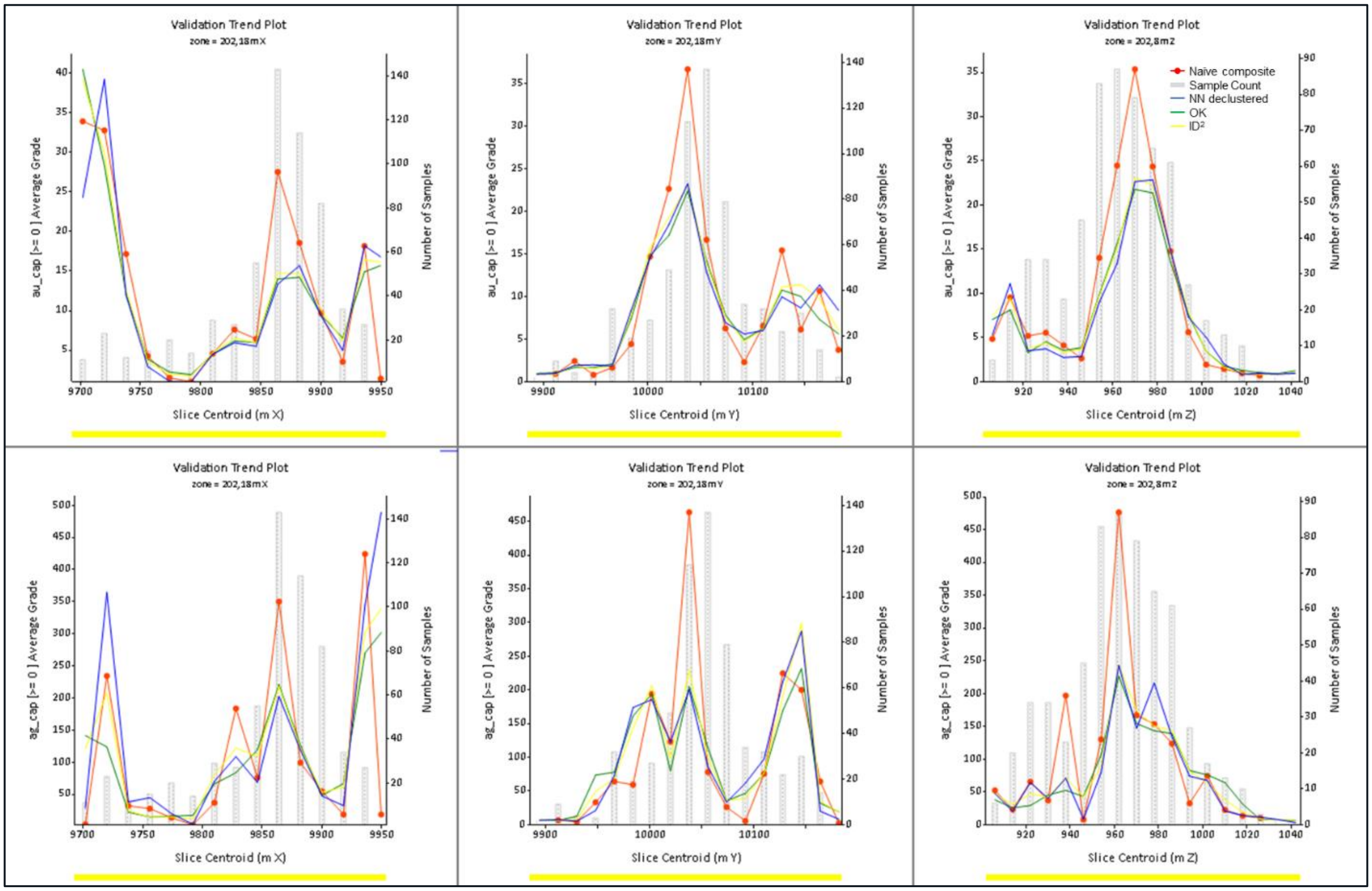
south, east–west, and horizontal swaths. The ID^2, declustered NN and OK models show similar trends in grades with the expected smoothing for each method. The observed trends show no significant metal bias in the estimate. Swath plots for gold and silver in the 21A Domain rhyolite and mudstones are illustrated in Figure 14-14 and Figure 14-15 respectively.

Figure 14-14: Swath Plot for Gold (top) and Silver (bottom) in Zone 201 - 21A Rhyolite, (top) Northing, (middle) Easting, (bottom) Elevation



Note: Figure prepared by Skeena, 2021

Figure 14-15: Swath Plot for Gold (left) and Silver (bottom) in Est_Zone 201 – 21A Mudstone, (top) Northing, (middle) Easting, (bottom) Elevation



Note: Figure prepared by Skeena, 2021.

14.11.2 Underground

14.11.2.1 Underground Model

The block model geometry and extents used for grade estimation in the underground model are summarized in Table 14-19.

Table 14-19: Details of Block Model Dimensions and Block Size for the Underground Model

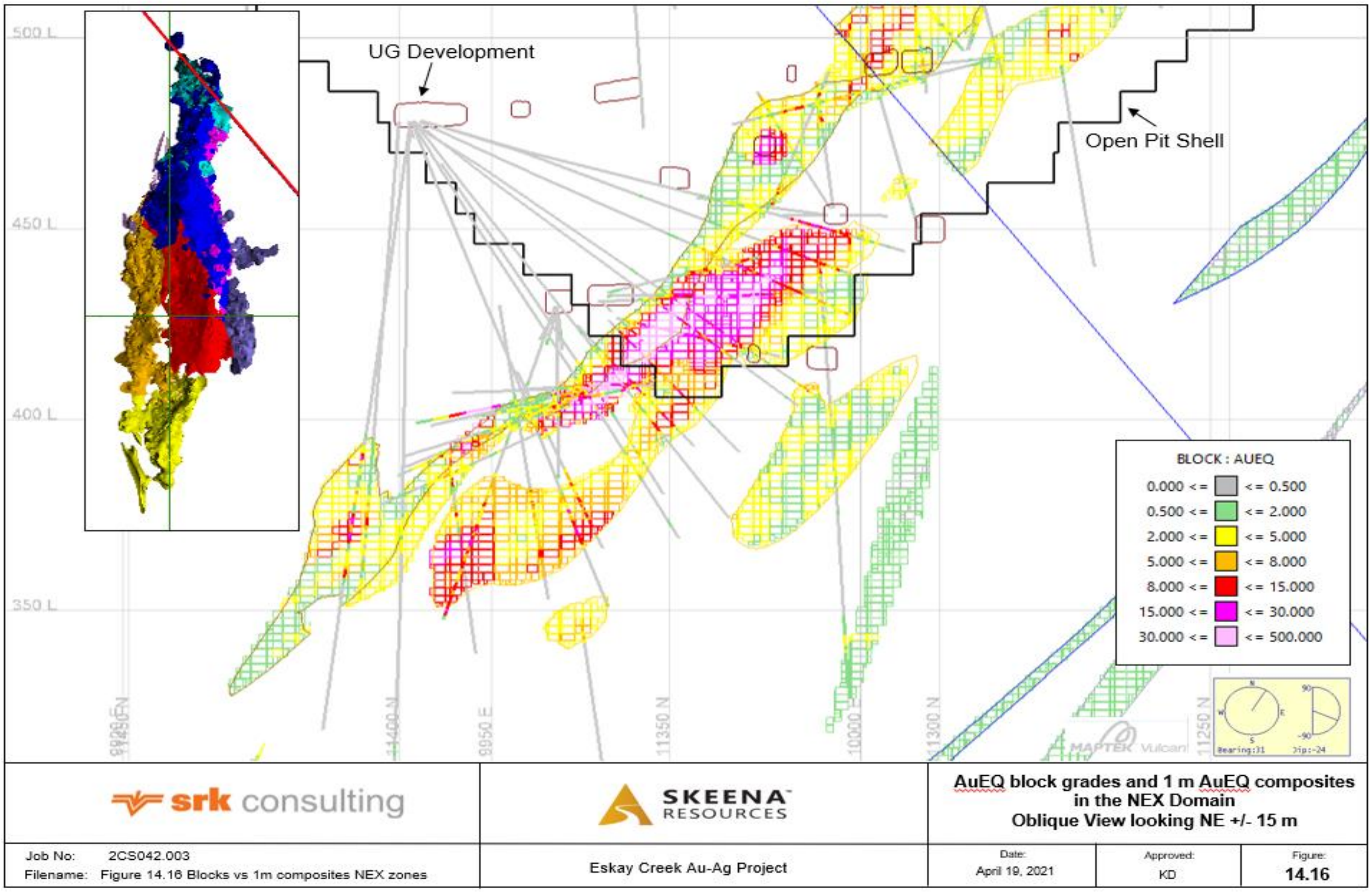
	Bearing	Plunge	Dip	Origin			End Offset			Block Size (metres)		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8508	-50	1188	3654	1500	3	3	2
Sub-block	90	0	0	9300	8508	-50	1188	3654	1500	1	1	1

Five domains were captured within the underground model: 22, HW, NEX, WT, and LP. OK was used to estimate gold and silver in all five domains. One-metre capped composites were used for the underground model. Gold and silver grades within mineralized domains were estimated in three successive passes with increasing search radii. Pass 1 approximated ⅔ of the variogram range, Pass 2 equalled the variogram range and Pass 3 equalled 2.5 times the variogram range. Hard boundaries during interpolation were honoured. Hard boundaries were used for composites within the 3 m restriction domain to limit the effect of high-grade smearing from mined-out intervals. For Pass 1 a minimum of eight and maximum of 10 composites were used per block. For Pass 2, a minimum of five and maximum of 15 composites were used per block and for Pass 3, a minimum of three and maximum of 15 composites were used per block. A maximum of two composites per drill hole was specified for all passes. A 1 m geotechnical solid around the underground workings was used as the depletion zone for reporting remaining resources.

14.11.2.2 Underground Model – Visual Validation

A visual inspection of the block estimates with drill hole composites in plan and cross-section was performed as a first pass check on the estimates. Good agreement between the composite grades and block estimates was observed, as well as suitably oriented estimates relative to variogram orientations (Figure 14-16).

Figure 14-16: Visual Check of the Underground Model Showing 1 m AuEQ Composites and Estimated AuEQ Block Grades in the NEX Domain



14.11.2.3 Underground Model – Comparison of Interpolation Models

To validate the OK estimates, gold and silver were estimated using ID^2 and NN declustered models to assess for global bias. Although variable between zones, the overall bias was <2% for gold and 1% for silver in the underground model. A difference of more than +10% was used as a guideline to indicate bias or significant over- or under-estimation. As seen in Table 14-20 and Table 14-21, the results are within acceptable limits.

Table 14-20: Global Validation of Gold

Zone	NN Declustered	ID2	OK	OK vs ID^2 (%)	OK vs NN Declustered (%)
101	1.49	1.49	1.47	-1	-1
703	2.82	2.89	2.78	-4	-1
801	2.45	2.49	2.51	1	2
802	5.16	5.25	5.22	-1	1
811	2.66	2.82	2.90	3	8
90	0.83	0.85	0.85	0	3
91	0.77	0.70	0.74	5	-4
92	1.07	0.92	0.87	-6	-23
93	0.71	0.72	0.72	0	2
94	0.96	0.89	0.92	3	-5
				0	-2

Table 14-21: Global Validation of Silver

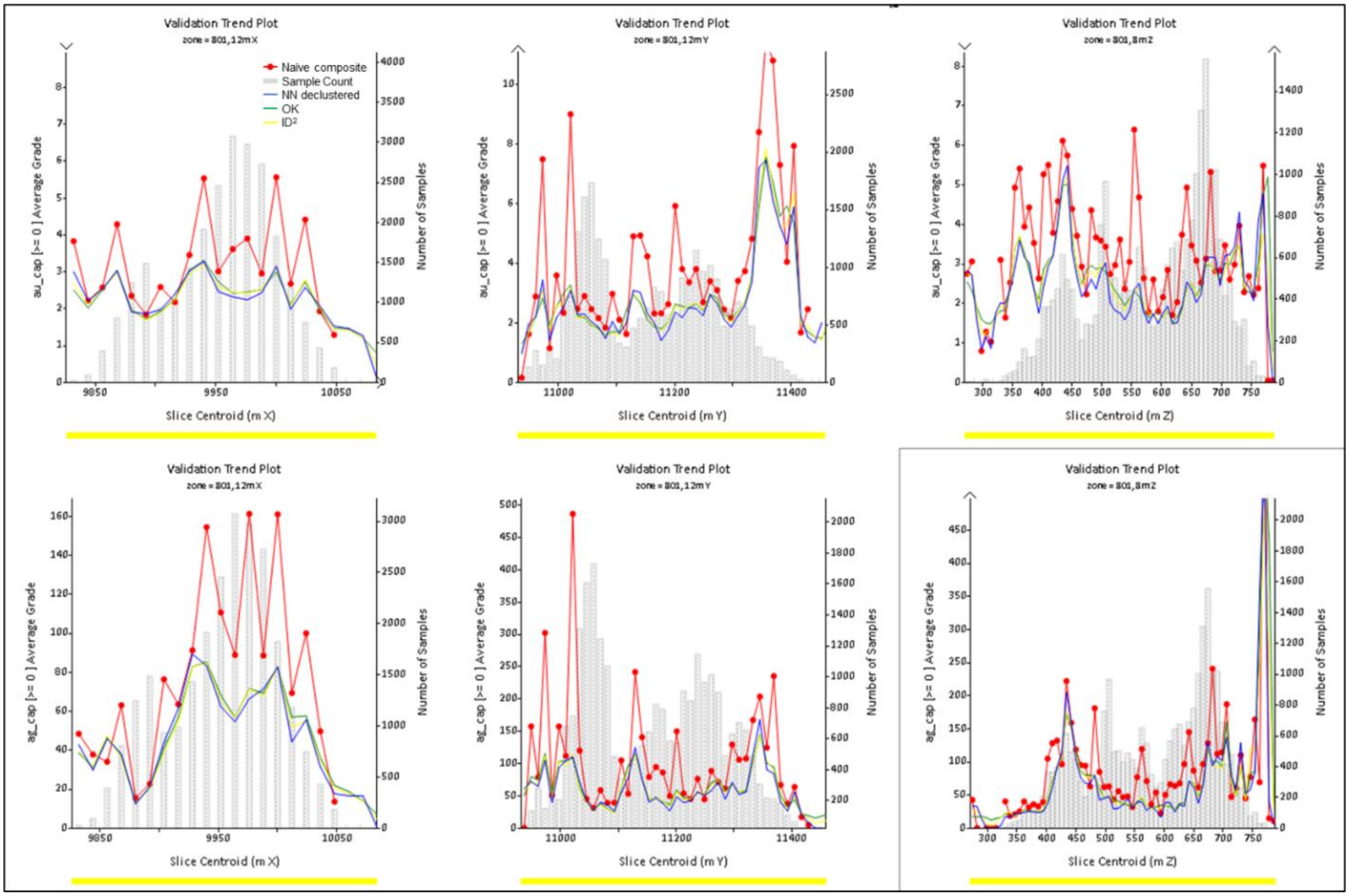
Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
90	7.57	7.90	8.11	-3	4
91	7.90	7.64	6.87	10	-3
92	4.47	4.19	4.31	-3	-7
93	12.56	14.75	13.83	6	15
94	10.67	10.69	10.27	4	0
101	53.20	45.47	48.30	-6	-17
703	61.48	67.08	68.09	-2	8

Zone	NN Declustered	OK	ID	OK vs ID^2 (%)	OK vs NN Declustered (%)
801	58.45	60.21	59.46	1	3
802	249.10	237.81	236.34	1	-5
811	13.61	14.82	14.65	1	8
				1	1

14.11.2.4 Underground Model – Swath Plots

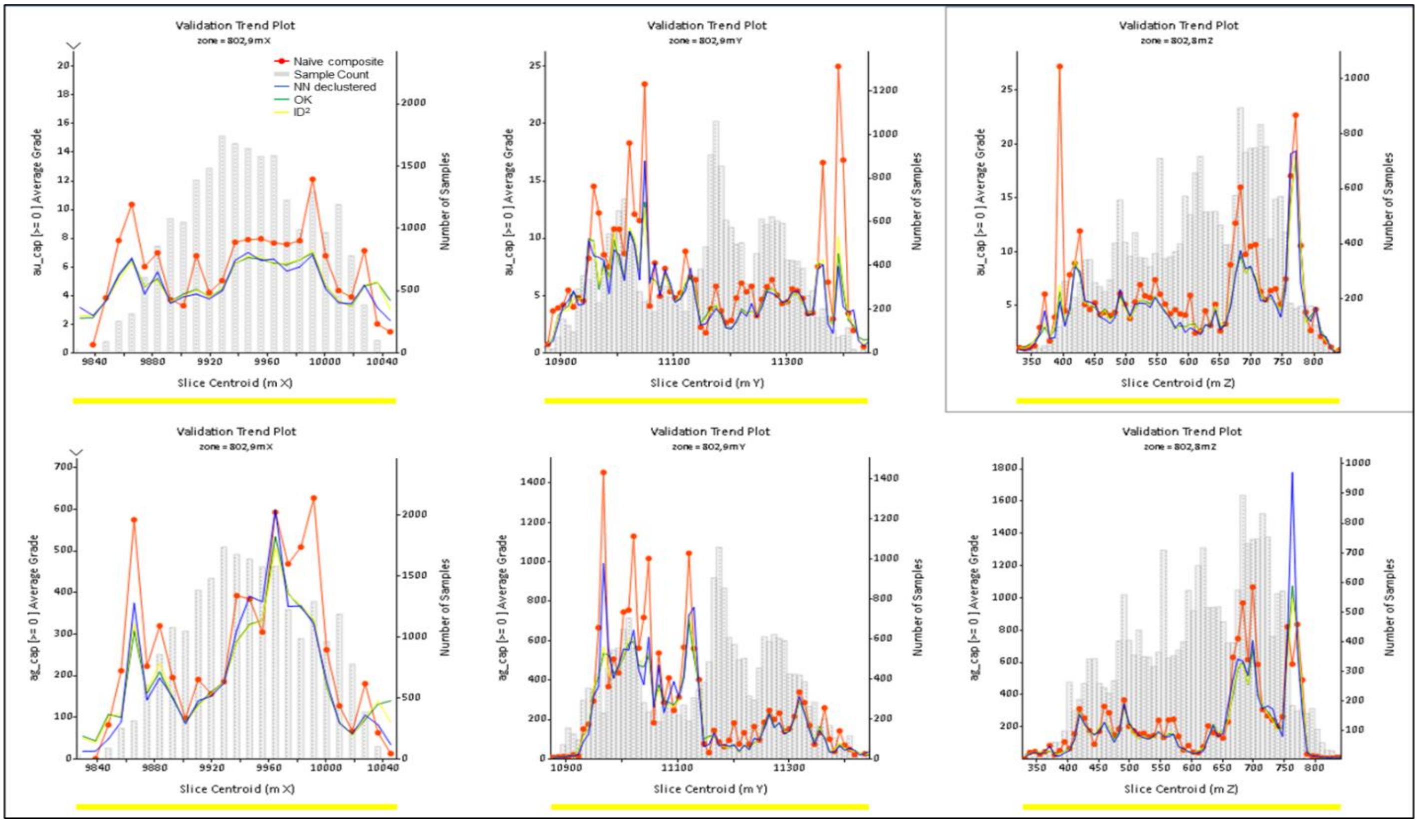
As part of the validation process, declustered composite samples (declustered NN model using 1 m blocks) and ID^2 were compared with OK block model grades in three principal directions to assess for grade and local trend discrepancies. The observed block trends follow the overall composite trends as was expected. Figure 14-17 and Figure 14-18 show OK, ID^2, and declustered NN and declustered composites for the HW and NEX Zones for gold and silver grades, respectively.

Figure 14-17: Swath Plot for Gold (top) and Silver (bottom) in Zone 801 – NEX Rhyolite, (top) Northing, (middle) Easting, (bottom) Elevation



Note: Figure prepared by Skeena, 2021.

Figure 14-18: Swath Plot for Gold (top) and Silver (bottom) in Zone 802 – NEX Mudstone, (top) Northing, (middle) Easting, (bottom) Elevation

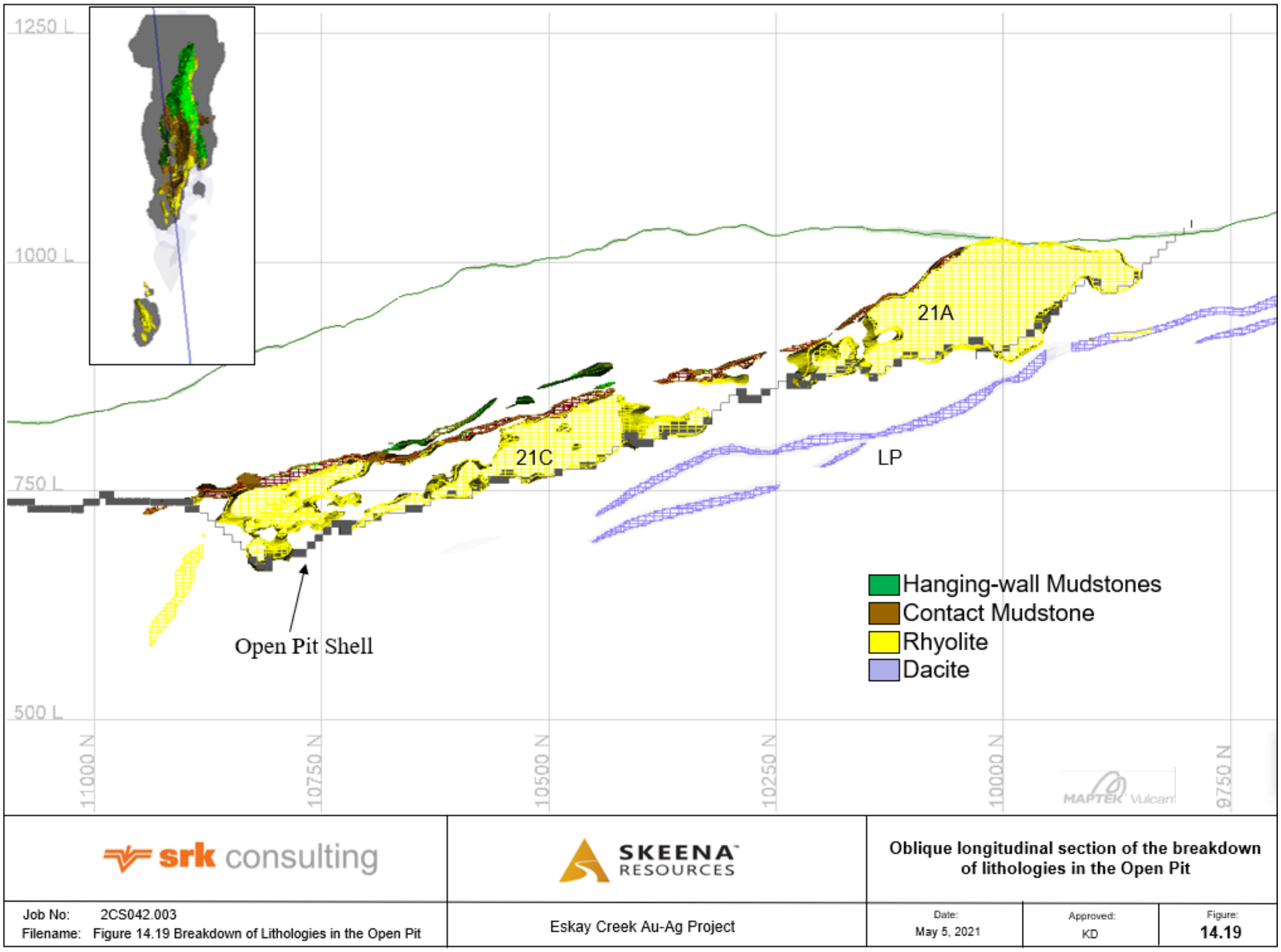


Note: Figure prepared by Skeena, 2021.

14.12 Rhyolite versus Mudstone Estimates

Most of the remaining mineralization on a tonnage basis at Eskay Creek is hosted in the rhyolite lithology, which is not enriched in the exhalative epithermal suite of elements (mercury–arsenic–antimony). Preferential historical development and mining of the bonanza-grade mineralization hosted in the Contact Mudstone resulted in extensive depletion of resources in this rocktype. The 2021 pit-constrained estimate indicates that on a tonnage weighted basis, 68% of the resource is hosted within the rhyolite facies with only 30% hosted in the remaining unmined mudstones/hanging-wall andesite (Figure 14-19). Less than 2% is hosted within the footwall Dacite. On an ounce-weighted basis, 52% of the open pit-constrained resource estimate is contained within the rhyolite with the remaining 48% hosted within the unmined sediments/Hanging Wall Andesite/Dacite.

Figure 14-19: Breakdown of Lithologies in the 21C, 21A and LP Domains (looking east)



Oblique longitudinal section of the breakdown of lithologies in the Open Pit

Job No: 2CS042.003
Filename: Figure 14.19 Breakdown of Lithologies in the Open Pit

Eskay Creek Au-Ag Project

Date: May 5, 2021
Approved: KD
Figure: 14.19

14.13 Mineral Resource Classification

Block model quantities and grade estimates for the Eskay Creek Project were classified using the 2014 CIM Definition Standards.

Mineral Resource classification is typically a subjective concept. Industry practices suggest that resource classification should consider the following: the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating all of the above requirements to delineate regular areas at similar resource classifications.

The QP is satisfied that the geological model honours the current geological interpretation and knowledge of the deposit. The location of the samples and the assay data are sufficiently reliable to support resource evaluation.

For mineralization in domains exhibiting good geological continuity using adequate drill hole spacing, the QP considers that blocks estimated during the first estimation pass using a minimum of four drill holes, an average distance of <15 m and a kriging variance (KV) of<0.3, to be classified as the Measured category. KV provides a relative measure of accuracy of the local kriged estimate with respect to data coverage.

Mineralization in domains exhibiting good geological continuity estimated during Pass 2 with a minimum of four drill holes were classified as Indicated.

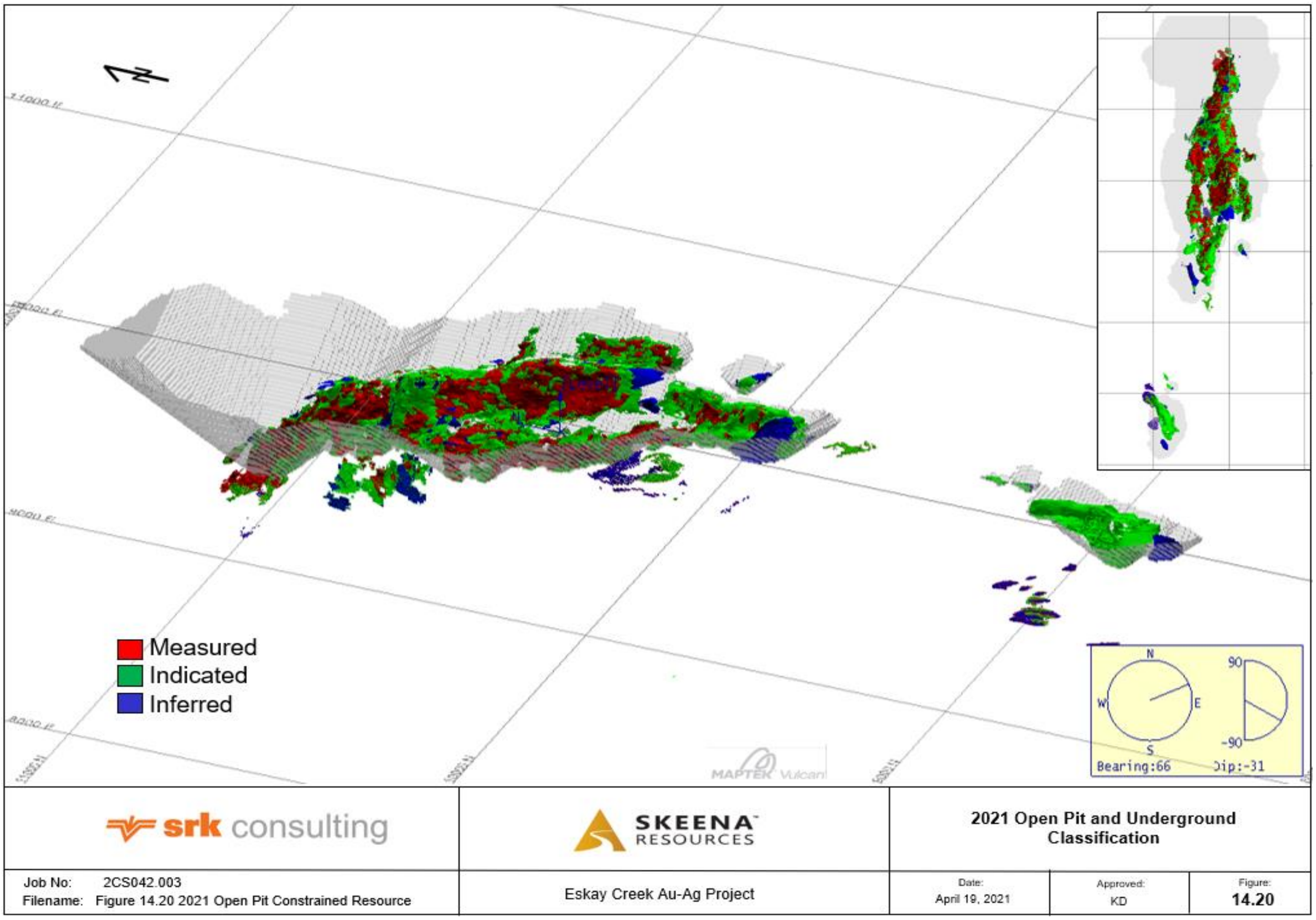
For Measured and Indicated blocks, the level of confidence is adequate for evaluating the potential economic viability of the deposit, as well as suitable for assessing technical and economic parameters to support mine planning.

Blocks estimated during Pass 3 pass, using search distances of 2.5 times the variogram range, and a KV of I<0.8 were classified in the Inferred category. For those blocks, the level of confidence is inadequate for evaluating the potential economic viability of the deposit, as well as unsuitable for assessing technical and economic parameters to support mine planning.

No Measured or Indicated Mineral Resources were classified in the low-grade envelope. Blocks in the low-grade envelope were classified as Inferred only if a minimum of three drill holes were used.

Figure 14-20 shows the distribution of the Measured, Indicated, and Inferred Mineral Resources in the open pit-constrained model.

Figure 14-20: Long Section View of the Mineral Resource Classification in Blocks Looking East in the Open Pit Model (looking east)



14.14 Mineral Resource Statement

The QP for the resource estimate is Ms. S. Ulansky, Senior Resource Geologist, P.Geo (EGBC#36085), an employee of SRK Consulting.

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014) defines a mineral resource as:

"(A) concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling".

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at appropriate cut-offs considering extraction scenarios and processing recoveries. To meet this requirement, SRK considers that major portions of the Eskay Creek Project are potentially amenable to open pit extraction, and minor areas are potentially amenable to underground mining.

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by open pit methods, SRK used a pit optimizer and reasonable mining assumptions to evaluate the proportion of the block model (Measured, Indicated, and Inferred blocks) that could be "reasonably expected" to be mined from the open pit.

The optimization parameters were selected based on experience, and benchmarking against similar projects (Table 14-22). Results from the pit optimization are used solely for testing "reasonable prospects for eventual economic extraction" by open pit methods. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

Table 14-22: Open Pit Constrained Scenario Assumptions Considered for Determining Cut-off Grades with Reasonable Prospects of Eventual Economic Extraction

Parameter	Value	Unit
Overall Pit Wall Angles	45	Degrees
Reference Mining Cost	3.00	US Dollars Per Tonne Mined
Processing Cost	15.50	US Dollars Per Tonne Processed
General and Administrative	6.00	US Dollars Per Tonne Processed
Mining Dilution	5	Percent
Mining Recovery	95	Percent
Gold Process Recovery	90	Percent
Silver Process Recovery	80	Percent
Sell Price Gold	1,700.00 x (0.95)	US Dollars Per Ounce (95% Payable)
Sell Price Silver	23.00 x (0.95)	US Dollars Per Ounce (95% Payable)
Transportation/Refining Costs	25	US Dollars Per Ounce AuEQ
Strip Ratio	9.89:1	Unitless

The block model quantities and grade estimates were also reviewed to determine the portions of the Eskay Creek Project having "reasonable prospects for eventual economic extraction" using a long-hole underground mining scenario. The parameters are summarized in Table 14-23.

Table 14-23: Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Mining costs	80	US Dollars Per Tonne Mined
Process cost	25	US Dollars Per Tonne Milled
General and Administrative	12	US Dollars Per Tonne Milled
All In Costs	117	US Dollars Per Tonne Milled
Process recovery Au	90	Percent
Process recovery Ag	80	Percent
Sell Price Gold	1700.00 x (0.95)	US Dollars Per Ounce (95% Payable)
Sell Price Silver	23.00 x (0.95)	US Dollars Per Ounce (95% Payable)
Transportation/Refining Costs	25	US Dollars Per Ounce AuEQ
Minimum Mining		Longhole: 5 m (L) x 10 m (H) x 2 m (W) Drift and Fill: 4 m (L) x 4 m (H) x 4 m (W)

The cut-off grade for the open pit model, using the parameters presented in Table 14-22, was determined to be 0.66 g/t AuEQ; however, a cut-off grade of 0.7 g/t AuEQ was selected for the estimate reporting. The long-hole mining and drift-and-fill underground mining method cut-off grades were calculated to be 2.4 g/t AuEQ and 2.8 g/t AuEQ, respectively. In the underground scenario, the steeply-dipping Water Tower Zone was determined to be amenable to the long-hole method, while the NEX, HW, 22 and LP Zones were more amenable to the drift-and-fill mining method.

The Mineral Resources amenable to open pit mining are presented in Table 14-24 and the Mineral Resource amenable to underground mining are presented in

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
Measured							
21A	1,863	4.9	3.9	71.8	291	233	4,303
21C	4,497	3.6	2.9	51.4	524	423	7,425
21B	1,997	10.9	7.4	257.5	697	474	16,533
21Be	1,640	8.8	5.8	220.5	462	305	11,630
21E	743	3.2	2.2	75.0	77	52	1,793
HW	919	5.8	3.6	163.9	172	107	4,840
NEX	4,540	5.5	3.8	125.2	804	557	18,271
WT	67	3.4	3.0	31.2	7	6	67
PMP	239	5.6	4.3	95.1	43	33	731
109	754	5.5	5.3	12.4	132	128	300
LP	52	1.2	1.1	9.2	2	2	15
Total Measured	**17,312**	**5.8**	**4.2**	**118**	**3,213**	**2,322**	**65,908**
Indicated							
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	3,764	3.4	2.7	46.1	406	330	5,583
21C	1,648	2.6	2.1	38.4	139	112	2,036

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
21B	3,100	3.9	2.9	75.3	390	289	7,501
21Be	848	5.1	3.9	92.4	140	105	2,522
21E	642	2.7	1.8	60.8	55	38	1,235
HW	1,470	3.9	2.5	104.5	185	118	4,938
NEX	3,171	2.4	1.8	40.3	244	188	4,104
WT	290	2.5	2.2	23.0	23	20	214
PMP	198	3.2	2.6	47.9	21	16	305
109	301	2.2	2.0	12.1	21	19	117
LP	1,465	1.1	0.9	9.6	51	45	545
Total Indicated	**20,342**	**2.9**	**2.2**	**52.5**	**1,903**	**1,439**	**34,362**
Measured + Indicated							
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	5,627	3.8	3.1	54.6	696	563	9,887
21C	6,145	3.4	2.7	47.9	663	535	9,461
21B	5,096	6.6	4.7	146.7	1,087	762	24,033
21Be	2,489	7.5	5.1	176.8	602	411	14,152
21E	1,385	2.9	2.0	68.4	131	90	3,047
HW	2,388	4.7	2.9	127.3	357	225	9,778
NEX	7,711	4.2	3.0	90.3	1,048	746	22,375
WT	358	2.7	2.3	24.5	31	27	282
PMP	437	4.5	3.5	73.7	64	50	1,036
109	1,055	4.5	4.3	12.3	153	148	416
LP	1,517	1.1	0.9	9.6	53	46	470
Total M + I	**37,654**	**4.2**	**3.1**	**82.8**	**5,116**	**3,761**	**100,270**
Inferred							
ENV	2,836	1.1	0.8	17.1	98	77	1,562
22	316	1.4	1.0	26.2	14	10	266
21A	938	1.1	0.8	24.5	34	24	739
21C	50	3.0	2.3	53.0	5	4	86
21B	564	2.0	1.6	26.0	36	30	471
21Be	22	3.3	2.7	41.0	2	2	29
21E	6	2.5	1.9	42.9	0.5	0.3	9
HW	324	3.3	2.0	92.0	34	21	958
NEX	30	2.5	2.1	25.7	2	2	25
WT	0.06	1.2	1.1	8.6	0.03	0.02	0.02
PMP	7	3.2	2.2	74.4	0.7	0.5	17
109	0.1	1.6	1.6	3.7	0.06	0.06	0.0
LP	145	1.0	2.3	9.0	5	4	40
Total Inferred	**5,239**	**1.4**	**1.0**	**25.0**	**231**	**174**	**4,203**

Table 14-25. The Mineral Resource considered potentially amenable to underground mining are reported exclusive of those Mineral Resources potentially amenable to open pit mining. In addition, mineralization that occurred within any historical workings, including an additional 0.20 m surrounding shell in the open pit model, were excluded from the open pit Mineral Resources tabulation. In the underground model, all mineralization that occurred within any historical workings, including a 1.0 m surrounding shell, was excluded from the underground Mineral Resource tabulation.

Table 14-24 presents the open pit constrained resources at a 0.7 g/t AuEQ cut-off grade outside of the 0.2 m exclusion zone and is shown in Figure 14-21. Table 14-25 shows the resources potentially amenable to underground mining methods above the 2.4 g/t AuEQ cut-off grade for long-hole mining, and 2.8 g/t AuEQ cut-off grade for drift and fill mining, outside the 1 m exclusion zone. The underground resource in shown in Figure 14-22.

Table 14-24: Open Pit Constrained* Mineral Resource Statement Reported at 0.7g/t AuEQ Cut-Off Grade by Domain

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
Measured							
21A	1,863	4.9	3.9	71.8	291	233	4,303
21C	4,497	3.6	2.9	51.4	524	423	7,425
21B	1,997	10.9	7.4	257.5	697	474	16,533
21Be	1,640	8.8	5.8	220.5	462	305	11,630
21E	743	3.2	2.2	75.0	77	52	1,793
HW	919	5.8	3.6	163.9	172	107	4,840
NEX	4,540	5.5	3.8	125.2	804	557	18,271
WT	67	3.4	3.0	31.2	7	6	67
PMP	239	5.6	4.3	95.1	43	33	731
109	754	5.5	5.3	12.4	132	128	300
LP	52	1.2	1.1	9.2	2	2	15
Total Measured	17,312	5.8	4.2	118	3,213	2,322	65,908
Indicated							
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	3,764	3.4	2.7	46.1	406	330	5,583
21C	1,648	2.6	2.1	38.4	139	112	2,036
21B	3,100	3.9	2.9	75.3	390	289	7,501
21Be	848	5.1	3.9	92.4	140	105	2,522
21E	642	2.7	1.8	60.8	55	38	1,235
HW	1,470	3.9	2.5	104.5	185	118	4,938
NEX	3,171	2.4	1.8	40.3	244	188	4,104
WT	290	2.5	2.2	23.0	23	20	214
PMP	198	3.2	2.6	47.9	21	16	305
109	301	2.2	2.0	12.1	21	19	117
LP	1,465	1.1	0.9	9.6	51	45	545
Total Indicated	20,342	2.9	2.2	52.5	1,903	1,439	34,362
Measured + Indicated							
22	3,445	2.1	1.4	48.2	230	158	5,334
21A	5,627	3.8	3.1	54.6	696	563	9,887
21C	6,145	3.4	2.7	47.9	663	535	9,461
21B	5,096	6.6	4.7	146.7	1,087	762	24,033
21Be	2,489	7.5	5.1	176.8	602	411	14,152
21E	1,385	2.9	2.0	68.4	131	90	3,047
HW	2,388	4.7	2.9	127.3	357	225	9,778
NEX	7,711	4.2	3.0	90.3	1,048	746	22,375
WT	358	2.7	2.3	24.5	31	27	282
PMP	437	4.5	3.5	73.7	64	50	1,036
109	1,055	4.5	4.3	12.3	153	148	416
LP	1,517	1.1	0.9	9.6	53	46	470
Total M + I	37,654	4.2	3.1	82.8	5,116	3,761	100,270
Inferred							
ENV	2,836	1.1	0.8	17.1	98	77	1,562
22	316	1.4	1.0	26.2	14	10	266
21A	938	1.1	0.8	24.5	34	24	739
21C	50	3.0	2.3	53.0	5	4	86
21B	564	2.0	1.6	26.0	36	30	471
21Be	22	3.3	2.7	41.0	2	2	29
21E	6	2.5	1.9	42.9	0.5	0.3	9

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
HW	324	3.3	2.0	92.0	34	21	958
NEX	30	2.5	2.1	25.7	2	2	25
WT	0.06	1.2	1.1	8.6	0.03	0.02	0.02
PMP	7	3.2	2.2	74.4	0.7	0.5	17
109	0.1	1.6	1.6	3.7	0.06	0.06	0.0
LP	145	1.0	2.3	9.0	5	4	40
Total Inferred	5,239	1.4	1.0	25.0	231	174	4,203

Table 14-25: Underground* Mineral Resource Statement Reported at a 2.4 g/t AuEQ Cut-Off Grade for Long-Hole Mining and 2.8 g/t AuEQ Cut-Off Grade for Drift and Fill Mining

Domain	Tonnes (000)	Grade			Contained Ounces		
		AuEQ g/t	Au g/t	Ag g/t	AuEQ Oz (000)	Au Oz (000)	Ag Oz (000)
Measured							
WT	102	6.0	5.9	13.3	20	19	44
HW	19	5.7	4.5	95.3	3	3	57
NEX	222	6.2	5.0	90.3	44	36	645
LP	2	6.7	6.4	18.7	0.5	0.4	1
Total Measured	**345**	**6.1**	**5.2**	**67.3**	**68**	**58**	**747**
Indicated							
WT	215	5.4	5.3	10.4	38	37	72
22	61	6.5	4.9	117.2	13	10	230
HW	20	5.9	4.7	94.0	4	3	62
NEX	87	5.7	5.0	54.4	16	14	152
LP	123	4.3	4.1	17.0	17	16	67
Total Indicated	**506**	**5.3**	**4.9**	**35.8**	**87**	**79**	**583**
Measured + Indicated							
22	61	6.5	4.9	117.2	13	10	230
WT	317	5.6	5.5	11.3	58	56	116
HW	39	5.9	4.6	94.6	7	6	119
NEX	309	6.1	5.0	80.1	60	50	797
LP	125	4.3	4.1	17.0	17	16	68
Total M + I	**851**	**5.7**	**5.0**	**48.6**	**155**	**137**	**1,330**
Inferred							
WT	79	4.6	4.5	7.2	12	11	18
22	221	5.5	4.1	99.4	39	29	706
HW	1	5.3	4.2	83.1	103	81	2
LP	129	4.0	3.8	14.6	17	16	61
Total Inferred	**429**	**4.9**	**4.1**	**57.0**	**67**	**57**	**787**

*Notes to accompany the Mineral Resource estimate statement:

- Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- The Qualified Person for the estimate is Ms. S Ulansky, P.Geo of SRK Consulting (Canada) who reviewed and validated the Mineral Resource estimate.
- The effective date of the Mineral Resource estimate is April 7, 2021.
- The number of metric tonnes and ounces were rounded to the nearest thousand. Any discrepancies in the totals are due to rounding.
- Open pit-constrained Mineral Resources are reported in relation to a conceptual pit shell.
- Reported underground Mineral Resources are exclusive of the Mineral Resources reported within the conceptual pit shell and reported using stope optimized shapes based on long-hole and drift-and-fill mining methods.
- Block tonnage was estimated from average specific gravity measurements using lithology groupings.

- All composites were capped where appropriate.
- Mineral Resources potentially amenable to open pit mining methods are reported at a cut-off grade of 0.7 g/t AuEQ and Mineral Resources potentially amenable to underground mining methods are reported at a cut-off grade of 2.4 g/t AuEQ for long-hole methods and 2.8 g/t AuEQ for drift-and-fill methods.
- Cut-off grades are based on a price of US$1,700/oz Au US$23/oz Ag, and gold recoveries of 90%, silver recoveries of 80% and without considering revenues from other metals. AuEQ = Au (g/t) + (Ag (g/t)/74).
- Open pit key assumptions for reasonable prospects of eventual economic extraction are as follows:
    - An overall pit wall angle of 45º;
    - A reference mining cost of US$3.00/t mined;
    - A processing cost of US$15.50/t processed;
    - General and administrative costs of US$6.00/t processed;
    - Mining dilution of 5%;
    - Mining recovery of 95%;
    - Transportation and refining costs of US$25/oz AuEQ;
- Underground key assumptions for reasonable prospects for eventual economic extraction are as follows:
    - A reference mining cost of US$80/t mined;
    - A processing cost of US$25/t milled;
    - General and administrative costs of US$12/t milled;
    - All in costs of US$117/t milled;
    - Transportation and refining costs of US$25/oz AuEQ;
- Estimates use metric units (metres, tonnes and g/t). Metals are reported in troy ounces (metric tonne * grade / 31.10348).
- The 2014 CIM Definition Standards were used for the reporting of Mineral Resources.
- Neither Skeena nor SRK is aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimates.

Figure 14-21: Oblique View of Open Pit Mineral Resources at a 0.7 g/t AuEq Cut-Off Grade

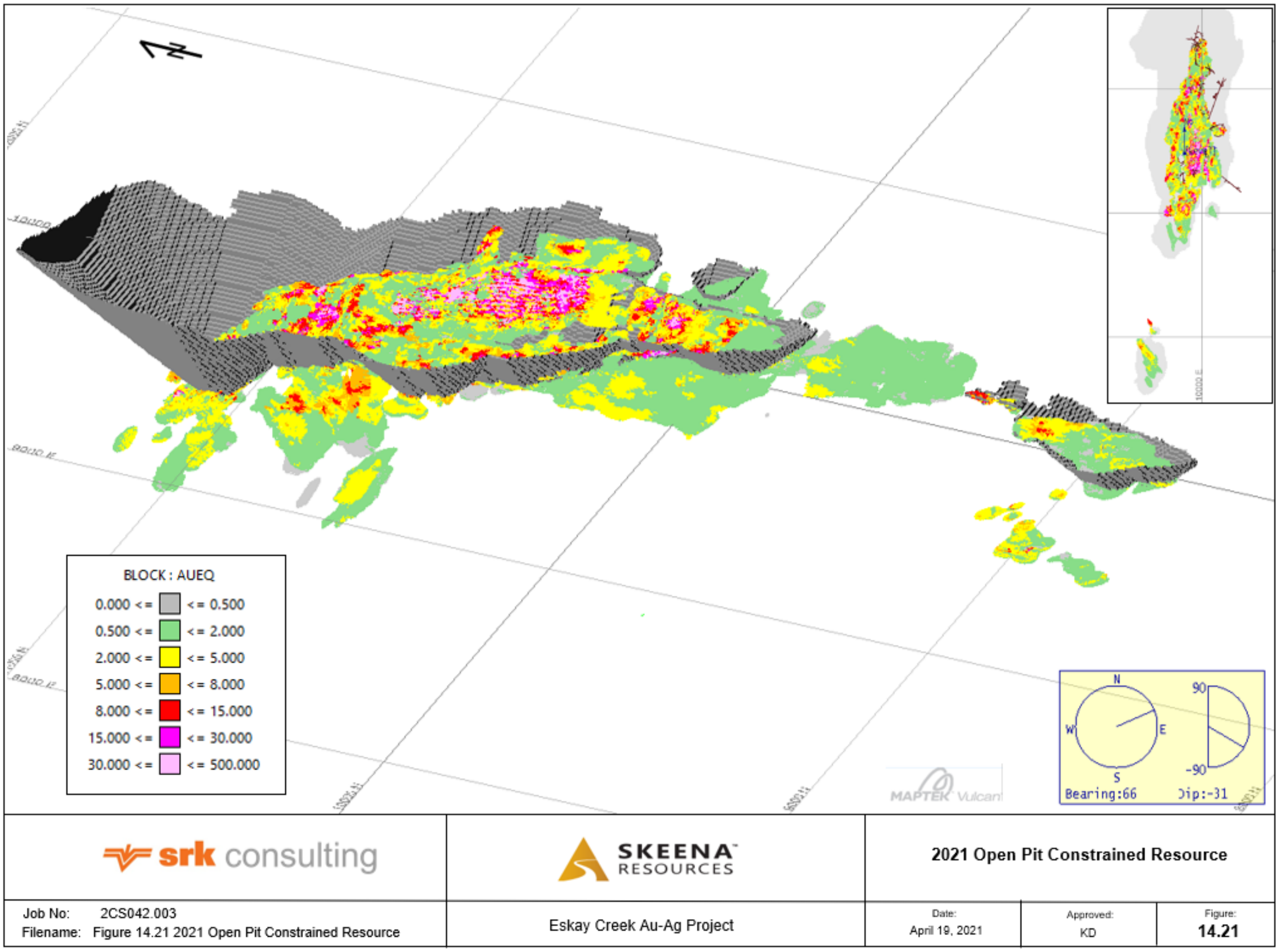
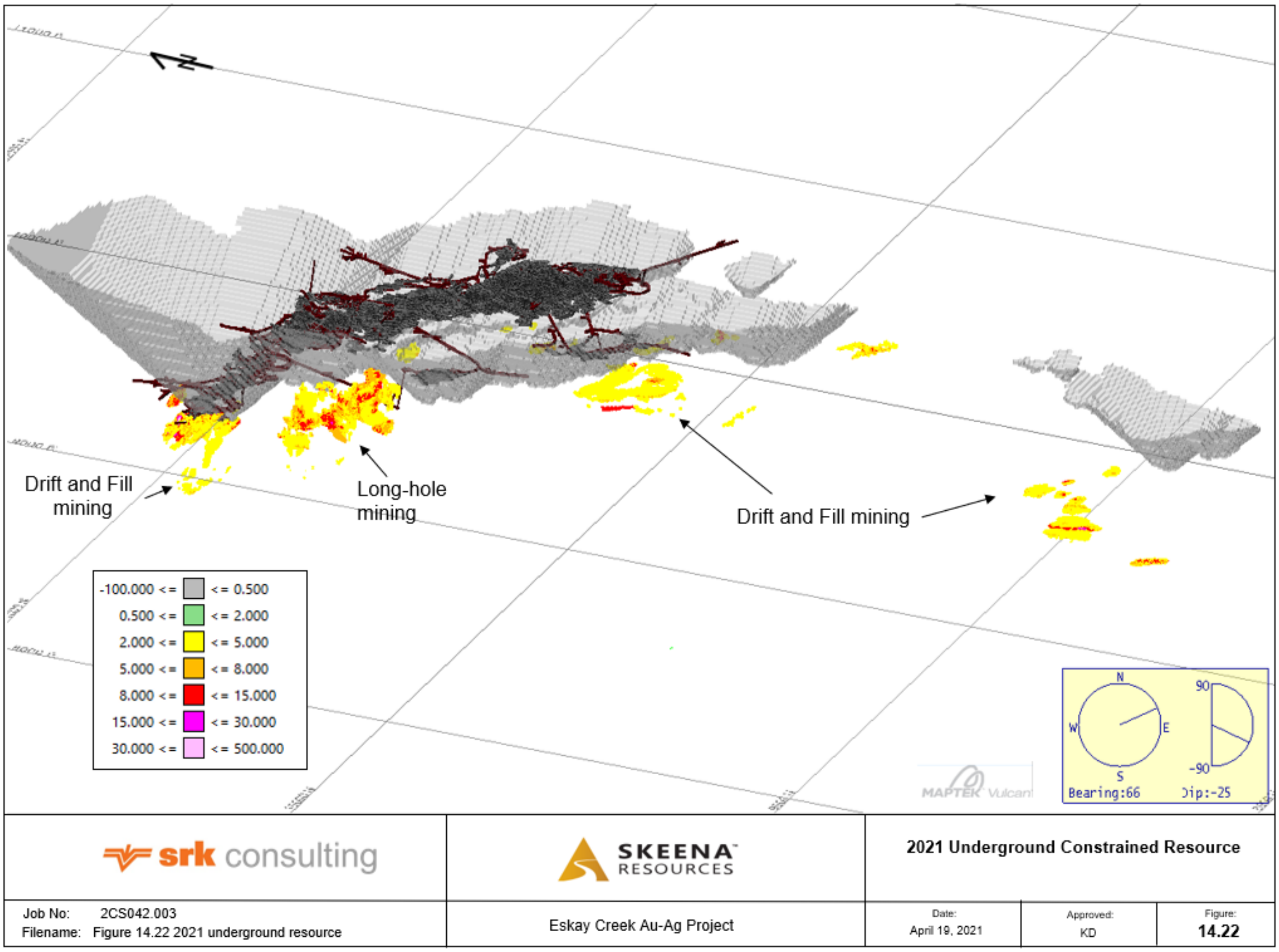


Figure 14-22: Oblique view of Underground Mineral Resources Remaining at a 2.4 g/t AuEQ Cut-Off Grade for Long-Hole Mining and 2.8 g/t AuEQ Cut-Off Grade for Drift-and-Fill Mining

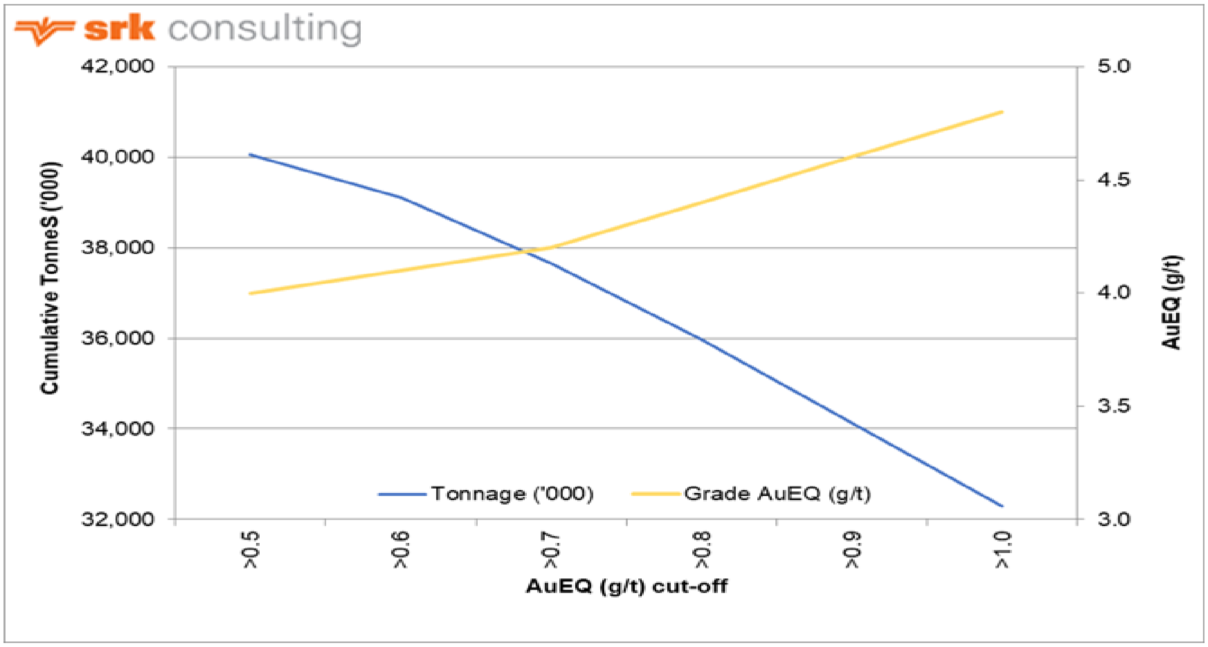


14.15 Grade Sensitivity Analysis

The Eskay Creek Mineral Resources were assessed in terms of cut-off grade selection by means of sensitivity analyses.

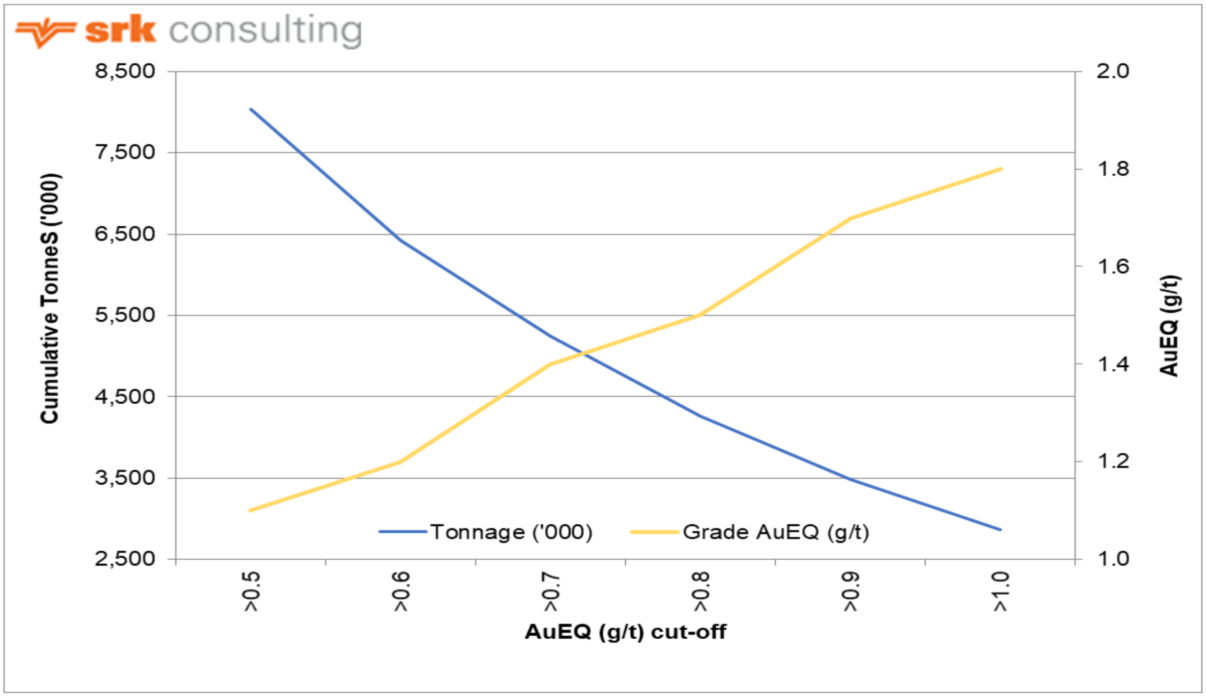
To illustrate this sensitivity, the global block model quantities and grade estimates are displayed at different cut-off grades in the open pit model as grade–tonnage curves in Figure 14-23 and Figure 14-24. The figures show that the Mineral Resource estimate is not sensitive to minor adjustments in cut-off grade selection as the average grade or the zone grades are substantially higher than the selected cut-offs and a significant difference in tonnage and ounces is not demonstrated. The reader is cautioned that numbers in the figures presented should not be misconstrued with a Mineral Resource statement apart from the base case scenario at 0.7 g/t AuEQ.

Figure 14-23: Open Pit Model Measured + Indicated Grade–Tonnage Sensitivity Curve



Note: Figure prepared SRK, 2021

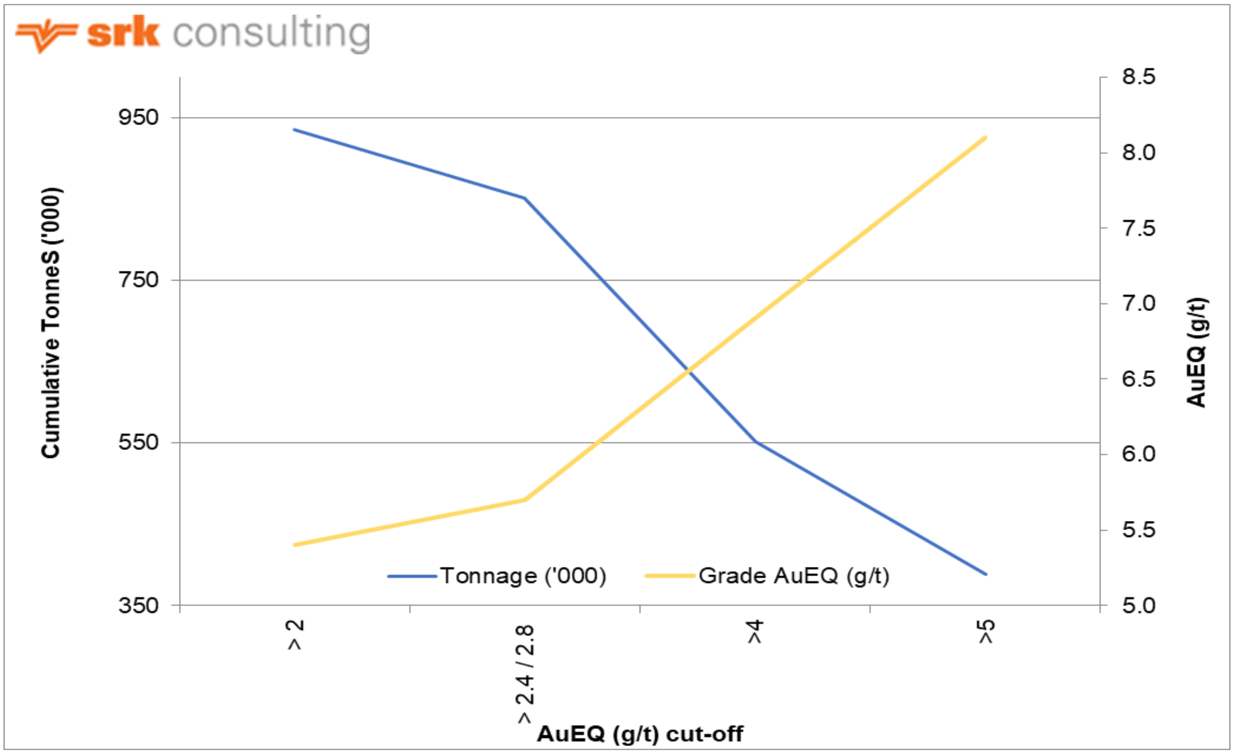
Figure 14-24: Open Pit Model Inferred Grade–Tonnage Sensitivity Curve



Note: Figure prepared by SRK, 2021

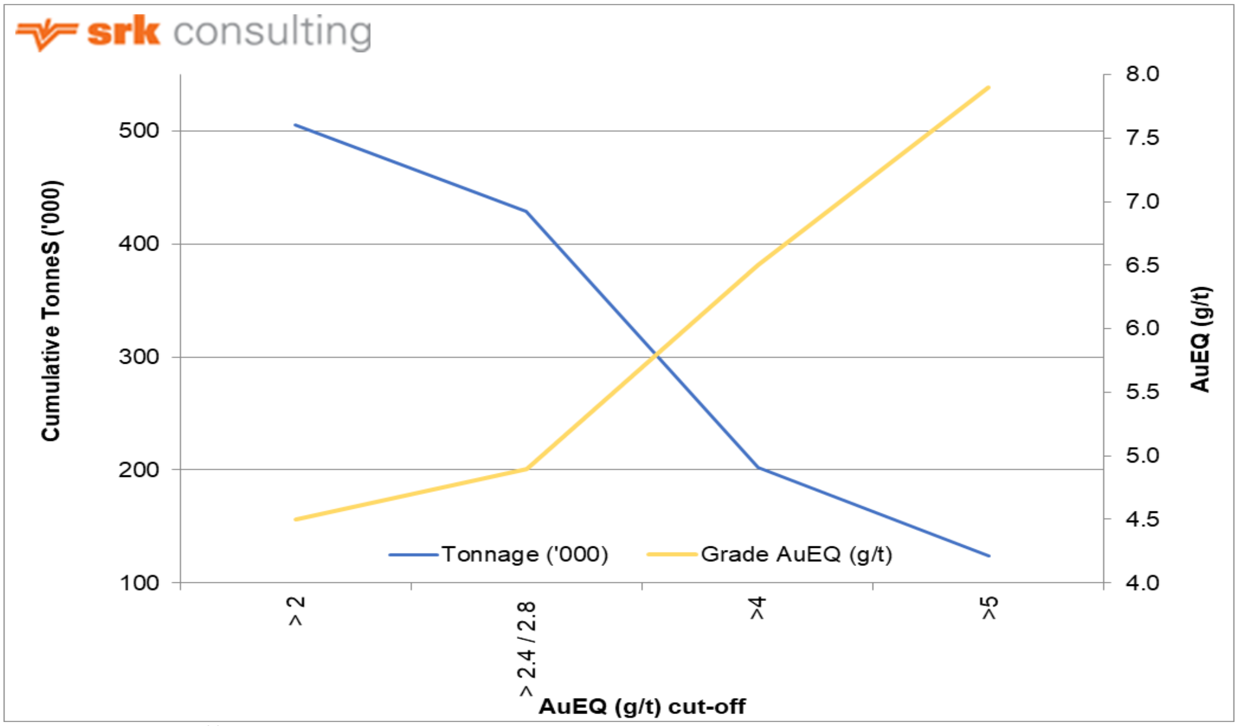
Figure 14-25 and Figure 14-26 presents global block model quantities and grade estimates within the underground model at different cut-off grades. The underground scenario is more sensitive to adjustments in cut-off grade selection due to the higher cut-off grades and selectivity of the mining methods. The reader is cautioned that the values presented in these figures should not be misconstrued with a Mineral Resource statement apart from the base case scenario at 2.4 g/t AuEq for long-hole mining and 2.8 g/t AuEq for drift-and-fill mining.

Figure 14-25: Underground Model Measured + Indicated Grade–Tonnage Sensitivity Curve



Note: Figure prepared by SRK, 2021

Figure 14-26: **Underground Model Inferred Grade–Tonnage Sensitivity Curve**



Note: Figure prepared by SRK, 2021

14.16 Reconciliation to Previous Mineral Resource Model

The large increase in the 2021 Mineral Resource estimate versus the 2019 estimate is a direct function of the expansion of the conceptual open pit to the north into the NEX Zone. In addition, several changes were made to the 2021 estimation methodology, including:

- The geological model and resource domain modelling were updated;
- The mudstones within the Hanging-Wall Andesite and the lower package beneath the Rhyolite was modelled;
- The 1 m geotechnical buffer around the mined-out stopes and lifts was reduced to 0.2 m in the open pit-constrained model and reduced from 3 m to 1 m in the underground model;
- 705 additional drill holes from the 22, 21A, 21C, 21B, 21E, HW, WT, PMP and LP domains;
- A change in classification strategy to include Measured material;
- The cut-off grade for the mineral resource potentially amenable to underground mining was reduced from 5.0 g/t AuEQ to 2.4 and 2.8 g/t AuEQ for long-hole and drift-and-fill mining methods, respectively;
- The 2021 estimate used specific gravity values based on lithology mean values, whereas the 2019 estimate used an empirical formula for density, which was less reliable for correlation.

14.17 Epithermal, Base Metal, and Metallurgical Estimates in the Pit Model for Metallurgical Characterization

The epithermal suite of elements (antimony, mercury, and arsenic), base metals (lead, copper, and zinc) and metallurgical elements (iron and sulphur) were estimated into the open pit block model to provide results for the metallurgical study. A high degree of variability of the epithermal elements exists between the different zones and rocktypes, and elevated concentrations occur in localized zones/pods. The Contact Mudstone lithology within the 21A and 21B Zones have elevated levels of arsenic, mercury, and antimony. The 21A Zone is geologically and geochemically equivalent to the 21B Zone, an area that accounted for the bulk of mineralization historically mined at Eskay Creek. Smelter penalties for the elevated concentrations of arsenic, mercury, and antimony in the 21B Zone were often prevented via blending with material from other zones while maintaining a profitable head grade (Barrick, 2004).

14.17.1 Epithermal, Base Metal and Metallurgical Elements Data Analysis

For all drilling campaigns prior to Skeena's Project involvement, iron and sulphur were not analysed. The epithermal and base metal elements were selectively sampled. Historical documentation notes that these elements were analysed when AuEQ >8 g/t; however, this was not always the case. This selective sampling process resulted in a dataset that is biased towards higher-grade material because lower-grade sample intervals were mostly excluded. The sampling inconsistencies are evident for all historical drilling campaigns, where the mineralization zones were either fully sampled, not sampled or intervals were selectively sampled. Historically, interval percentages ranged from 98% in the 22 Zone to as low as 19% in the 21E Zone. Infill drilling in the 21A, 21C, 21B, 21E, HW, and PMP Zones has improved interval percentages, giving greater confidence in these mining domains. Table 14-26 through

Domain	Zone	No. of Gold Assays	Lead		Copper		Zinc	
			No. of Lead Assays	%	No. of Copper Assays	%	No. of Zinc Assays	%
22 Zone	101	4,351	4,351	100	4,351	100	4,351	100
21A	201	9,756	6,782	70	6,782	70	6,782	70
	202	1,103	836	76	836	76	836	76
	203	5	0	0	0	0	0	0
21C	301	29,787	11,328	38	11,328	38	11,327	38
	302	5,730	2,946	51	2,946	51	2,946	51
	303	1,533	1,049	68	1,049	68	1,049	68
21B	401	21,723	8,181	38	8,181	38	8,180	38
	402	16,710	8,192	49	8,192	49	8,202	49
21Be	501	19,714	6,896	35	6,896	35	6,897	35
	502	8,505	3,295	39	3,295	39	3,300	39
21E	601	1,762	1,384	79	1,384	79	1,384	79
	602	1,110	634	57	634	57	634	57
	603	1,633	1,207	74	1,207	74	1,207	74
HW	703	16,612	7,223	43	7,223	43	7,223	43
NEX	801	26,883	6,273	23	6,273	23	6,276	23
	802	24,522	7,830	32	7,830	32	7,830	32
WT	811	2,989	1,180	39	1,180	39	1,180	39
LP	90	4,518	2,247	50	2,247	50	2,247	50

Domain	Zone	No. of Gold Assays	Lead		Copper		Zinc	
			No. of Lead Assays	%	No. of Copper Assays	%	No. of Zinc Assays	%
	91	480	223	46	223	46	223	46
	92	186	178	96	178	96	178	96
	93	553	189	34	189	34	189	34
	94	1,182	453	38	453	38	453	38
PMP	95	2,868	1,325	46	1,325	46	1,325	46
109	99	13,419	5,057	38	5,057	38	5,052	38

Table 14-28 show the percent of intervals assayed for the epithermal, base metal and metallurgical elements in relation to total gold assays, within each of the zones. Figure 14-27 is a cross section of the 21A Domain showing sampling bias where drill holes are either fully sampled, non-sampled or selectively sampled.

Table 14-26: Percentage of Intervals Estimated for the Epithermal Elements in Relation to Total Gold Assays According to Zone

Domain	Zone	No. of Gold Assays	Antimony		Mercury		Arsenic	
			No. of Antimony Assays	%	No. of Mercury Assays	%	No. of Arsenic Assays	%
22 Zone	101	4,351	4,331	100	4,331	100	4,329	99
21A	201	9,756	6,489	67	6,087	62	6,536	67
	202	1,103	818	74	681	62	819	74
	203	5	0	0	0	0	0	0
21C	301	29,787	10,546	35	10,394	35	10,547	35
	302	5,730	2,665	47	2,662	46	2,677	47
	303	1,533	1,004	65	1,004	65	997	65
21B	401	21,723	7,815	36	7,526	35	6,568	30
	402	16,710	8,294	50	8,034	48	7,023	42
21Be	501	19,714	8,648	44	8,599	44	5,909	30
	502	8,505	3,187	37	3,132	37	2,486	29
21E	601	1,762	1,333	76	1,324	75	1,333	76
	602	1,110	571	51	559	50	571	51
	603	1,633	1,081	66	1,079	66	1,081	66
HW	703	16,612	6,405	39	6,257	38	5,503	33
NEX	801	26,883	5,833	22	5,738	21	5,310	20
	802	24,522	7,893	32	7,846	32	6,852	28
WT	811	2,989	885	30	876	29	885	30
LP	90	4,518	1,104	24	1,049	23	1,118	25
	91	480	192	40	192	40	192	40
	92	186	178	96	178	96	178	96
	93	553	83	15	78	14	87	16
	94	1,182	170	14	155	13	132	11
PMP	95	2,868	1,196	42	1,197	42	1,230	43
109	99	13,419	5,042	38	4,939	37	3,925	29

Table 14-27: Percentage of Intervals Estimated for the Base Metal Elements in Relation to Total Gold Assays According to Zone

Domain	Zone	No. of Gold Assays	Lead		Copper		Zinc	
			No. of Lead Assays	%	No. of Copper Assays	%	No. of Zinc Assays	%
22 Zone	101	4,351	4,351	100	4,351	100	4,351	100
21A	201	9,756	6,782	70	6,782	70	6,782	70
	202	1,103	836	76	836	76	836	76
	203	5	0	0	0	0	0	0
21C	301	29,787	11,328	38	11,328	38	11,327	38
	302	5,730	2,946	51	2,946	51	2,946	51
	303	1,533	1,049	68	1,049	68	1,049	68
21B	401	21,723	8,181	38	8,181	38	8,180	38
	402	16,710	8,192	49	8,192	49	8,202	49
21Be	501	19,714	6,896	35	6,896	35	6,897	35
	502	8,505	3,295	39	3,295	39	3,300	39
21E	601	1,762	1,384	79	1,384	79	1,384	79
	602	1,110	634	57	634	57	634	57
	603	1,633	1,207	74	1,207	74	1,207	74
HW	703	16,612	7,223	43	7,223	43	7,223	43
NEX	801	26,883	6,273	23	6,273	23	6,276	23
	802	24,522	7,830	32	7,830	32	7,830	32
WT	811	2,989	1,180	39	1,180	39	1,180	39
LP	90	4,518	2,247	50	2,247	50	2,247	50
	91	480	223	46	223	46	223	46
	92	186	178	96	178	96	178	96
	93	553	189	34	189	34	189	34
	94	1,182	453	38	453	38	453	38
PMP	95	2,868	1,325	46	1,325	46	1,325	46
109	99	13,419	5,057	38	5,057	38	5,052	38

Table 14-28: Percentage of Intervals Estimated for the Metallurgical Elements in Relation to Total Gold Assays According to Zone

Domain	Zone	No. of Gold Assays	Iron		Sulphur	
			No. of Iron Assays	%	No. of Sulphur Assays	%
22 Zone	101	4,351	3,039	70	3,039	70
21A	201	9,756	3,640	37	3,640	37
	202	1,103	401	36	401	36
	203	5	0	0	0	0

Domain	Zone	No. of Gold Assays	Iron		Sulphur	
			No. of Iron Assays	%	No. of Sulphur Assays	%
21C	301	29,787	3,477	12	3,477	12
	302	5,730	1,539	27	1,539	27
	303	1,533	485	32	485	32
21B	401	21,723	2,659	12	2,659	12
	402	16,710	699	4	699	4
21Be	501	19,714	383	2	383	2
	502	8,505	95	1	95	1
21E	601	1,762	1,166	66	1,166	66
	602	1,110	207	19	207	19
	603	1,633	635	39	635	39
HW	703	16,612	994	6	994	6
NEX	801	26,883	0	0	0	0
	802	24,522	26	0	26	0
WT	811	2,989	144	5	144	5
LP	90	4,518	442	10	442	10
	91	480	99	21	99	21
	92	186	165	89	165	89
	93	553	40	7	40	7
	94	1,182	11	1	11	1
PMP	95	2,868	138	5	138	5
109	99	13,419	9	0	9	0

Correlations between the epithermal and base metal elements, in relation to gold and silver assays per zone, were generated with the purpose of using regression techniques for the missing intervals. Relationships with gold and silver were moderate, at best, in only a select few zones. Without strong associations with either gold or silver it was not possible to generate regression relationships to populate the missing intervals. Therefore, the gold-equivalent mineralization domains were used for estimating the spatial extent of the epithermal, base metal elements and metallurgical elements as it was considered that sub-domaining would have biased the outcome due to artefacts produced by the missing samples. Variogram ranges were determined for each of the elements and this approach was considered appropriate for metallurgical characterization studies.

Table 14-29 to Table 14-31 summarize the statistical analysis of the epithermal, base metal and metallurgical elements within each of the zones.

Table 14-29: Summary Statistics for Drill Hole Epithermal Element Assays by Zone

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
Antimony ppm								
ENV	1	Rhyolite	10,679	38.9	5.6	0.1	10.7	10,100
ENV	2	-	40,934	220.7	15.2	0.0	50.0	286,000
ENV	3	-	10,746	156.7	7.1	0.0	50.0	61,800
ENV	4	-	2,302	685.4	11.1	2.5	97.0	249,000
ENV	5	-	7,552	470.5	17.8	0.2	100.0	655,000
22 Zone	101	Rhyolite	4,331	322.1	5.3	2.5	100.0	64,240

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
21A	201	Rhyolite	6,489	691.8	5.3	0.1	94.8	114,700
	202	Contact Mudstone	818	24,960.5	3.3	10.0	318.0	591,000
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	10,546	286.4	2.5	2.5	100.0	31,900
	302	Contact Mudstone	2,665	1,480.0	7.4	6.0	205.0	327,000
	303	Hanging Wall Sediments	1,004	2,954.8	3.4	2.5	300.0	149,000
21B	401	Rhyolite	7,815	3,263.9	5.5	0.0	200.0	483,500
	402	Contact Mudstone	8,294	17,118.6	2.6	12.0	900.0	545,000
21Be	501	Rhyolite	8,648	2,294.4	3.9	17.0	300.0	163,000
	502	Contact Mudstone	3,187	7,407.9	2.8	14.0	600.0	516,400
21E	601	Rhyolite	1,333	491.0	4.5	2.5	148.5	58,100
	602	Contact Mudstone	571	2,552.7	3.4	16.8	200.0	78,700
	603	Hanging Wall Sediments	1,081	12,857.1	4.5	32.0	398.5	651,000
HW	703	Hanging Wall Sediments	6,405	2,228.5	3.3	2.5	500.0	334,000
NEX	801	Rhyolite	5,833	1,620.9	4.9	10.7	200.0	230,000
	802	Contact Mudstone	7,893	2,376.6	4.8	19.0	300.0	342,000
WT	811	Rhyolite	885	335.0	5.7	8.0	100.0	46,500
LP	90	Dacite	1,104	326.6	11.2	1.1	50.0	117,600
	91	Even Lower Mudstone	192	132.3	2.5	2.5	35.0	2,670
	92	Footwall Andesite	178	19.3	1.6	2.5	9.5	233
	93	Lower Mudstone	83	219.0	1.0	13.3	121.0	1,220
	94	Rhyolite	170	191.6	2.3	12.0	100.0	4,800
PMP	95	Rhyolite	1,196	2,660.7	5.4	18.0	599.0	382,000
109	99	Rhyolite	5,042	266.3	3.6	44.0	100.0	50,800
Mercury ppm								
ENV	1	Rhyolite	10,546	2.8	2.4	0.0	1.0	311
ENV	2	-	39,657	8.9	11.8	0.0	1.0	6,820
ENV	3	-	10,534	4.4	3.6	0.0	1.0	737
ENV	4	-	2,274	9.5	4.9	0.0	1.0	969
ENV	5	-	7,426	4.0	2.5	0.0	1.0	334
22 Zone	101	Rhyolite	4,331	7.1	2.6	0.0	3.0	637
21A	201	Rhyolite	6,087	88.5	5.8	0.0	14.0	18,800
	202	Contact Mudstone	681	1,587.4	3.4	0.0	105.5	100,000
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	10,394	11.2	2.2	0.1	5.0	887
	302	Contact Mudstone	2,662	26.0	1.9	0.5	11.0	693
	303	Hanging Wall Sediments	1,004	36.3	2.1	0.5	8.0	723
21B	401	Rhyolite	7,526	140.3	6.3	0.5	14.0	34,375
	402	Contact Mudstone	8,034	904.7	2.9	0.5	66.0	44,775
21Be	501	Rhyolite	8,599	68.9	3.4	0.1	21.0	8,875
	502	Contact Mudstone	3,132	342.2	2.9	0.5	52.0	17,590
21E	601	Rhyolite	1,324	17.0	2.9	0.5	5.0	990
	602	Contact Mudstone	559	16.9	1.8	0.5	6.0	260
	603	Hanging Wall Sediments	1,079	22.0	3.2	0.5	9.0	1,898
HW	703	Hanging Wall Sediments	6,257	33.8	1.4	0.5	15.0	1,414
NEX	801	Rhyolite	5,738	26.4	2.5	0.5	10.0	1,940
	802	Contact Mudstone	7,846	38.5	2.1	0.5	14.0	2,488
WT	811	Rhyolite	876	6.4	2.4	0.1	2.0	227
LP	90	Dacite	1,049	6.6	2.1	0.0	2.0	143
	91	Even Lower Mudstone	192	5.1	2.0	0.1	1.7	63
	92	Footwall Andesite	178	1.2	1.3	0.3	0.8	13
	93	Lower Mudstone	78	25.0	1.3	1.0	10.0	142

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
	94	Rhyolite	155	14.9	1.8	0.6	5.5	223
PMP	95	Rhyolite	1,197	35.1	4.7	0.5	14.0	4,160
109	99	Rhyolite	4,939	14.0	1.2	0.5	9.0	236
Arsenic ppm								
ENV	1	Rhyolite	10,677	298.0	3.0	0.3	100.0	24,500
ENV	2	-	39,499	231.5	3.3	0.0	100.0	58,000
ENV	3	-	10,219	234.8	3.6	0.005	100.0	35,600
ENV	4	-	2,126	213.1	2.9	2.5	100.0	140,000
ENV	5	-	7,447	294.9	7.3	0.8	100.0	180,000
22 Zone	101	Rhyolite	4,329	1,072.8	3.5	10.0	281.0	155,000
21A	201	Rhyolite	6,536	635.3	6.1	6.0	146.0	162,800
	202	Contact Mudstone	819	27,413.5	2.7	10.0	3,965.0	540,000
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	10,547	243.3	1.3	2.5	200.0	7,310
	302	Contact Mudstone	2,677	661.2	3.1	2.5	312.0	47,600
	303	Hanging Wall Sediments	997	451.8	0.1	6.0	300.0	2,890
21B	401	Rhyolite	6,568	551.3	5.1	0.1	200.0	110,000
	402	Contact Mudstone	7,023	1,755.6	5.6	33.0	700.0	530,000
21Be	501	Rhyolite	5,909	1,725.7	1.5	6.0	500.0	19,500
	502	Contact Mudstone	2,486	1,280.1	1.9	50.0	600.0	60,000
21E	601	Rhyolite	1,333	349.4	1.6	12.0	175.0	5,100
	602	Contact Mudstone	571	340.1	1.9	36.0	200.0	9,500
	603	Hanging Wall Sediments	1,081	609.8	1.7	17.0	300.0	13,150
HW	703	Hanging Wall Sediments	5,503	754.8	2.2	6.0	400.0	100,000
NEX	801	Rhyolite	5,310	536.8	2.3	50.0	300.0	27,000
	802	Contact Mudstone	6,852	669.2	1.3	50.0	400.0	15,000
WT	811	Rhyolite	885	540.9	2.5	22.0	200.0	25,500
LP	90	Dacite	1,118	576.0	6.3	1.9	302.0	120,000
	91	Even Lower Mudstone	192	1,149.6	3.6	21.9	279.1	51,100
	92	Footwall Andesite	178	157.2	0.8	2.5	120.0	892
	93	Lower Mudstone	87	760.4	1.3	93.0	500.0	7,964
	94	Rhyolite	132	631.9	1.9	50.0	300.0	9,500
PMP	95	Rhyolite	1,230	594.0	1.5	21.0	308.5	10,100
109	99	Rhyolite	3,925	597.9	1.5	50.0	300.0	10,800

Table 14-30: Summary Statistics for Drill Hole Base Metal Assays by Zone

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
Lead %								
ENV	1	Rhyolite	10,707	0.015	6.8	0.000	0.002	5.230
ENV	2	-	42,680	0.660	7.7	0.000	0.005	22.200
ENV	3	-	7,791	0.152	5.5	0.000	0.010	20.950
ENV	4	-	2,831	0.174	5.2	0.000	0.010	15.600
ENV	5	-	7,923	0.014	10.6	0.000	0.005	7.680
22 Zone	101	Rhyolite	4,351	0.113	5.1	0.000	0.010	16.150
21A	201	Rhyolite	6,782	0.120	3.0	0.000	0.018	10.920
	202	Contact Mudstone	836	0.101	4.7	0.000	0.010	7.150
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	11,328	0.117	3.9	0.000	0.020	20.000
	302	Contact Mudstone	2,946	0.452	2.6	0.000	0.090	15.500

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
	303	Hanging Wall Sediments	1,049	0.939	2.8	0.000	0.030	20.200
21B	401	Rhyolite	8,181	0.507	3.3	0.000	0.030	20.000
	402	Contact Mudstone	8,192	1.975	1.8	0.000	0.260	53.150
21Be	501	Rhyolite	6,896	1.192	2.4	0.000	0.200	24.400
	502	Contact Mudstone	3,295	2.134	1.8	0.000	0.260	24.000
21E	601	Rhyolite	1,384	0.059	3.5	0.000	0.010	3.650
	602	Contact Mudstone	634	0.433	3.1	0.000	0.010	10.750
	603	Hanging Wall Sediments	1,207	0.740	6.1	0.000	0.010	7.150
HW	703	Hanging Wall Sediments	7,223	2.278	1.8	0.000	0.270	52.000
NEX	801	Rhyolite	6,273	0.842	2.4	0.001	0.130	22.620
	802	Contact Mudstone	7,830	2.068	1.8	0.001	0.410	27.720
WT	811	Rhyolite	1,180	0.103	4.8	0.000	0.010	12.590
LP	90	Dacite	2,247	0.422	2.7	0.000	0.080	20.000
	91	Even Lower Mudstone	223	0.188	2.5	0.001	0.020	3.350
	92	Footwall Andesite	178	0.119	2.8	0.000	0.010	3.290
	93	Lower Mudstone	189	0.919	1.6	0.002	0.255	10.200
	94	Rhyolite	453	0.591	1.7	0.003	0.165	7.010
PMP	95	Rhyolite	1,325	0.156	2.7	0.001	0.040	5.300
109	99	Rhyolite	5,057	1.489	1.8	0.005	0.580	65.360
Zinc %								
ENV	1	Rhyolite	10,707	0.033	4.9	0.000	0.010	6.690
ENV	2	-	42,699	0.111	7.7	0.000	0.010	44.400
ENV	3	-	7,791	0.236	5.6	0.001	0.020	32.530
ENV	4	-	2,836	0.285	5.0	0.002	0.020	22.000
ENV	5	-	7,923	0.370	5.4	0.000	0.012	13.050
22 Zone	101	Rhyolite	4,351	0.167	4.7	0.001	0.020	23.100
21A	201	Rhyolite	6,782	0.194	2.7	0.000	0.032	13.520
	202	Contact Mudstone	836	0.228	3.5	0.002	-	12.500
	203	Hanging Wall Sediments	0	-	-	-		-
21C	301	Rhyolite	11,327	0.213	3.4	0.000	0.040	22.580
	302	Contact Mudstone	2,946	0.782	2.6	0.001	0.200	27.320
	303	Hanging Wall Sediments	1,049	1.661	2.7	0.001	0.115	33.100
21B	401	Rhyolite	8,180	0.893	3.5	0.000	0.070	39.020
	402	Contact Mudstone	8,202	3.496	1.8	0.001	0.490	33.950
21Be	501	Rhyolite	6,897	1.967	2.5	0.001	0.310	43.000
	502	Contact Mudstone	3,300	3.701	1.8	0.005	0.480	39.440
21E	601	Rhyolite	1,384	0.115	3.7	0.001	0.020	10.350
	602	Contact Mudstone	634	0.796	2.9	0.004	0.080	19.080
	603	Hanging Wall Sediments	1,207	1.810	4.6	0.010	0.080	13.930
HW	703	Hanging Wall Sediments	7,223	3.400	1.8	0.001	0.460	33.680
NEX	801	Rhyolite	6,276	1.411	2.5	0.001	0.210	48.880
	802	Contact Mudstone	7,830	3.190	1.8	0.005	0.630	35.100
WT	811	Rhyolite	1,180	0.217	4.8	0.002	0.030	21.450
LP	90	Dacite	2,247	0.653	2.6	0.001	0.090	21.100
	91	Even Lower Mudstone	223	0.317	2.3	0.001	0.050	7.200
	92	Footwall Andesite	178	0.197	2.2	0.002	0.020	3.140
	93	Lower Mudstone	189	1.819	1.9	0.003	0.470	20.700
	94	Rhyolite	453	0.841	1.7	0.001	0.190	11.000
PMP	95	Rhyolite	1,325	0.297	3.3	0.001	0.080	21.000
109	99	Rhyolite	5,052	2.266	1.6	0.010	0.840	31.800

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
Copper %								
ENV	1	Rhyolite	10,679	0.003	7.8	0.000	0.001	1.487
ENV	2	-	42,461	0.010	8.9	0.000	0.005	5.520
ENV	3	-	7,774	0.021	6.6	0.000	0.010	4.180
ENV	4	-	2,824	0.031	6.6	0.000	0.010	5.200
ENV	5	-	7,924	0.007	6.2	0.000	0.005	3.200
22 Zone	101	Rhyolite	4331	0.013	4.8	0.000	0.002	1.700
21A	201	Rhyolite	6,773	0.018	3.4	0.000	0.005	1.341
	202	Contact Mudstone	836	0.023	3.8	0.000	0.010	1.510
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	11,327	0.034	3.5	0.000	0.010	5.440
	302	Contact Mudstone	2,946	0.090	2.7	0.001	0.020	4.780
	303	Hanging Wall Sediments	1,049	0.223	2.8	0.000	0.010	5.240
21B	401	Rhyolite	8,157	0.121	3.6	0.000	0.010	5.660
	402	Contact Mudstone	8,184	0.501	2.3	0.001	0.050	26.400
21Be	501	Rhyolite	6,849	0.281	3.2	0.000	0.030	10.700
	502	Contact Mudstone	3,266	0.528	2.2	0.002	0.040	9.870
21E	601	Rhyolite	1,384	0.015	4.0	0.000	0.003	1.500
	602	Contact Mudstone	634	0.130	3.0	0.000	0.010	3.950
	603	Hanging Wall Sediments	1,206	0.024	5.3	0.001	0.010	2.290
HW	703	Hanging Wall Sediments	7,193	0.348	1.9	0.000	0.040	10.000
NEX	801	Rhyolite	6,274	0.136	3.5	0.000	0.010	8.580
	802	Contact Mudstone	7,822	0.379	2.4	0.001	0.040	35.000
WT	811	Rhyolite	1,180	0.025	4.0	0.000	0.010	2.400
LP	90	Dacite	2,050	0.014	2.5	0.000	0.010	0.780
	91	Even Lower Mudstone	223	0.210	2.1	0.000	0.007	0.304
	92	Footwall Andesite	178	0.009	2.9	0.000	0.002	0.280
	93	Lower Mudstone	189	0.034	2.7	0.001	0.010	0.890
	94	Rhyolite	445	0.020	2.3	0.001	0.010	0.670
PMP	95	Rhyolite	1,325	0.060	3.3	0.000	0.010	4.220
109	99	Rhyolite	4,577	0.031	4.8	0.002	0.010	3.280

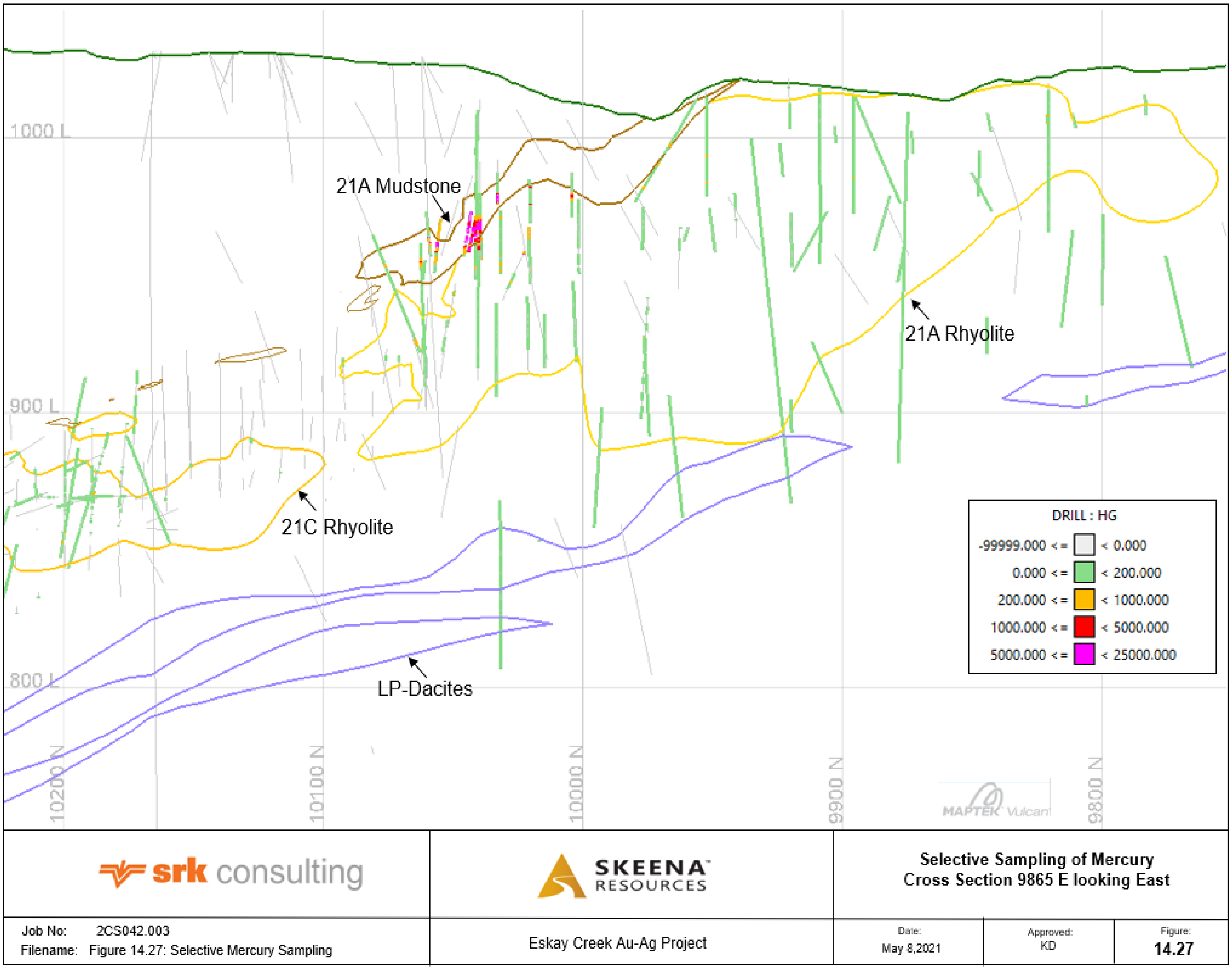
Table 14-31: Summary Statistics for Drill Hole Metallurgical Assays by Zone

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
Sulphur ppm								
ENV	1	Rhyolite	1,849	6,777.3	1.3	50	4,200	147,000
ENV	2	-	18,510	14,929.3	1.1	50	9,800	426,000
ENV	3	-	458	9,741.5	1.2	100	4,800	68,800
ENV	4	-	1,414	14,455.7	1.0	400	8,100	145,500
ENV	5	-	3,754	9,630.7	1.2	100	5,600	273,000
22 Zone	101	Rhyolite	3,039	10,035.4	1.7	100	5,200	223,000
21A	201	Rhyolite	3,640	15,037.1	0.8	1,500	12,200	230,000
	202	Contact Mudstone	401	39,629.9	1.3	3,800	22,700	271,000
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	3,477	11,905.8	0.8	400	9,900	154,500
	302	Contact Mudstone	1,539	39,436.6	0.7	800	36,500	192,000
	303	Hanging Wall Sediments	485	35,867.0	0.8	800	31,500	202,000

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
21B	401	Rhyolite	2,659	12,149.9	0.5	1,500	11,000	80,100
	402	Contact Mudstone	699	30,868.5	0.5	2,500	29,450	151,500
21Be	501	Rhyolite	383	23,150	1.8	300	10,150	365,000
	502	Contact Mudstone	95	39,603	0.7	5,800	33,550	226,000
21E	601	Rhyolite	1,166	9,078	0.6	100	8,000	81,800
	602	Contact Mudstone	207	16,897	0.6	1,400	15,950	47,800
	603	Hanging Wall Sediments	635	29,610	1.0	600	25,050	278,000
HW	703	Hanging Wall Sediments	994	36,112	0.8	1,000	29,400	276,000
NEX	801	Rhyolite	0	-	-	-	-	-
	802	Contact Mudstone	26	31,700	0.4	13,900	29,600	59,900
WT	811	Rhyolite	144	18,887	1.0	21,000	9,700	88,800
LP	90	Dacite	442	863,88	0.8	7,100	61,500	439,000
	91	Even Lower Mudstone	99	53,055	0.6	8,900	40,450	162,500
	92	Footwall Andesite	165	40,632	0.7	1,700	32,250	191,000
	93	Lower Mudstone	40	150,217	0.7	8,000	136,500	377,000
	94	Rhyolite	11	16,290	0.5	8,500	12,200	34,300
PMP	95	Rhyolite	138	10,296	0.6	1,500	9,100	53,000
109	99	Rhyolite	9	51,277	0.6	34,600	39,500	131,500
Iron %								
ENV	1	Rhyolite	1,849	1.2	0.7	0.370	1.030	14.750
ENV	2	-	18,510	3.9	0.7	0.130	3.210	31.400
ENV	3	-	458	3.2	0.	0.830	2.290	10.550
ENV	4	-	1,414	5.6	5.7	0.400	6.370	13.950
ENV	5	-	3,754	4.3	0.7	0.400	3.540	10.300
22 Zone	101	Rhyolite	3,039	1.3	0.9	0.340	0.980	15.500
21A	201	Rhyolite	3,640	1.4	0.5	0.280	1.290	18.650
	202	Contact Mudstone	401	2.1	0.4	0.210	1.995	7.850
	203	Hanging Wall Sediments	0	-	-	-	-	-
21C	301	Rhyolite	3,477	1.5	0.5	0.290	1.270	9.690
	302	Contact Mudstone	1,539	3.0	0.5	0.070	2.775	14.700
	303	Hanging Wall Sediments	485	5.4	0.5	0.130	5.120	17.700
21B	401	Rhyolite	2,659	1.3	0.3	0.490	1.250	6.530
	402	Contact Mudstone	699	3.2	0.5	0.850	3.080	12.450
21Be	501	Rhyolite	383	2.4	1.5	0.340	1.285	30.100
	502	Contact Mudstone	95	4.3	0.5	1.510	3.800	9.810
21E	601	Rhyolite	1,166	1.2	0.6	0.380	1.060	8.470
	602	Contact Mudstone	207	3.2	0.6	0.940	2.585	9.940
	603	Hanging Wall Sediments	635	3.8	0.5	0.380	3.320	14.750
HW	703	Hanging Wall Sediments	994	4.5	0.6	0.100	4.040	17.450
NEX	801	Rhyolite	0	-	-	-	-	-
	802	Contact Mudstone	26	4.0	0.3	2.680	3.540	7.450
WT	811	Rhyolite	144	3.3	0.9	0.770	1.650	10.600
LP	90	Dacite	442	8.6	0.6	2.310	6.680	36.700
	91	Even Lower Mudstone	99	5.7	0.5	1.480	4.800	15.750
	92	Footwall Andesite	165	5.5	0.4	1.680	4.990	15.200
	93	Lower Mudstone	40	12.6	0.6	1.090	11.700	30.400
	94	Rhyolite	11	1.4	0.3	1.010	1.230	2.250

Domain	Zone	Rocktype	No. of Samples	Mean	CV	Min	Median	Max
PMP	95	Rhyolite	138	1.2	0.3	0.620	1.100	3.100
109	99	Rhyolite	9	5.1	0.03	2.760	4.565	7.390

Figure 14-27: Long Section 9865 E Showing Selective Sampling of Mercury in the 21A and 21C Domains Within a 20 m Window (unsampled drill hole traces are shown in grey)



14.17.2 Compositing

Epithermal, base metal and metallurgical elements were composited to 2 m, using the same intervals determined for gold and silver composites. Since the epithermal and base metal elements are all considered penalty elements, a conservative approach was undertaken for compositing. To ensure that the estimate wasn't unduly affected by the missing or unsampled intervals, the unsampled intervals were allocated a default value of -66 (= missing) prior to compositing and ignored during estimation, thereby removing the risk of underestimating the values of the penalty elements.

14.17.3 Evaluation of Outliers

Capping of high-grade assays was applied to the epithermal, base metal and metallurgical elements by zone using the 2 m composites. High-grade capping was examined using four tools: (1) histograms, (2) log probability plots, (3) capping statistics, and (4) percent metal loss values. Visual inspections of the high-grade outliers in relation to the surrounding data was also undertaken to ensure that the locations were spatially disassociated. Less than 1% of the data was capped for high-grade outliers, excluding the low-grade envelope, which was capped more aggressively (Table 14-32 to Table 14-34). Several zones show percent metal loss values of >5%, which are the result of a limited number of extreme high-grade outlier samples.

Table 14-32: Capping Statistics for the Epithermal Elements by Zone

Domain	Zone	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
Mercury ppm											
ENV	1	7,618	177	70	1	0.0%	2.58	2.1	2.56	2.0	1%
ENV	2	260,920	9,840	500	45	0.0%	7.94	12.2	6.09	4.6	23%
ENV	3	6,579	354	100	16	0.2%	4.14	2.8	3.99	2.4	4%
ENV	4	1,692	969	100	18	1.1%	8.06	4.6	6.09	2.4	24%
ENV	5	5,115	104	70	11	0.2%	3.81	1.9	3.78	1.8	1%
22 Zone	101	2,853	292	80	7	0.2%	6.61	1.9	6.38	1.5	4%
21A	201	3,916	12,120	4,000	4	0.1%	72.69	4.7	68.00	3.5	6%
	202	343	27,370	15,000	4	1.2%	1,294.52	2.6	1,221.16	2.3	6%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	6,188	282	140	14	0.2%	9.75	1.6	9.63	1.5	1%
	302	1,424	388	250	4	0.3%	23.29	1.5	23.02	1.4	1%
	303	518	584	300	4	0.8%	28.92	2.0	28.05	1.8	3%
21B	401	4,318	28,663	5,000	12	0.3%	118.36	6.2	101.27	3.9	14%
	402	4,380	31,125	20,000	6	0.1%	793.72	2.9	786.15	2.9	1%
21Be	501	4,663	6,181	1,900	5	0.1%	60.91	2.7	59.40	2.2	2%
	502	1,822	11,328	7,000	6	0.3%	283.60	2.8	274.58	2.5	3%
21E	601	836	403	200	5	0.6%	15.15	2.1	14.64	1.8	3%
	602	302	108	80	5	1.7%	13.96	1.3	13.64	1.3	2%
	603	764	834	300	3	0.4%	17.35	2.5	16.51	2.0	5%
HW	703	3,433	600	250	6	0.2%	30.07	1.3	29.83	1.2	1%
NEX	801	3,288	819	300	7	0.2%	21.90	1.8	21.47	1.6	2%
	802	4,466	1,000	500	9	0.2%	31.92	1.7	31.66	1.6	1%
WT	811	551	176	60	10	1.8%	6.07	2.3	5.53	1.8	9%
LP	90	660	106	50	11	1.7%	6.22	1.9	5.90	1.6	5%
	91	116	56	25	5	4.3%	4.49	1.8	3.98	1.5	11%
	92	98	7	7	1	1.0%	1.18	1.0	1.18	0.1	0%
	93	43	104	80	2	4.7%	23.68	1.1	22.83	1.1	4%
	94	106	223	50	6	5.7%	16.94	1.7	13.81	1.0	18%
PMP	95	697	2,972	300	5	0.7%	28.52	4.3	23.40	1.6	18%
109	99	2,794	185	100	10	0.4%	13.89	1.0	13.82	1.0	0%
Antimony ppm											
ENV	1	7,726	8,625	1,300	8	0.1%	35.50	4.9	32.26	2.1	9%
ENV	2	26,981	136,000	6,000	89	0.3%	178.35	11.7	110.43	3.8	38%
ENV	3	6,690	61,800	6,000	17	0.3%	144.27	6.6	126.13	3.3	13%

Domain	Zone	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
ENV	4	1,712	9,353	6,000	15	0.9%	428.42	8.3	224.97	3.2	47%
ENV	5	5,197	59,355	9,000	41	0.8%	320.00	7.0	219.21	4.1	31%
22 Zone	101	2,853	42,920	15,000	3	0.1%	295.65	4.4	279.47	3.2	5%
21A	201	4,255	62,777	30,000	6	0.1%	603.78	4.2	589.13	3.9	2%
	202	456	505,959	300,000	8	1.8%	18,999.60	3.4	17,139.85	3.2	10%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	6,276	9,277	3,500	19	0.3%	256.17	1.7	253.22	1.6	1%
	302	1,426	234,300	16,000	9	0.6%	1,261.02	6.0	923.74	2.2	27%
	303	518	95,894	30,000	7	1.4%	2,297.49	3.0	2,019.48	2.3	12%
21B	401	4,474	282,314	100,000	15	0.3%	2,710.37	4.9	2,446.19	4.0	10%
	402	4,528	373,300	300,000	5	0.1%	14,504.39	2.5	14,463.21	2.5	0%
21Be	501	4,692	98,646	60,000	15	0.3%	1,999.39	3.3	1,961.67	3.2	2%
	502	1,853	197,000	110,000	6	0.3%	6,271.74	2.4	6,156.08	2.2	2%
21E	601	842	32,599	3,500	13	1.5%	425.15	3.3	358.90	1.6	16%
	602	310	42,578	25,000	5	1.6%	2,022.08	2.6	1,917.52	2.4	5%
	603	765	638,864	300,000	4	0.5%	7,476.71	5.3	6,695.33	4.5	10%
HW	703	3,519	120,665	40,000	6	0.2%	1,906.27	2.5	1,842.31	2.1	3%
NEX	801	3,337	230,000	40,000	10	0.3%	1,281.61	4.7	1,157.80	3.0	10%
	802	4,494	162,101	50,000	10	0.2%	1,800.28	3.5	1,685.52	2.7	6%
WT	811	557	31,629	2,000	7	1.3%	315.83	5.0	216.91	3.6	31%
LP	90	692	46,061	700	15	2.2%	265.09	7.3	109.39	1.3	59%
	91	116	1,899	700	6	5.2%	113.96	2.3	59.64	1.3	48%
	92	98	133	133	0	0.0%	18.20	1.3	18.20	1.3	0%
	93	46	766	500	4	8.7%	218.23	0.7	211.46	0.6	3%
	94	115	3,555	500	6	5.2%	194.31	1.8	156.73	0.8	19%
PMP	95	698	132,903	30,000	3	0.4%	1,952.67	3.8	1,656.91	2.1	15%
109	99	2,850	262	2,000	9	0.3%	261.85	2.3	242.02	1.0	8%
Arsenic ppm											
ENV	1	7,726	24,500	10,000	9	0.1%	278.07	3.0	274.24	2.8	1%
ENV	2	26,341	44,208	3,000	152	0.6%	214.60	2.9	198.75	1.8	7%
ENV	3	6,438	27,450	3,000	35	0.5%	230.12	3.2	206.09	1.8	10%
ENV	4	1,633	12,500	1,900	16	1.0%	201.46	3.0	172.33	1.6	14%
ENV	5	5,147	74,923	6,000	4	0.1%	274.81	4.4	256.35	1.9	7%
22 Zone	101	2,853	132,852,162	30,000	4	0.1%	1,046.90	3.4	986.58	2.2	6%
21A	201	4,277	135,656	30,000	6	0.1%	600.24	6.1	518.50	3.5	14%
	202	456	400,000	300,000	5	1.1%	23,723.54	2.6	23,122.73	2.5	3%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	6,278	6,100	2,000	28	0.4%	238.57	1.2	232.60	0.9	3%

Domain	Zone	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
	302	1,434	43,665	20,000	4	0.3%	622.50	3.1	591.79	2.3	5%
	303	517	1,703	1,625	2	0.4%	452.11	0.8	451.95	0.8	0%
21B	401	3,919	95,143	12,000	13	0.3%	480.69	4.6	420.04	2.4	13%
	402	3,937	512,225	30,000	13	0.3%	1,693.40	5.4	1484.22	1.9	12%
21Be	501	3,434	16,800	14,000	11	0.3%	1,711	1	1708.13	1.5	0%
	502	1,502	60,000	12,000	11	0.7%	1,219.27	2.0	1156.05	1.5	5%
21E	601	842	5,100	3,000	7	0.8%	349.87	1.5	339.81	1.4	3%
	602	310	4,808	2,000	4	1.3%	332.13	1.4	311.40	1.0	6%
	603	765	11,439	5,000	5	0.7%	540.86	1.6	526.21	1.4	3%
HW	703	3,121	100,000	6,500	5	0.2%	715.95	2.7	683.30	1.1	5%
NEX	801	3,102	24,500	7,000	10	0.3%	500.64	2.0	482.67	1.5	4%
	802	3,985	10,814	5,500	6	0.2%	618.21	1.1	615.80	1.0	0%
WT	811	557	25,500	5,000	7	1.3%	534.98	2.0	477.59	1.6	11%
LP	90	703	59,186	2,500	10	1.4%	554.35	4.2	446.23	1.0	20%
	91	116	19,872	2,000	8	6.9%	894.65	2.6	475.65	1.1	47%
	92	98	557	557	0	0.0%	151.60	0.7	151.60	0.7	0%
	93	49	3,911	2,000	2	4.1%	689.61	1.0	635.44	0.7	8%
	94	95	9,500	2,000	8	8.4%	719.44	1.6	584.82	1.0	19%
PMP	95	701	10,000	5,000	4	0.6%	570.85	1.3	557.53	1.2	2%
109	99	2,329	7,082	4,000	17	0.7%	585.66	1.2	575.94	1.2	2%

Table 14-33: Capping Statistics for the Base Metal Elements by Zone

DOMAIN	ZONE	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
Lead %											
ENV	1	7,759	3.87	0.65	26	0.3%	0.01	5.9	0.01	4.4	7%
20.97	2	28,333	20.97	3.00	99	0.3%	0.07	7.5	0.05	4.7	21%
ENV	3	4,958	19.00	5.00	65	1.3%	0.17	5.2	0.14	4.1	14%
ENV	4	2,079	15.60	4.00	21	1.0%	0.18	4.8	0.15	3.9	19%
ENV	5	5,495	7.68	0.35	20	0.4%	0.14	12.0	0.09	3.3	36%
22 Zone	101	2,870	11.58	3.00	6	0.2%	0.10	3.9	0.09	3.0	6%
21A	201	4,465	4.67	3.10	3	0.1%	0.11	2.1	0.11	2.1	1%
	202	469	4.43	1.50	4	0.9%	0.09	4.1	0.07	2.8	19%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	6,688	8.71	3.00	14	0.2%	0.10	3.0	0.10	2.4	3%
	302	1,589	9.21	4.50	13	0.8%	0.40	2.0	0.39	1.8	3%

DOMAIN	ZONE	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
	303	546	14.31	6.50	15	2.7%	0.70	2.5	0.63	2.3	9%
21B	401	4,785	20.00	11.50	19	0.4%	0.45	3.1	0.45	3.0	2%
	402	4,630	22.61	18.00	4	0.1%	1.74	1.8	1.74	1.7	0%
21Be	501	3,937	24.21	17.00	13	0.3%	1.05	2.3	1.04	2.2	1%
	502	1,963	19.95	15.00	23	1.2%	1.94	1.7	1.93	1.6	1%
21E	601	868	1.76	1.00	5	0.6%	0.05	2.7	0.05	2.5	2%
	602	345	4.62	3.00	10	2.9%	0.33	2.3	0.31	2.2	7%
	603	838	2.52	1.30	9	1.1%	0.05	4.3	0.04	3.5	14%
HW	703	4,054	32.00	18.00	9	0.2%	2.00	1.7	1.90	1.7	5%
NEX	801	3,600	16.97	9.50	23	0.6%	0.77	2.0	0.75	1.9	2%
	802	4,526	21.84	18.00	7	0.2%	1.81	1.7	1.80	1.7	0%
WT	811	709	9.05	1.50	9	1.3%	0.10	4.5	0.08	2.7	18%
LP	90	1,301	11.18	4.50	10	0.8%	0.40	2.2	0.38	1.9	5%
	91	133	2.31	1.00	4	3.0%	0.16	2.1	0.14	1.6	14%
	92	98	1.28	1.28	1	1.0%	0.11	1.9	0.11	1.9	0%
	93	107	5.58	3.00	0	0.0%	0.85	1.4	0.77	1.2	9%
	94	271	5.97	2.90	11	4.1%	0.65	1.5	0.60	1.3	9%
PMP	95	762	4.48	1.00	10	1.3%	0.13	2.1	0.12	1.5	10%
109	99	2,941	21.60	15.00	6	0.2%	1.43	1.4	1.42	1.3	0%
Copper %											
ENV	1	7,726	1.11	0.16	6	0%	0.00	6.3	0.00	3.1	33%
ENV	2	28,209	4.99	1.00	29	0%	0.01	8.3	0.01	4.8	10%
ENV	3	4,944	3.20	0.80	15	0%	0.02	5.9	0.02	3.9	15%
ENV	4	2,073	3.30	0.80	15	1%	0.03	5.8	0.02	3.8	23%
ENV	5	5,497	2.26	0.10	12	0%	0.01	4.8	0.01	1.1	14%
22 Zone	101	2,853	0.83	0.60	2	0%	0.01	3.5	0.01	3.4	0%
21A	201	4,459	0.82	0.50	8	0%	0.02	2.6	0.02	2.5	0%
	202	469	0.93	0.50	1	0%	0.02	3.0	0.02	2.5	5%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	6,687	2.17	0.61	15	0%	0.03	2.5	0.03	2.1	3%
	302	1,589	2.07	1.00	9	1%	0.08	2.0	0.08	1.8	4%
	303	546	4.26	3.00	4	1%	0.17	2.6	0.17	2.5	2%
21B	401	4,771	4.41	3.00	14	0%	0.10	3.4	0.10	3.3	1%
	402	4,626	22.37	7.00	6	0%	0.44	2.3	0.42	2.0	3%
21Be	501	3,913	8.57	4.50	21	1%	0.24	3.1	0.23	2.8	5%
	502	1,947	8.66	6.00	6	0%	0.46	2.0	0.46	2.0	1%
21E	601	868	0.45	0.15	10	1%	0.01	2.7	0.01	2.1	15%
	602	345	1.51	1.00	5	1%	0.10	2.3	0.10	2.2	4%
	603	838	0.79	0.25	9	1%	0.02	3.1	0.02	1.9	12%

DOMAIN	ZONE	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
HW	703	4,038	10.00	3.00	8	0%	0.30	1.9	0.30	1.8	1%
NEX	801	3,600	5.19	3.00	10	0%	0.11	3.0	0.10	2.8	3%
	802	42,522	10.89	5.00	6	0%	0.31	2.1	0.31	2.0	1%
WT	811	709	1.69	0.20	9	1%	0.02	3.6	0.02	1.7	22%
LP	90	1,199	0.40	0.20	4	0%	0.01	1.9	0.01	1.6	7%
	91	133	0.21	0.09	6	5%	0.02	1.7	0.02	1.3	15%
	92	98	0.14	0.14	1	1%	0.01	2.2	0.01	2.2	0%
	93	107	0.55	0.20	1	1%	0.03	2.1	0.03	1.3	11%
	94	266	0.35	0.10	4	2%	0.02	1.6	0.02	1.1	11%
PMP	95	762	3.07	0.50	6	1%	0.05	2.7	0.05	1.6	10%
109	99	2,716	2.09	0.80	10	0%	0.03	3.7	0.03	2.8	7%
Zinc %											
ENV	1	7,759	4.13	1.00	25	0.3%	0.03	4.3	0.03	3.1	6%
ENV	2	28,339	44.40	3.00	164	0.6%	0.11	7.2	0.08	3.8	27%
ENV	3	4,958	27.25	4.00	73	1.5%	0.26	5.3	0.18	3.4	30%
ENV	4	2,081	22.00	6.00	20	1.0%	0.28	4.7	0.23	3.6	19%
ENV	5	5,495	4.65	0.50	20	0.4%	0.03	3.4	0.03	1.5	9%
22 Zone	101	2,870	18.31	4.00	7	0.2%	0.15	3.7	0.14	1.9	5%
21A	201	4,465	5.78	3.00	8	0.2%	0.18	2.1	0.18	1.9	2%
	202	469	7.71	2.00	5	1.1%	0.20	3.0	0.17	2.0	17%
	203	0	-	-	-	-	-	-	-	-	--0
21C	301	6,687	15.28	4.00	13	0.2%	0.18	2.7	0.18	2.1	4%
	302	1,589	15.90	10.00	4	0.3%	0.69	1.9	0.68	1.8	1%
	303	546	24.36	16.00	5	0.9%	1.25	2.5	1.20	2.4	4%
21B	401	4,784	38.95	24.00	12	0.3%	0.78	3.2	0.77	3.1	1%
	402	4,631	31.49	30.00	1	0.0%	3.07	1.8	3.07	1.8	0%
21Be	501	3,939	32.96	30.00	6	0.2%	1.69	2.3	1.69	2.3	0%
	502	1,963	31.37	30.00	4	0.2%	3.34	1.7	3.34	1.7	0%
21E	601	868	5.09	1.00	13	1.5%	0.10	2.8	0.09	1.9	10%
	602	345	9.13	6.50	6	1.7%	0.63	2.3	0.61	2.2	3%
	603	838	4.86	4.00	3	0.4%	0.14	2.5	0.14	2.4	1%
HW	703	5,054	31.30	27.00	9	0.2%	3.00	1.7	3.00	1.7	0%
NEX	801	3,600	32.73	20.00	10	0.3%	1.25	2.1	1.23	2.0	1%
	802	4,526	32.51	27.00	7	0.2%	2.78	1.6	2.77	1.6	0%
WT	811	709	15.31	1.50	16	2.3%	0.20	4.3	0.14	2.1	32%
LP	90	1,301	18.90	10.00	6	0.5%	0.64	2.3	0.62	2.2	2%
	91	133	4.22	1.00	4	3.0%	0.29	1.9	0.24	1.3	17%
	92	98	1.73	1.10	2	2.0%	0.19	1.7	0.17	1.5	6%
	93	107	16.27	9.00	3	2.8%	1.68	1.6	1.59	1.5	5%

DOMAIN	ZONE	# Samples	Maximum	Cap Value	No. Cap	% Cap	Uncapped Composites		Capped Composites		% Metal Lost
							Mean	CV	Mean	CV	
	94	271	11.00	5.00	6	2.2%	0.92	1.6	0.86	1.4	6%
PMP	95	762	16.05	2.00	8	1.0%	0.25	2.8	0.22	1.5	12%
109	99	2,941	26.30	17.00	10	0.3%	2.17	1.3	2.15	1.3	1%

Table 14-34: Capping Statistics for Metallurgical Elements by Zone

Domain	Zone	# Samples	Maximum	Topcut	No. cut	% cut	Uncapped Composites		Capped Composites		% Metal Lost
							mean	CV	mean	CV	
Sulphur ppm											
ENV	1	1,288	124,972	50,000	6	0.5%	6,768	1.3	6,636	1.1	2%
ENV	2	11,884	381,476	120,000	5	0.0%	13,912	1.0	13,872	1.0	0%
ENV	3	278	67,586	40,000	10	3.6%	9,467	1.2	9,078	1.1	4%
ENV	4	1,189	74,600	40,000	83	7.0%	13,919	1.0	13,382	0.9	4%
ENV	5	2,820	77,900	50,000	12	0.4%	9,175	1.1	9,143	1.1	0%
22 Zone	101	2,041	162,178	110,000	5	0.2%	9,438	1.4	9,396	1.4	0%
21A	201	2,371	140,836	90,000	7	0.3%	14,568	0.7	14,499	0.6	0%
	202	208	255,145	200,000	3	1.4%	35,978	1.2	35,498	1.1	1%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	2,144	123,938	50,000	5	0.2%	11,397	0.6	11,312	0.6	1%
	302	745	135,915	90,000	7	0.9%	28,423	0.5	28,252	0.5	1%
	303	259	176,356	75,000	11	4.2%	36,199	0.6	34,686	0.5	4%
21B	401	1,648	56,775	28,000	15	0.9%	11,691	0.5	11,614	0.4	1%
	402	364	122,059	60,000	8	2.2%	31,155	0.4	30,831	0.4	1%
21Be	501	235	204,492	100,000	10	4.3%	20,159	1.5	18,422	1.2	9%
	502	54	176,173	80,000	3	5.6%	38,087	0.7	36,225	0.5	5%
21E	601	732	46,024	30,000	2	0.3%	8,727	0.5	8,704	0.5	0%
	602	120	39,282	32,000	4	3.3%	16,166	0.5	16,020	0.5	1%
	603	512	268,859	75,000	7	1.4%	26,537	0.8	25,321	0.5	5%
HW	703	646	185,581	80,000	23	3.6%	33,797	0.6	32,915	0.6	3%
NEX	801	0	-	-	-	-	-	-	-	-	-
	802	31	58,000	58,000	31	100.0%	31,697	0.3	31,697	0.3	0%
WT	811	89	74,807	40,000	8	9.0%	17,529	0.9	15,709	0.8	10%
LP	90	268	361,452	300,000	3	1.1%	80,462	0.7	80,083	0.7	0%
	91	59	124,223	90,000	10	16.9%	55,077	0.6	51,998	0.5	6%
	92	89	108,616	100,000	2	2.2%	39,375	0.5	39,185	0.5	0%
	93	20	361,409	300,000	3	15.0%	146,079	0.7	142,558	0.7	2%
	94	8	26,140	20,000	1	12.5%	15,652	0.3	14,884	0.3	5%
PMP	95	95	34,867	15,000	10	10.5%	9,919	0.4	9,575	0.3	3%
109	99	0	-	-	-	-	-	-	-	-	-

Domain	Zone	# Samples	Maximum	Topcut	No. cut	% cut	Uncapped Composites		Capped Composites		% Metal Lost
							mean	CV	mean	CV	
					Iron %						
ENV	1	1,288	13.2	7.0	3	0.2%	1	0.7	1.22	0.6	1%
ENV	2	11,884	29.5	13.0	5	0.0%	4	0.7	3.92	0.7	0%
ENV	3	278	10.4	8.0	4	1.4%	3.18	0.6	3.16	0.6	1%
ENV	4	1,189	9.8	9.8	0	0.0%	5.75	0.4	5.75	0.4	0%
ENV	5	2,820	9.9	9.5	3	0.1%	4.61	0.6	4.61	0.6	0%
22 Zone	101	2,041	10.8	8.5	9	0.4%	1.30	0.8	1.29	8.5	0%
21A	201	2,371	12.2	3.5	14	0.6%	1.39	0.4	1.38	0.3	1%
	202	208	6.6	5.0	2	1.0%	2.14	0.4	2.12	0.4	1%
	203	0	-	-	-	-	-	-	-	-	-
21C	301	2,144	8.0	6.0	6	0.3%	1.41	0.5	1.42	0.4	0%
	302	745	11.2	10.0	1	0.1%	3.04	0.5	3.04	0.5	0%
	303	259	12.5	10.0	6	2.3%	5.82	0.4	5.793	0.4	0%
21B	401	1,648	4.8	4.0	2	0.1%	1.30	0.3	1.30	0.3	0%
	402	364	10.7	7.8	5	1.4%	3.27	0.4	3.26	0.4	0%
21Be	501	235	17.9	10.0		0.0%	2.16	1.2	2.06	1.0	5%
	502	54	7.9	7.0		0.0%	4.09	0.4	4.05	0.4	1%
21E	601	732	7.2	7.2	0	0.0%	1.15	0.5	1.15	0.5	0%
	602	120	8.3	8.0	3	2.5%	3.27	0.5	3.27	0.5	0%
	603	512	9.2	8.0	4	0.8%	3.85	0.4	3.85	0.4	0%
HW	703	646	15.2	10.0	6	0.9%	4.38	0.5	4.37	0.5	0%
NEX	801	0	-	-	-	-	-	-	-	-	-
	802	31	7.0	-	0	0.0%	3.93	0.3	3.93	0.3	0%
WT	811	89	10.3	8.0	11	12.4%	3.20	0.8	3.14	0.8	2%
LP	90	268	27.0	20.0	3	1.1%	8.18	0.5	8.11	0.5	1%
	91	59	13.3	12.0	3	5.1%	5.96	0.5	5.95	0.5	1%
	92	89	11.0	9.0	5	5.6%	5.46	0.3	5.41	0.3	1%
	93	20	27.2	20.0	4	20.0%	12.21	0.6	11.43	0.6	6%
	94	8	1.9	1.9	0	0.0%	1.35	0.2	1.35	0.2	0%
PMP	95	95	3.4	2.1	3	3.2%	1.35	0.2	1.13	0.2	1%
109	99	0	-	-	-	-	-	-	-	-	-

14.17.4 Block Model Details

The epithermal, base metal and metallurgical elements used the same block model geometry and extents as the gold and silver block model with 9 x 9 x 4 m parent blocks, and 3 x 3 x 2 m subblocks, where subblocks occur around the zone boundaries.

14.17.5 Estimation Parameters

Due to selective sampling, insufficient data were available to produce reliable variograms necessary for a kriged estimate. Therefore, the block model grades were estimated using ID^2. A NN declustered model was also estimated to determine declustered composite statistics for validation purposes.

The final parameters selected for the epithermal, base metals and metallurgical element estimates are presented in Table 14-35. A discretization grid of 4 x 4 x 3 m was used during all estimation runs. Ranges were determined for each of the elements from the variogram (Table 14-36). The estimate was generated in two consecutively longer passes. Pass 1 used a minimum of five and a maximum of 16 samples at the variogram range and Pass 2 used a minimum of three samples and a maximum of 16 at two times the variogram range to ensure that at least two drill holes were used for the estimate. An octant search was used to aid in declustering using two samples per octant, and hard boundaries were honoured in all zones. A third pass at four times the variogram range was used to aid in validation for the LP domain.

Table 14-35: Interpolation Parameters for the Epithermal, Base Metal, Metallurgical Elements by Zone

Zone	Search Pass	Orientation	Ranges	No. of Composites		Max Composites per Drill Hole	Max Samples per Octant
				Minimum	Maximum		
ALL	1	Gold Variogram Orientation	1 x variogram range	5	16	2	2
	2		2 x variogram range	3	16	2	2
LP	3		4 x variogram range	3	16	2	2

Table 14-36: Ranges for the Epithermal, Base Metal, Metallurgical Elements by Zone

	Base Metals									Epithermal									Metallurgical					
	Lead			Copper			Zinc			Arsenic			Mercury			Antimony			Sulphur			Iron		
Vario Code	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)	Major (Y)	Semi (X)	Minor (Z)
101	60	40	40	50	35	25	60	35	35	60	50	50	60	50	30	60	40	40	45	40	35	50	45	35
201	60	40	40	25	20	20	45	25	25	50	30	30	50	20	15	50	25	20	70	45	20	45	40	15
202	50	30	20	35	25	20	50	30	20	50	30	20	60	35	20	40	25	20	50	35	10	60	40	15
203	50	30	20	25	25	20	50	30	20	40	40	20	40	35	20	40	25	20	30	20	18	30	20	15
204	30	25	15	30	25	15	30	25	15	30	25	15	30	25	15	30	25	15	30	25	15	30	25	15
3011	30	20	15	30	20	15	20	20	15	40	25	15	30	25	15	30	25	15	60	25	15	40	30	20
3012	30	20	20	30	20	10	30	20	10	30	20	10	30	20	10	30	20	10	n/a					
302	75	45	20	55	50	25	45	40	25	50	40	25	30	30	20	30	30	20	40	40	20	50	40	15
303	40	20	20	35	18	15	40	20	20	35	18	15	35	18	15	35	18	15	35	20	15	30	20	15
401	65	40	20	65	40	15	65	40	20	45	45	30	45	40	20	35	35	25	65	35	15	45	30	20
4011	35	30	10	20	20	15	35	30	10	30	25	20	15	15	10	25	25	10	n/a					
402	70	60	10	70	60	10	70	60	10	60	60	10	70	60	10	70	60	10	70	45	20	60	50	20
501	50	50	20	50	50	15	50	50	20	50	30	15	60	45	10	30	25	10	n/a					
502	35	15	10	25	15	10	40	15	10	40	15	15	50	25	10	30	30	15	n/a					
601	40	30	20	30	30	15	50	20	15	55	50	25	35	35	20	30	30	20	50	45	10	30	30	20
603	55	35	15	45	40	20	45	45	15	50	50	20	45	45	20	45	45	15	40	40	20	40	30	15
7034	35	15	10	20	15	10	40	15	10	30	15	10	25	25	10	20	15	10	25	25	10	25	25	10
7035	50	40	15	45	35	20	35	35	20	45	25	10	50	20	15	45	25	15	25	25	15	25	25	10
7038	35	35	10	40	35	10	35	30	10	25	15	5	40	40	20	30	30	15	n/a					
801	62	55	15	45	20	15	50	50	15	45	30	15	50	30	15	45	45	20	n/a					
811	35	35	15	30	30	15	50	25	20	40	40	20	45	45	20	50	25	20	30	30	20	30	30	15
802	50	50	15	55	40	15	55	55	15	40	35	10	40	35	15	45	40	10	n/a					
90	35	35	35	35	35	35	35	35	35	60	30	30	45	30	30	55	30	30	35	30	30	45	35	20
93	55	30	20	75	20	20	75	20	20	50	20	20	40	20	20	40	20	20	40	20	20	40	20	20
94	40	40	40	45	40	20	40	35	20	30	20	20	25	20	20	25	20	20	20	20	15	20	20	15
95	40	20	20	30	25	10	35	20	10	30	25	10	25	25	10	30	20	10	30	20	10	25	20	15
99	25	20	18	25	20	10	30	20	15	25	20	20	30	20	20	25	25	15	n/a					

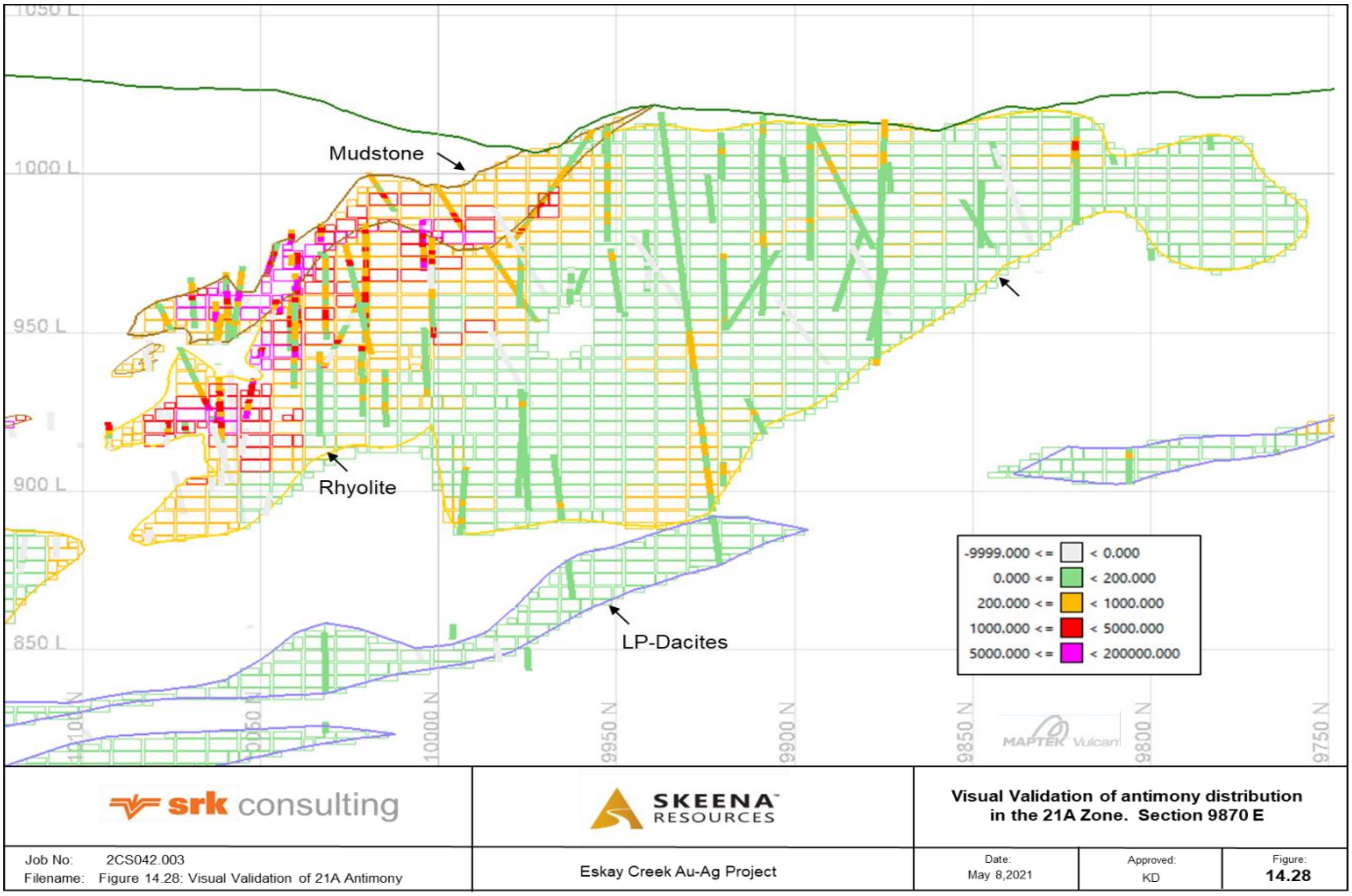
14.17.6 Block Model Validation

The block model estimates were validated for the elements using several methods to ensure an unbiased estimate; these include a visual review of the block model, grade distribution evaluations using swath plots, and global validation.

14.17.7 Open Pit Model – Visual Validation

Section and plan view visual inspections of the block model were conducted for each element to evaluate final estimated grades with the neighbouring informing composites. In addition, domain coding accuracies were checked during this stage. Figure 14-28 shows estimated antimony block grades in relation to 2 m antimony composite intervals in the 21A Domain. Overall, the data show good agreement, and no major discrepancies between block grades and composites were observed.

Figure 14-28: Example of Visual Validation of Antimony Distribution in the 21A Domain (Looking East)



14.17.8 Comparison of Interpolation Models

The ID^2 model was compared against the NN declustered model to check for the occurrence of global bias. Although variability exists between the different zones for both the ID^2 and NN estimates, there is an average difference of less than 5% for all elements (Table 14-37 to Table 14-39) confirming that global bias is not a concern for the estimates. For the LP Domain (Zones 91 to 93) higher percent differences were noted. Seeing that these zones occur mostly below the conceptual pit shell and contain low numbers of composites in relation to the total number of blocks, higher variability was not considered to be a concern.

Table 14-37: Comparison ID^2 vs Declustered NN Estimates within Each Zone for the Epithermal Elements

Zone	Arsenic			Mercury			Antimony		
	NN Declustered	ID^2	ID2 vs NN Declustered	NN Declustered	ID^2	ID^2 vs NN Declustered	NN Declustered	ID^2	ID^2 vs NN Declustered
101	939	971	3	6.3	6.4	1	287	295	3
201	358	392	9	42.0	52.1	19	424	477	11
202	12,059	12,586	4	577.6	643.4	10	7,473	8,747	15
301	213	217	2	7.5	7.8	4	214	223	4
302	649	649	0	22.2	23.0	4	785	827	5
303	467	465	-1	21.9	23.3	6	1,483	1,562	5
401	301	306	2	50.1	51.0	2	1,322	1,173	-13
402	1,228	1,247	2	500.0	517.5	3	8,533	8,980	5
501	1,484	1,487	0	46.1	46.7	1	989	1,025	3
502	1,035	1,056	2	211.3	218.2	3	3,855	4,184	8
601	320	341	6	14.2	14.5	2	367	370	1
602	287	316	9	13.1	13.0	-1	1,845	1,727	-7
603	536	527	-2	13.6	14.5	6	5,022	5,120	2
703	638	588	-9	23.3	24.8	6	1,437	1,413	-2
801	385	380	-1	15.0	16.0	6	610	667	9
802	527	537	2	24.1	25.3	5	1,125	1,207	7
811	653	545	-20	6.6	6.8	3	291	273	-6
90	491	518	5	6.8	8.0	15	109	119	9
91	829	480	-73	7.4	2.6	-191	103	62	-65
92	123	131	6	0.8	0.8	1	19	13	-44
93	817	704	-16	18.6	15.7	-19	263	193	-36
94	535	685	22	11.1	13.1	15	135	167	19
95	438	467	6	16.2	17.8	9	1,011	1,095	8
99	572	589	3	30.3	30.0	-1	242	248	2
Average Difference			-2	Average Difference		-4	Average Difference		-2

Table 14-38: Comparison of ID2 vs Declustered NN Estimates within each Zone for the Base Metal Elements

Zone	Lead			Copper			Zinc		
	NN Declustered	ID^2	ID^2 vs NN Declustered	NN Declustered	ID^2	ID^2 vs NN Declustered	NN Declustered	ID^2	ID^2 vs NN Declustered
101	0.080	0.086	7	0.013	0.015	13	0.135	0.141	4%
201	0.087	0.101	14	0.015	0.017	12	0.146	0.171	15%
202	0.052	0.052	0	0.017	0.017	0	0.127	0.125	-2%
301	0.087	0.090	3	0.022	0.023	4	0.150	0.159	6%
302	0.393	0.397	1	0.077	0.078	1	0.678	0.690	2%
303	0.468	0.544	14	0.135	0.15	10	0.915	1.031	11%
401	0.257	0.264	3	0.051	0.053	4	0.458	0.453	-1%
402	1.294	1.325	2	0.29	0.297	2	2.193	2.251	3%
501	0.735	0.752	2	0.124	0.133	7	1.151	1.206	5%
502	1.355	1.462	7	0.27	0.295	8	2.295	2.402	4%
601	0.050	0.050	0	0.012	0.012	0	0.089	0.090	1%
602	0.286	0.276	-4	0.091	0.089	-2	0.559	0.548	-2%
603	0.039	0.038	-3	0.014	0.014	0	0.122	0.124	2%
703	1.433	1.503	5	0.21	0.225	7	2.113	2.250	6%
801	0.530	0.554	4	0.055	0.063	13	0.858	0.896	4%
802	1.365	1.487	8	0.203	0.223	9	2.066	2.130	3%
8011	0.061	0.067	9	0.016	0.018	11	0.111	0.112	1%
90	0.193	0.239	19	0.012	0.012	0	0.291	0.361	19%
91	0.181	0.154	-18	0.017	0.015	-13	0.230	0.212	-8%
92	0.109	0.078	-40	0.011	0.008	-38	0.133	0.109	-22%
93	0.623	0.434	-44	0.023	0.018	-28	0.587	0.668	12%
94	0.504	0.547	8	0.016	0.018	11	0.748	0.828	10%
95	0.091	0.092	1	0.028	0.03	7	0.168	0.167	-1%
99	1.405	1.376	-2	0.026	0.025	-4	2.139	2.133	0%
Average Difference			0	Average Difference		1	Average Difference		3

Table 14-39: Comparison of ID2 vs Declustered NN Estimates within each Zone for the Metallurgical Elements

ZONE	Sulphur ppm			Iron %		
	NN Declustered	ID^2	ID^2 vs NN Declustered	NN Declustered	ID^2	ID^2 vs NN declustered
101	10,678	10,055	-6	1.36	1.392	2
201	12,749	12,943	1	1.37	1.339	-2
202	30,131	24,951	-21	2.020	2.163	7
203	-	-	-			
301	10,337	10,666	3	1.311	1.319	1
302	26,902	28,501	6	2.903	2.895	0
303	33,534	34,978	4	5.34	5.057	-6
401	12,502	11,962	-5	1.333	1.351	1
402	34,214	32,225	-6	3.384	3.368	0
501	-	-	-			
502	-	-	-			
601	8,266	8,439	2	1.059	1.079	2
602	-		-			
603	24,244	24,840	2	4.097	3.736	-10
703	32,875	30,200	-9	4.375	4.365	0
801	-	-	-			
802	-	-	-			
811	16,226	16,666	3	2.862	2.77	-3
90	68,041	55,216	-23	7.4	7.476	1
91	56,422	58,505	4	5.777	6.020	4
92	36,076	36,844	2	5.580	5.631	1
93	31,634	48,533	35	11.496	10.718	-7
94	15,917	13,574	-17	1.470	1.276	-15
95	8,918	9,703	8	1.097	1.091	-1
99	-	-	-	-	-	-
Average Difference			1	Average Difference		-1

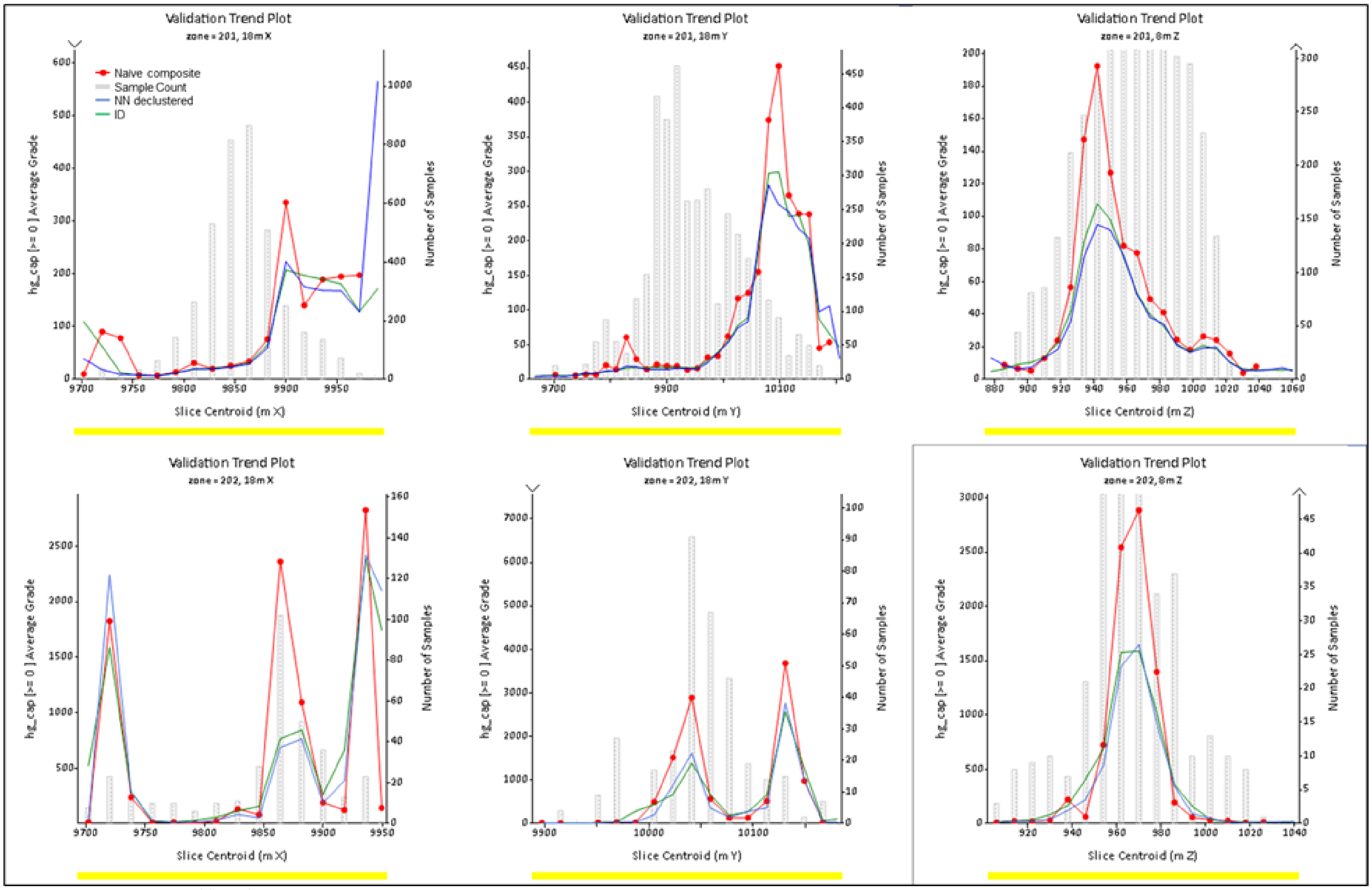
14.17.9 Swath Plots

Swath plots were generated in three orthogonal directions to graphically display grade distribution in each of the zones in north–south, east–west, and horizontal directions throughout the deposit.

All zones and all elements (antimony, arsenic, mercury, lead, copper, zinc, iron, and sulphur) were visually assessed using swath plots in three directions. Grade variations from the ID^2 model were compared to the NN grade distribution, along with declustered composite data determined from a NN model. The swath plots showed acceptable correspondence between grade distributions, although the ID^2 model inherently smoothed the results.

An example of mercury swath plots in the 21A Domain is shown in Figure 14-29 which depicts the naive composite grade (red line), block model ID^2 grade (green line), and declustered NN grade (blue line).

Figure 14-29: Swath Plots of Mercury in Zone 201 -- 21A Rhyolite (top) and Zone 202 -- 21A Mudstone (bottom). Left (Northing), middle (Easting) and right (Elevation)



Note: Figure prepared by Skeena, 2021

14.17.10 Base Metal, Epithermal, and Metallurgical Element Concentrations

The average estimated epithermal, base metal, and metallurgical concentrations remaining in each domain within the pit shell at the resource cut off grade of AuEq > 0.7 g/t is shown in Table 14-40.

Table 14-40: Epithermal, Base Metal, and Metallurgical Concentrations Remaining in Each of the Domains within the Open Pit Shell at 0.7 g/t AuEq

Domain	Mercury ppm	Arsenic ppm	Antimony ppm	Lead %	Copper %	Zinc %	Fe %	S %
ENV	11	200	266	0.220	0.031	0.337	1.736	0.672
22	6	763	241	0.106	0.015	0.167	1.333	0.955
21A	102	1,352	1,184	0.095	0.018	0.165	1.373	1.397
21C	13	320	446	0.193	0.045	0.340	1.716	1.424
21B	96	489	1,865	0.378	0.073	0.636	1.665	1.609
21Be	75	1,345	1,572	0.808	0.136	1.299	-	-
21E	15	429	2,561	0.066	0.020	0.149	1.922	1.373
HW	23	514	1,302	1.353	0.203	2.022	1.782	1.332
NEX	20	445	765	0.937	0.118	1.380	-	-
WT	8	240	311	0.100	0.028	0.177	0.839	0.501
LP	24	1,229	301	0.706	0.021	1.165	-	-
PMP	17	469	1,061	0.093	0.030	0.167	1.057	0.946
109	28	594	245	1.359	0.025	2.098	-	-

14.18 Factors that May Affect the Estimates

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to the density values applied to the mineralized zones; changes to geological shape and continuity assumptions; potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures; changes to the input values used to generate the AuEQ cut-off grade; changes to metallurgical recovery assumptions; changes in assumptions of marketability of final product; changes to the conceptual input assumptions for assumed open pit operations, changes to the input assumptions for assumed underground operations; variations in geotechnical, hydrogeological and mining assumptions; changes to environmental, permitting and social license assumptions.

15 MINERAL RESERVE ESTIMATES

15.1 Mining Method and Mining Costs

The Eskay Creek Project is planned to be an open pit operation using conventional mining equipment. No underground mining is considered.

All work is based on the mine plans generated by AGP.

Costs are based on first principles build-up of operating and capital costs for the life of the project with current vendor quotations for consumables and maintenance. Mining capital costs were based on vendor submissions.

The current resource model dated 7 April 2021 is used for all mine design work. Only Measured and Indicated Mineral Resources were used in the estimation of Mineral Reserves for the Eskay Creek Project. Inferred Mineral Resources were considered as waste.

15.1.1 Geotechnical Considerations

Based on the available geotechnical and hydrogeological data, AGP determined slope design criteria for the current PFS (Table 15-1, and see discussion in Section 16.3). These may be updated and refined once additional levels of confidence in geotechnical conditions have been established. In AGP's experience, the noted criteria are practical estimates of achievable slope configurations. If the project is feasible at the noted inter-ramp slope angles, any improvements or further optimization that can be achieved because of additional geotechnical study for given scenarios will be value additive. If on the other hand project economics are marginal when these criteria are incorporated, this simply highlights the necessity of determining/confirming the project's geotechnical conditions as soon as practical.

Table 15-1: 2021 PFS Open Pit Slope Design Criteria

Slope Domain	SLOPN Code	Overall Slope (°)	Description
'weak' rock slopes	1	34	includes contact and hanging wall mudstones, fault zones, generally RMR < 40
'competent' rock slopes	2	46	Includes andesites, rhyolites, generally RMR >40

The various criteria were loaded into the geologic model for use by AGP by lithological unit. The criteria are used for pit optimization as well as pit design work.

15.1.2 Economic Pit Shell Development

The final pit designs are based on pit shells using the Lerchs–Grossmann (LG) procedure in MinePlan software. The parameters for the pit shells are shown in Table 15-2.

Table 15-2: Pit Optimization Parameters

Description	Units	Value	Gold Value	Silver Value
Exchange rates				
CAD	US$ =	1.26		
Resource model				
Block classification used		M+I		
Block model height	m	4		
Mining bench height	m	8		
Metal prices				
Price	$/oz		1475	20
Royalty	%		2%	2%
Smelting, refining, transportation terms				
Payable for 20 g/t Au concentrate	%		72.88	71.25
Payable for 25 g/t Au concentrate	%		75.5	73
Payable for 35 g/t Au concentrate	%		80.75	76.5
Payable for 45 g/t Au concentrate	%		86	80
Minimum deduction	unit, g/dmt		0	0
Participation (on profits)	%		100	100
Bulk concentrate treatment charge	$/dmt	0		
Refining	$/oz		0	0
Concentrate moisture	%	12		
Transit losses	%	0.5		
Concentrate transportation cost	C$/wmt	141.60		
Metallurgical Information				
Recovery [1]	A constant	-0.3112 * (Au con g/t) + 96.753		
	B constant	-0.0075 * (Au con g/t) -1.0716		
	%		A*(1-EXP(B*Au))	0.93 * Au rec
Power cost				
Cost of power	C$/Kwhr	$0.05		
Fuel cost				
Diesel fuel cost to site	C$/ l	$1.03		
Mining cost [2]			NAG	PAG
Waste base rate - 880 elevation	C$/t		3.02	3.71
Incremental rate - above	C$/t/4m bench		($0.007)	($0.007)
Incremental rate - below	C$/t/4m bench		$0.016	$0.016
Mill feed base rate - 880 elevation	C$/t		2.43	2.43
Incremental rate - above	C$/t/4m bench		$0.008	$0.008
Incremental rate - below	C$/t/4m bench		$0.013	$0.013
Processing [3]				
Processing cost	C$/t mill feed	$22.80		
Water treatment	C$/t mill feed	$1.70		
Total processing cost	C$/t mill feed	$24.50		

Description	Units	Value	Gold Value	Silver Value
General and administrative cost				
G&A cost	C$/t mill feed	$6.06		
Total process and G&A				
Process + G&A	C$/t mill feed	$30.56		

Note: [1] maximum NSR value used from 20–45 g/t Au concentrate grades, but if no Fe or S values available 25 g/t Au concentrate used with Au rec = 92%, Ag rec = 97%. [2] mining costs based on using 144 t haul trucks. [3] process costs based on 2.5 Mt/a dry throughput

Ultimate pits were generated using a revenue factor of 0.9 or metal price of $1,328 /oz. These were used as the basis for the design.

15.1.3 Cut-off

For the statement of reserves for the Eskay Creek Project, the marginal NSR value per tonne that was previously stored in the block model was used as the cut-off. As the NSR is inclusive of all revenues and royalties, applying a C$30.56/t cut-off represents the marginal cut-off to flag initial feed and waste blocks. This cut-off value represents the preliminary process and site G&A costs.

15.1.4 Dilution

The open pit resource model was provided as an undiluted percentage type model, such that the grades from the wireframes were reported into separate percentage parcels of ore and waste in each block. The provided feed percentage values exclude underground workings and all material within 0.2 m of their original solids. These underground solids were viewed on several plan views with ORE% values and the workings appear to have been properly adjusted in a consistent manner. As the mine workings were mostly backfilled, they were included in the waste percentage.

To account for mining dilution, AGP modelled contact dilution into the in-situ resource blocks. To determine the amount of dilution, and the grade of the dilution, the size of the block in the model was examined. The block size within the model was 9 x 9 m in plan view, and 4 m high. Mining would be completed on 8 m lifts for waste and 4 m lifts for mill feed, if required, and the equipment selected is capable of mining in that manner.

The percentage of dilution is calculated for each contact side using an assumed 1.25 m contact dilution distance. This dilution skin thickness was selected by considering the spatial nature of the mineralization, proposed grade control methods, size of equipment, GPS-assisted digging accuracy, and blast heave.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 21.1% more tonnes and 16.6% lower gold grade than the in-situ feed summary. The grade dilution percentage was lower than the feed tonnage percentage since the mineralized waste blocks included some grade. The average grade of the dilution material was 0.16 g/t Au and 3.65 g/t Ag. AGP considers these dilution percentages to be reasonable considering the expected seasonal working conditions as well as mining through underground workings.

15.1.5 Pit Design

Pit designs were developed for the north and south pit areas. The north pit will consist of five phases, while the south pit will only contain a single small phase. The pit optimization shells used to determine the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. Each pit phase was designed to accommodate the recommended mining fleet. Mining occurs on 8 m benches with catch benches spaced either 8 m or 16 m vertically depending on lithology type. The haul roads are 30.2 m in width with a road grade of 10%.

The mine schedule plans to deliver 26.4 Mt of mill feed grading 3.37 g/t Au and 94.4 g/t Ag over a mine life of 10 years. Waste tonnage totalling 212 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio will be 8.0:1.

The mine schedule assumes a maximum of 2.9 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. A maximum descent rate of eight benches per year per phase was applied.

The proposed mine life includes three years of pre-stripping and 10 years of mining. Mill feed will be stockpiled during the pre-production years. A technical sample will be mined in Year -3 so that process performance of the mill can be evaluated with a larger sample than drill hole samples.

15.1.6 Mine Reserves Statement

The Mineral Reserves for the Eskay Creek Project are based on the conversion of the Measured and Indicated Mineral, Resources within the current mine plan. Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted directly to Probable Mineral Reserves. The estimates were prepared under the supervision of Willie Hamilton, P.Eng. of AGP, a QP as defined under NI 43-101.

For the statement of Mineral Reserves for the Eskay Creek Project, the marginal NSR value per tonne that was stored to the block model previously was used as the value for cut-off application. As the NSR is inclusive of all revenues and royalties, applying a C$30.56/t cut-off represents the marginal cut-off to flag initial feed and waste blocks. This cut-off value represents the preliminary process and site G&A costs.

This estimate has an effective date of 30 June 2021. The total reserves for the Eskay Creek Project are shown in metric units in Table 15-3.

Table 15-3: Proven and Probable Reserves – Summary for Eskay Creek Project

Reserve Class	Tonnes (Mt)	Grade			Contained Ounces		
		Au (g/t)	Ag (g/t)	AuEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)
Proven	13.5	4.25	124	5.81	1.85	53.7	2.53
Probable	12.9	2.46	64	3.26	1.02	26.5	1.35
Total	26.4	3.37	94	4.57	2.87	80.2	3.88

*Note: This mineral reserve estimate has an effective date of June 30, 2021, and is based on the mineral resource estimate dated April 7, 2021 for Skeena Resources by SRK Consulting. The Mineral Reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final design pit based on a US$1,475/oz gold price and US$20.00/oz silver price. An NSR cut-off of C$30.56/t was used to define the marginal cut-off material. The life-of-mine mining cost averaged C$3.14/t mined, preliminary processing costs are C$24.50/t ore and G&A was C$6.06/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. Gold concentrate grades varied from 20 to 45 g/t gold. Silver recovery was assumed to be 93% of the gold recovery.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves.

16 MINING METHODS

16.1 Overview

Open pit mining was selected for the PFS, based on the size of the resource, grade tenor, grade distribution and proximity to topography. AGP's opinion is that with current metal pricing levels, knowledge of the mineralization and previous mining activities, open pit mining offers the most reasonable approach for development.

The Project is located predominately to the south of Tom Mackay Creek with a small portion extending to the north. Infrastructure is located on the south side of Tom Mackay Creek, with the pit now also extending to the north beyond Tom Mackay Creek. Underground mining has previously been conducted in the northern portion of the project at depth, so additional details have been incorporated for mining near old workings. The potential for underground development beneath the open pit was examined in preliminary evaluations during the PFS but has not been included as part of this PFS. There is still potential for the inclusion of underground mining in future mining studies.

The mine plan is based on Measured and Indicated Mineral Resources. Inferred Mineral Resources are too speculative geologically to have economic considerations applied to them, so are treated as waste in this PFS.

16.2 Geological Model Importation

The 2021 resource estimates were created using Leapfrog software for mineralization domains and Vulcan software for block modelling. SRK provided Skeena with support and review of the updated resource model, together with a resource estimate completed in compliance with NI 43-101 and a technical report prepared using the requirements of Form 43-101F1. Skeena provided AGP with regularized resource models in Hexagon MinePlan® block model format for open pit mine planning. The original Vulcan resource models were sub-blocked models. The final resource models provided to AGP for mine design were single mineralization percentage models.

Framework details of the different open pit block models are provided in Table 16-1. Resource model item descriptions are shown in Table 16-2 while the final open pit mine planning model items are displayed in Table 16-3. The mining model created by AGP in MinePlan includes additional items for mine planning purposes. MinePlan was used for the mining portion of the PFS, using their Lerchs Grossmann (LG) shell generation, pit and WRSF design and mine scheduling tools.

Only Measured and Indicated Mineral Resources were used for the PFS. The density provided in the resource model is based on the lithological model, with the mean value of measurements selected as the density for each lithology considered, except for the barite-rich Mudstone in the 21C Domain.

Table 16-1: Open Pit Model Framework

Framework Description	Skeena Resource Open Pit Model (Value)	Final PFS Open Pit Model (Value)
MinePlan file 10 (control file)	PFS10.dat	pfs310.dat
MinePlan file 15 (model file)	ore15.AGP	pfs315.m03
X origin (m)	9,300	9,300
Y origin (m)	8,508	8,508
Z origin (m) (max)	1470	1470
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	132	132
Number of blocks in Y direction	406	406
Number of blocks in Z direction	380	380
X block size (m)	9	9
Y block size (m)	9	9
Z block size (m)	4	4

Table 16-2: Resource Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
AUOK	0	400	0.001	g/t	gold grade (PAYABLE)
AGOK	0	20000	0.1	g/t	silver grade (PAYABLE)
CUPAY	0	5	0.0001	%	copper grade (PAYABLE)
SBPAY	0	300000	1	ppm	antimony grade (PAYABLE)
PBPEN	0	20	0.0001	%	lead grade (PENALTY)
ZNPEN	0	30	0.0001	%	zinc grade (PENALTY)
CUPEN	0	5	0.0001	%	copper grade (PENALTY)
ASPEN	0	300,000	1	ppm	arsenic grade (PENALTY)
HGPEN	0	14,000	0.1	ppm	mercury grade (PENALTY)
SBPEN	0	300,000	1	ppm	antimony grade (PENALTY)
SPEN	0	95000	0.1	ppm	sulfur grade (PENALTY)
FEPEN	0	20	0.0001	%	iron grade (PENALTY)
SG	0	4	0.0001	g/cm3	density
ZONE	0	900	1	-	mineralized zones according to lithology groupings
BUFER	0	670	1	-	zone around the underground workings used to limit grade smearing
ESTZN	0	980000	1	-	Estimation zone
DOMAN	0	99	1	-	historical mineralized zone areas (MAIN GROUPING TO FOCUS ON), 1-99
ROCK	0	32	1	-	rock type
RESAT	0	3	1	-	classification based on whole model (1=Measure; 2=Indicated; 3=Inferred)
TRIZN	0	71000	1	-	zones used to estimate base metals and impurities (without a BUFFER zone)
AUEQ	0	500	0.0001	g/t	gold equivalent (PAYABLE)
SPCT	0	10	0.0001	%	sulfur grade
ORE%	0	100	0.01	%	percentage of DOMAIN 1-99 including the portion of mined out material
MINE%	0	100	0.01	%	percentage of stopes and underground workings

Field Name	Min	Max	Precision	Units	Comments
TOPO%	0	100	0.01	%	Percentage of block below topography
MINE	0	100	0.0001	-	fraction of stopes and underground workings
ORE	0	1	0.0001	-	Fill vol or fraction in re-blocked domains
AUFES	0	500000	0.001	-	ratio of Au to FE+S (for metallurgical studies)

Table 16-3: Open Pit Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
AU	0	400	0.001	g/t	gold grade (PAYABLE)
AG	0	20000	0.1	g/t	silver grade (PAYABLE)
PB	0	20	0.0001	%	lead grade (PENALTY)
ZN	0	30	0.0001	%	zinc grade (PENALTY)
CU	0	5	0.0001	%	copper grade (PENALTY)
AS	0	300000	1	ppm	arsenic grade (PENALTY)
HG	0	14000	0.1	ppm	mercury grade (PENALTY)
SB	0	300000	1	ppm	antimony grade (PENALTY)
S	0	95000	0.1	ppm	sulfur grade (PENALTY)
FE	0	20	0.0001	%	iron grade (PENALTY)
SG	0	4	0.0001	g/cm3	density
ZONE	0	900	1	-	mineralized zones according to lithology groupings
DOMAN	0	99	1	-	historical mineralized zone areas (MAIN GROUPING TO FOCUS ON), 1-99
ROCK	0	32	1	-	rocktype
RESAT	0	3	1	-	resource classification (1=Measure; 2=Indicated; 3=Inferred, 9=default)
AUEQ	0	500	0.0001	g/t	gold equivalent (PAYABLE), reference only, not used in NSR calculations
SPCT	0	10	0.0001	%	sulfur grade
ORE%	0	100	0.01	%	percentage of DOMAIN 1-99 including the portion of mined out material
MINE%	0	100	0.01	%	percentage of stopes and underground workings
TOPO%	0	100	0.01		Percentage of block below topography
AUFES	0	500000	0.001	-	ratio of Au to FE+S (for metallurgical studies)
ARD	1	2	1	-	Acid rock drainage (1=PAG, 2=NAG), where PAG = default
SLOPE	0	10	1	-	Slope domain: 1= weak slope, 2=competent slope
SLOPN	0	10	1	-	Slope domain: 1= weak, 2=competent, 3=weak North ext, 4=competent North ext
NSR1	0	100000	0.01	C$/t	Net Smelter Return (excludes process costs)
TMP1	0	100000	0.01		Temporary item for debugging pything scripts
CON1	0	99	1	g/t	Gold concentrate grade for NSR1 (20-45 g/t)
MINE	0	1	1	-	Value =1 for entire model
DEF	0	1	1	-	Block flag (0=default, 1= MI block with no Fe or S value)
VLT1	-1000	20000	0.01	C$/t	Value per tonne for run 1 pit shells
VLB1	-10000	1000000	1	C$	Value per block for run 1 pit shells
DAU	0	400	0.001	g/t	diluted gold grade (PAYABLE)

DAG	0	20000	0.1	g/t	diluted silver grade (PAYABLE)
DPB	0	20	0.0001	%	diluted lead grade (PENALTY)
DZN	0	30	0.0001	%	diluted zinc grade (PENALTY)
DCU	0	5	0.0001	%	diluted copper grade (PENALTY)
DAS	0	300000	1	ppm	diluted arsenic grade (PENALTY)
DHG	0	14000	0.1	ppm	diluted mercury grade (PENALTY)
DSB	0	300000	1	ppm	diluted antimony grade (PENALTY)
DSPCT	0	10	0.0001	%	diluted sulfur grade (PENALTY)
DFE	0	20	0.0001	%	diluted iron grade (PENALTY)
BLOKT	0	9999	0.01	t	block tonnage
OWFL	0	1	1	-	Ore/waste flag, where 0= waste, 1=ore
DTON	0	9999	0.01	t	diluted block tonnage
DDEN	0	4	0.0001	t/m3	diluted block density
DORE%	0	100	0.01	%	diluted ore percentage
DWAS%	0	100	0.01	%	diluted waste percentage
ROUTE	0	9	1	-	routing number if different cut-off grades to be applied
BERM	0	99	0.01	m	berm width for pit design

16.3 Open Pit Geotechnics

16.3.1 2020 QP Inspections and Geotechnical Investigations

AGP has completed four ‘Qualified Person’ (QP) mining-geotechnical inspections of the project site, two most recently in July and October 2020, for the current PFS study.

During AGP’s recent inspections, the following tasks were completed:

- meetings with Skeena geology and exploration staff to discuss and review open pit geotechnical drill plans and status;
- collection and compilation of site geological and geotechnical data;
- review of local and regional geology, including relevant reports, plans and sections;
- completion of domain-scale geotechnical logging of select intervals of drill core available at the time of inspection; and
- vehicular and on-foot traversing of drill access roads, rock slopes, historic portals and plant site, and geotechnical mapping and rock mass characterization focused on verifying and supplementing existing information, including lithology, rock mass strength, and discontinuity characteristics.

As described in Section 10 an open pit (geomechanical) drilling program was completed between August and November 2020. The program consisted of 14 inclined boreholes at depths between approximately 25 and 180 m distributed within the perimeter of the proposed pit walls. Coordinates and depths of these boreholes were initially determined based on the current configuration of the Open Pits (North and South Pits) and then reviewed and revised by Skeena. The azimuth and dip of each borehole were selected to determine rock properties and fracture orientations around the pit perimeter while faults and weak points were deliberately targeted to collect geotechnical information. A Reflex ACT™ orientation tool was deployed every 15 m to determine proper alignment of each borehole.

Primary objectives of the geomechanical drill program in the open pit area included:

- capturing data regarding rock quality, rock strength, faults, groundwater, and permeability conditions for developing the pit wall geometry;
- determining various geological units in the open pit area;
- determining rock mass and rock mechanics properties;
- determining the geological-structural, geotechnical, geomechanical and hydrogeological units/parameters of the various rock types in the pit walls;
- determining groundwater levels and hydrogeological values (e.g., hydraulic conductivity of bedrock); and
- collecting representative samples for the laboratory strength testing program.

A summary of related field activities conducted by Skeena, Ausenco, and AGP during the 2020 investigation program includes the following:

- ground truthing and stake test pit/borehole locations;
- surface mapping;
- supervising rock core logging, and sampling of boreholes for the laboratory program;
- geotechnical logging of cores from geotechnical-hydrogeological boreholes, and determining rock mass rating (RMR) and basic rock properties;
- conducting Pneumatic Packer Tests when applicable in rock at approximately 30 m intervals;
- installing groundwater wells and monitoring instruments including installation of Casagrande and vibrating wire piezometers (VWPs);
- conducting point load tests (compressive strength);
- collecting digital photographic logs for both borehole and test pit programs; and
- preparing geotechnical logs combining Rock Mass Rating (RMR) properties, geological units, hydraulic conductivity values from packer tests, well installation diagrams, and rock properties from on-site core tests.

Figure 16-1: 2020 Open Pit Drill Program

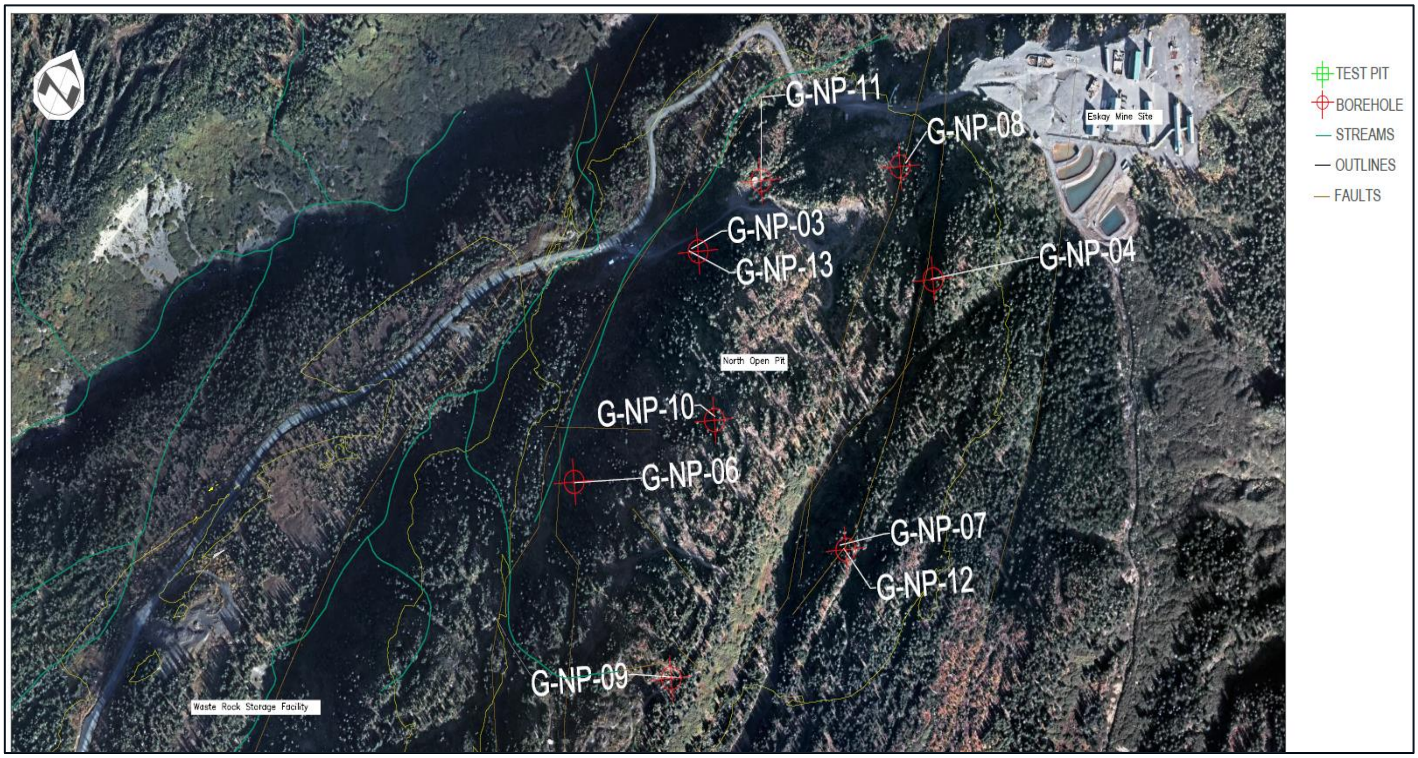
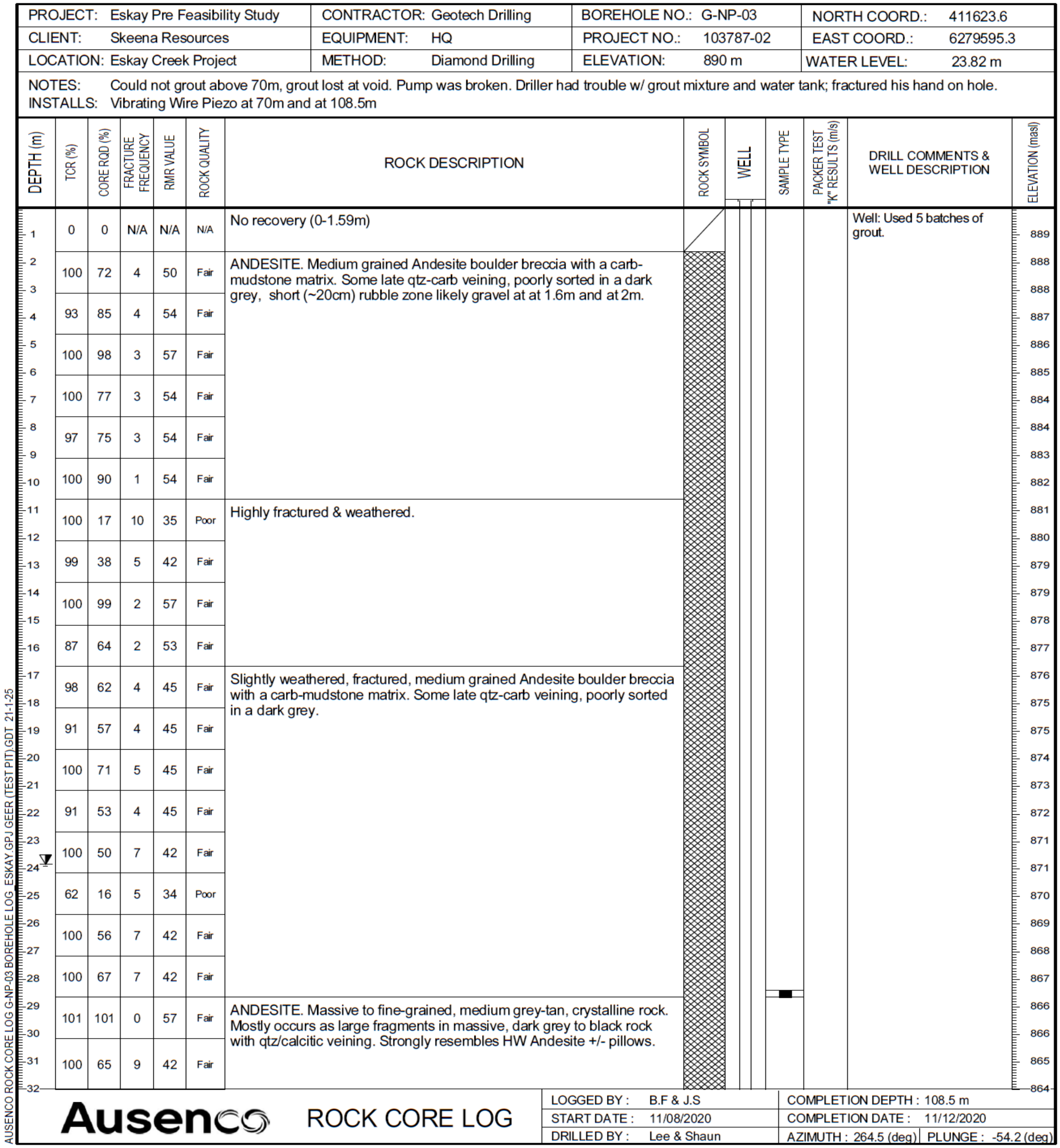


Table 16-4: Open Pit 2020 Geotechnical Drilling Investigation Summary

Borehole ID	Easting (m)	Northing (m)	Azimuth (Degrees)	Dip (Degrees)	Depth Reached (m drilled)
G-SP-01 A	6278005	410722	343	-68	79.8
G-SP-01 B	6278008	410723	343	-90	25.0
G-SP-02	6277947	410717	105.5	-70.4	80.9
G-NP-03	6279595	411624	264.5	-54.2	108.5
G-NP-04	6279525	412000	133	-60	150.3
G-NP-06	6279292	411399	276	-59	125.5
G-NP-07	6279162	411829	32.6	-50.2	170.1
G-NP-08	6279686	411960	26.5	-60	136.5
G-NP-09	6279005	411532	162.5	-70	159.8
G-NP-10	6279359	411632	168	-73	178.9
G-NP-11	6279688	411734	305.5	-74.7	150.1
G-NP-11 R	6279689	411736	298.5	-73.5	67.9
G-NP-12	6279162	411829	112.5	-68.6	170.0
G-NP-13	6279595	411625	176	-79.3	156.7

Figure 16-2: Example 2020 Ausenco Geomechanical Log



PROJECT:	Eskay Pre Feasibility Study	CONTRACTOR:	Geotech Drilling	BOREHOLE NO.:	G-NP-03
CLIENT:	Skeena Resources	EQUIPMENT:	HQ	PROJECT NO.:	103787-02
LOCATION:	Eskay Creek Project	METHOD:	Diamond Drilling	ELEVATION:	890 m

NORTH COORD.: 411623.6
EAST COORD.: 6279595.3
WATER LEVEL: 23.82 m

NOTES: Could not grout above 70m, grout lost at void. Pump was broken. Driller had trouble w/ grout mixture and water tank; fractured his hand on hole.
INSTALLS: Vibrating Wire Piezo at 70m and at 108.5m

TCR (%)	CORE RQD (%)	FRACTURE FREQUENCY	RMR VALUE	ROCK QUALITY	ROCK DESCRIPTION
0	0	N/A	N/A	N/A	No recovery (0-1.59m)
100	72	4	50	Fair	ANDESITE. Medium grained Andesite boulder breccia with a carb-mudstone matrix. Some late qtz-carb veining, poorly sorted in a dark grey, short (~20cm) rubble zone likely gravel at at 1.6m and at 2m.
93	85	4	54	Fair	
100	98	3	57	Fair	
100	77	3	54	Fair	
97	75	3	54	Fair	
100	90	1	54	Fair	
100	17	10	35	Poor	Highly fractured & weathered.
99	38	5	42	Fair	
100	99	2	57	Fair	
87	64	2	53	Fair	
98	62	4	45	Fair	Slightly weathered, fractured, medium grained Andesite boulder breccia with a carb-mudstone matrix. Some late qtz-carb veining, poorly sorted in a dark grey.
91	57	4	45	Fair	
100	71	5	45	Fair	
91	53	4	45	Fair	
100	50	7	42	Fair	
62	16	5	34	Poor	
100	56	7	42	Fair	
100	67	7	42	Fair	
101	101	0	57	Fair	ANDESITE. Massive to fine-grained, medium grey-tan, crystalline rock. Mostly occurs as large fragments in massive, dark grey to black rock with qtz/calcitic veining. Strongly resembles HW Andesite +/- pillows.
100	65	9	42	Fair	

DRILL COMMENTS & WELL DESCRIPTION: Well: Used 5 batches of grout.

DEPTH (m)	ELEVATION (masl)
1	889
2	888
3	888
4	887
5	886
6	885
7	884
8	884
9	883
10	882
11	881
12	880
13	879
14	879
15	878
16	877
17	876
18	875
19	875
20	874
21	873
22	872
23	871
24	871
25	870
26	869
27	868
28	867
29	866
30	866
31	865
32	864

Ausenco ROCK CORE LOG

LOGGED BY : B.F & J.S
START DATE : 11/08/2020
DRILLED BY : Lee & Shaun
COMPLETION DEPTH : 108.5 m
COMPLETION DATE : 11/12/2020
AZIMUTH : 264.5 (deg)
PLUNGE : -54.2 (deg)

AUSENCO ROCK CORE LOG G-NP-03 BOREHOLE LOG ESKAY.GPJ GEER (TEST PIT).GDT 21-1-25

Figure 16-3: Example 2020 Ausenco Photo Log G-NP-11 Boxes 14-17

Photo 14: BOX # 14, 55.5m – 58.7m

Photo 15: BOX #15, 58.7m – 69.4m

Photo 16: BOX #16, 69.4 – 72.6m

Photo 17: BOX #17, 72.6m – 76.1m

16.3.2 Open Pit Mining Geotechnical Assessment

Following completion of the 2020 drilling program, AGP conducted a compilation, review, and assessment of available geotechnical data and information to determine initial estimates of suitable pit slope angles for PFS-level mine planning tasks.

AGP's assessment is based primarily on geotechnical data from drilling, logging, mapping, sampling, and laboratory testing completed for the current study, in consideration of economic pit shells, geologic models, and relevant background reports.

Overall, the data indicates generally 'fair' rock mass conditions within the mining zone with 'poorer' quality rock masses and local bench-scale slope instability likely to be encountered in zones within and proximal to mudstone units, and adjacent to fault zones.

Rock Quality Designation (RQD) data collected by Skeena and their contractors since 2018, field verified by AGP, typically ranges from zero, in upper hole intervals and fault zones, to 20% to 50%+ in most drill runs located in the relatively more competent andesite and rhyolite units. Joint spacing typically varies from 0.2 to 0.6 m but is significantly less in many cases. Intact rock strength varies from R1 to R5 (per the widely used ISRM system), with andesites and rhyolites typically reporting strengths in the R3 (25 to 50 MPa) range. Typical joint characteristics include slightly rough to smooth to slicken-sided surfaces, often with soft clayey infill greater than 5 mm thick.

Examples of geotechnical data collected and compiled for the current study are presented in the figures below.

Figure 16-4: 2018-2020 Drill Program ISRM R0-R6 Strength Ratings – plan view

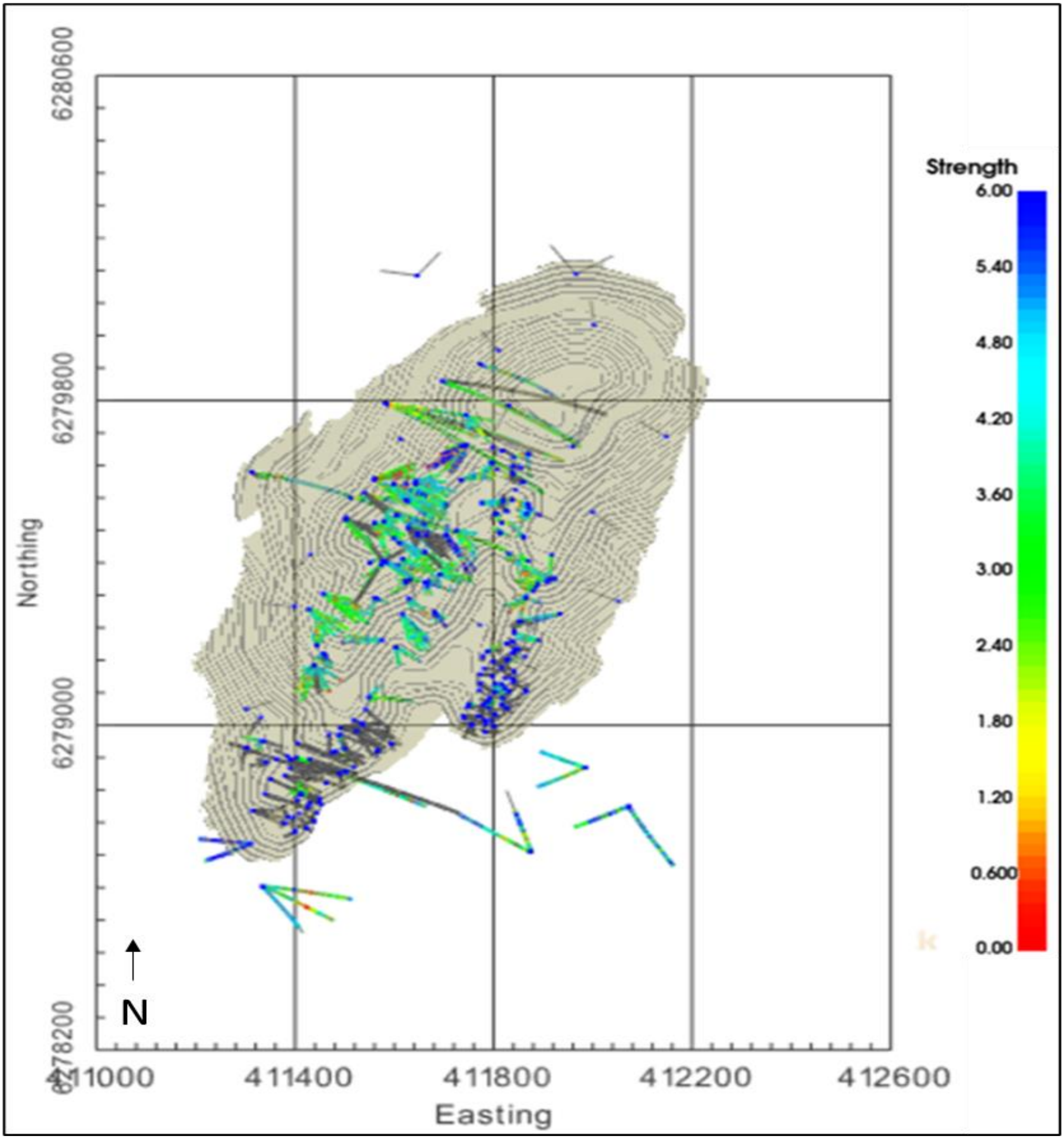


Figure 16-5: 2018-2020 Drill Program ISRM R0-R6 Strength Ratings – oblique view

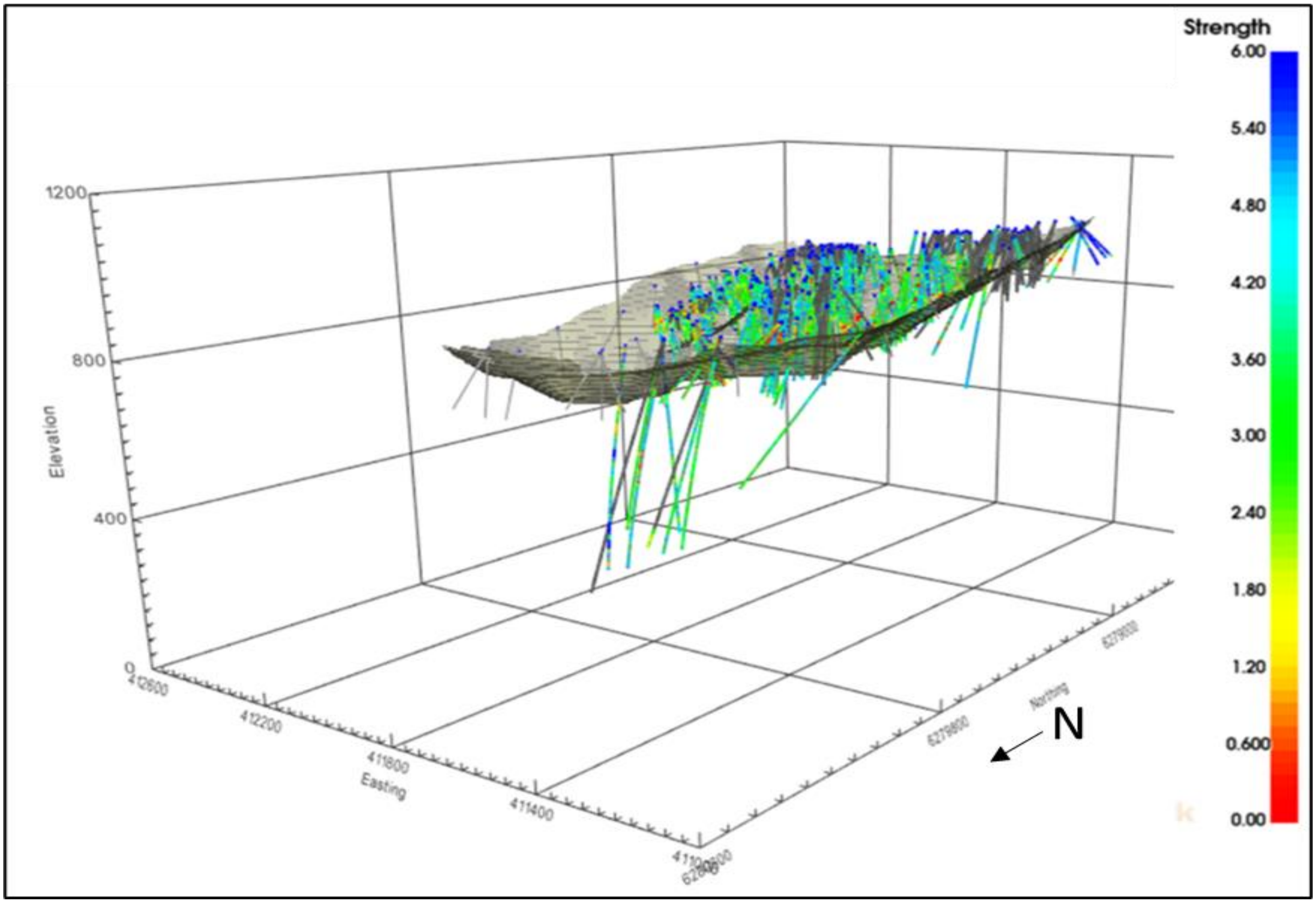


Figure 16-6: 2018-2020 Drill Program RQD Percent Data – plan view

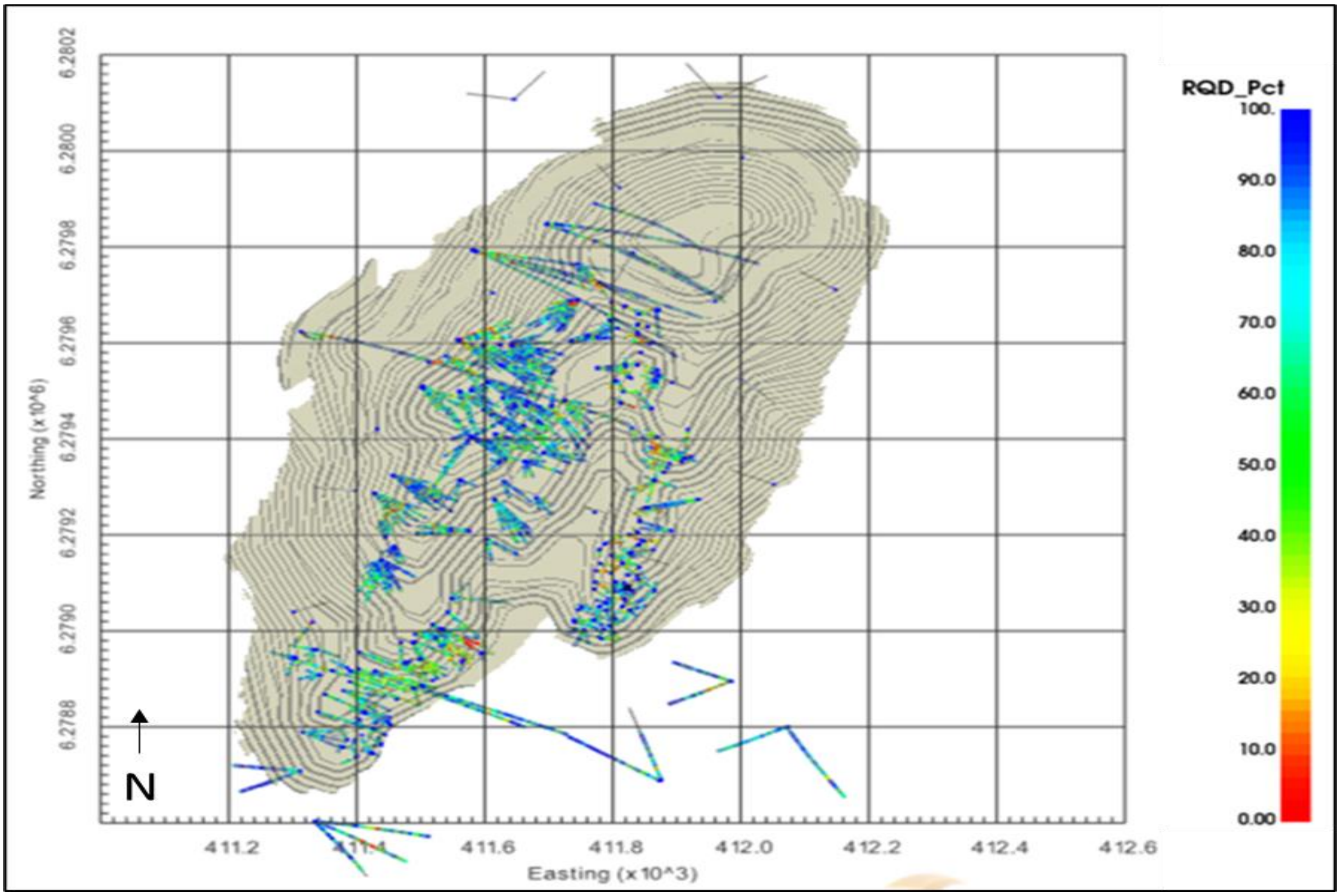


Figure 16-7: 2018-2020 Drill Program RQD Percent Data – oblique view

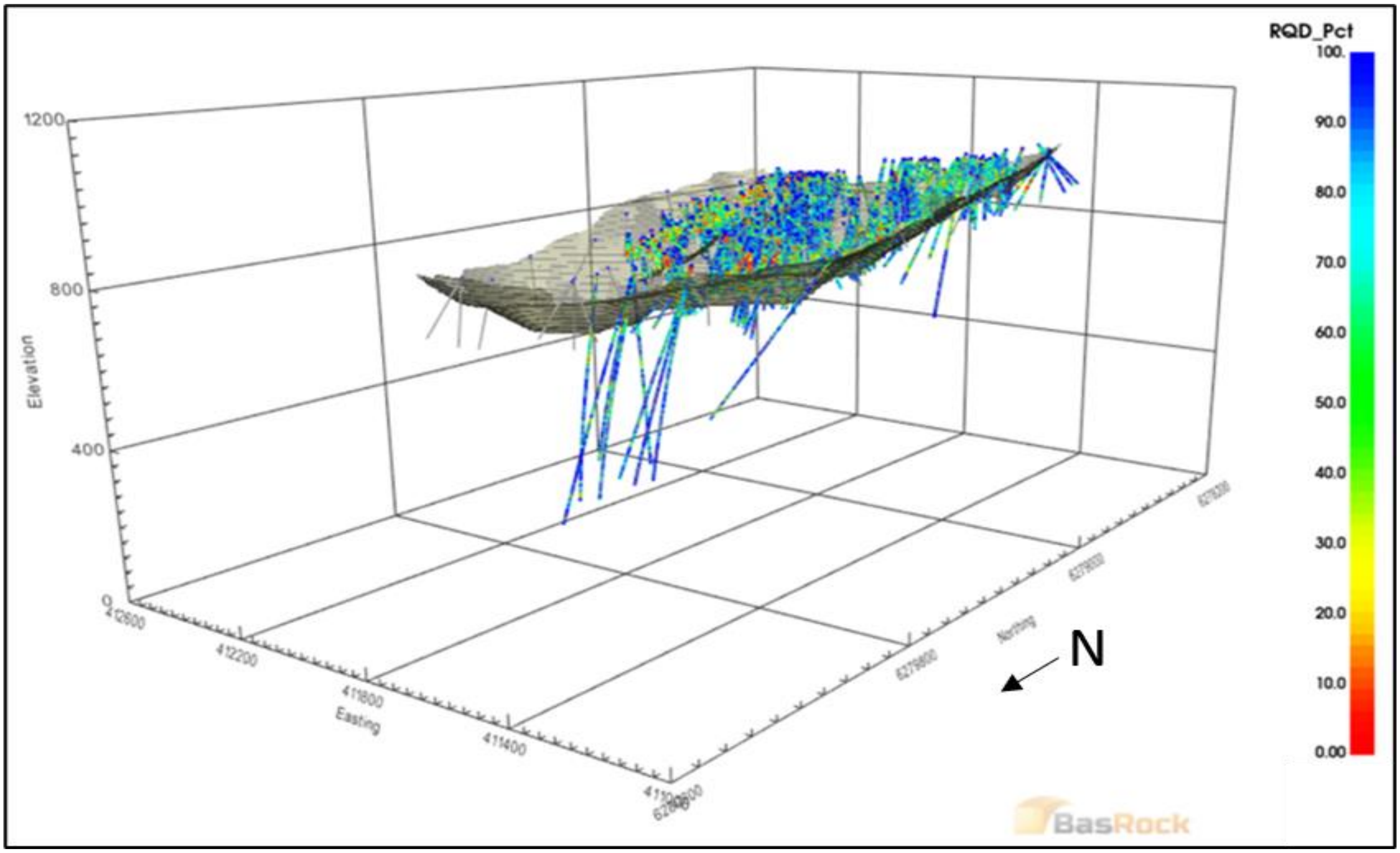
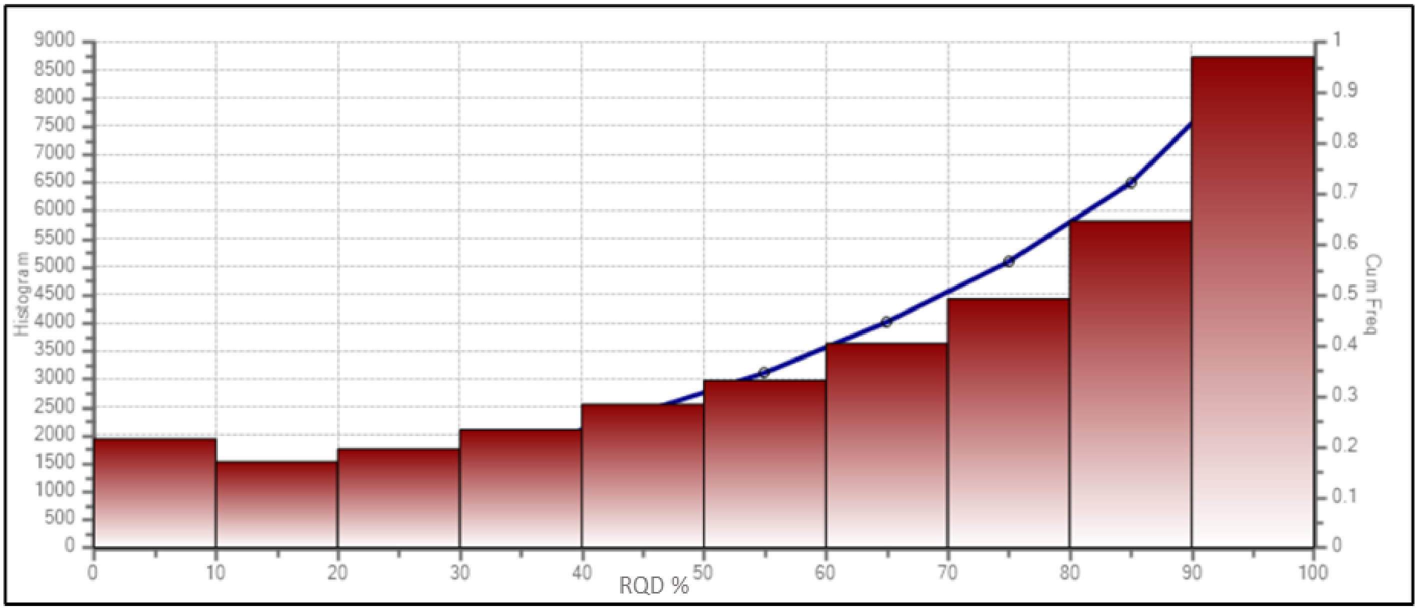


Figure 16-8: **2018-2020 Drill Program RQD Data Histogram**

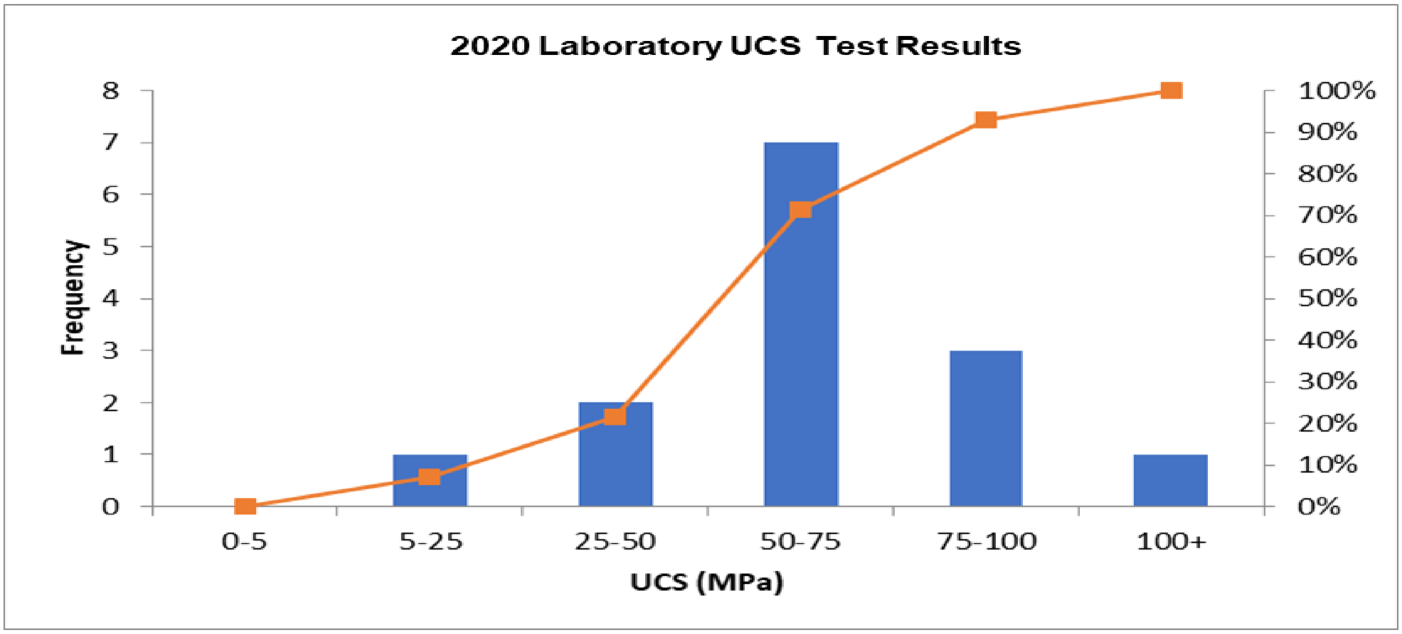


Point Load Tests (PLTs) were conducted on site throughout the drilling program with a Matest Point Load Tester. Further laboratory strength test data was determined for several 'representative' samples collected during the 2020 drilling campaign. Samples were sent to Tetra Tech laboratories in Nanaimo for the testing program.

Table 16-5: **Laboratory Strength Test Data of 2020 Drill Samples**

Borehole ID	From (m)	To (m)	UCS (MPa)	2020 Laboratory UCS Tests Statistics	
G-SP-01A	13	13.5	56.82	Mean	64.8
G-SP-01A	44.7	44.9	66.97	Standard Error	6.2
G-SP-02	44.53	44.7	71.24	Median	69.1
G-NP-04	28.07	28.17	85.8	Standard Deviation	23.2
G-NP-04	115.58	115.8	72.82	Range	84.3
G-NP-06	13.61	14.31	88.21	Minimum	18.8
G-NP-06	111.8	111.95	53.31	Maximum	103.1
G-NP-07	122.8	123.01	72.22	Count	14.0
G-NP-08	136.2	136.5	30.56	Largest (1)	103.1
G-NP-09	77.7	77.9	83.5	Smallest (1)	18.8
G-NP-11R	12.2	12.35	18.75		
G-NP-12	12.15	12.46	62.69		
G-NP-12	96.78	97.02	41.41		
G-NP-03	28.39	28.65	103.08		

Figure 16-9: Laboratory Strength Test Data – 2020 Drill Samples



Estimates for ranges of rock and discontinuity strengths are based on a review of the test data and relevant experience in similar rock masses. Intact rock strengths for andesites and rhyolites are estimated in the 25 to 50 MPa range, and 1 to 25 MPa for hangingwall and contact mudstones. Joint and discontinuity strengths are estimated between 20 to 35 degrees friction (lower in mudstones), with faults likely between 15 to 25 degrees, both with zero to nominal cohesion.

Table 16-6: Summary of Estimates of Rock Mass Conditions from 2020 Drill Core

Parameter	Andesite	Rhyolite	Mudstone	Dacite	Tuff
Avg TCR%	96	97	92	60	103
Avg RQD%	68	69	48	23	74
Avg FF/m	6	6	8	10	2
Average of JCR-89	16	16	16	10	16
Avg ISRM Strength Estimate (MPa)	58	78	47	41	56
Avg RMR-89	55	55	47	36	58

Estimates of RMR89 values from the 2020 geotechnical data typically range from lows of 40 to an average of 55 for the hanging wall andesite and rhyolite geotechnical units. RMR89 values for the mudstones are significantly lower, ranging from lows of 20 to an average of 47.

RMR89 ranges have been used to estimate rock mass strength and deformation parameters. Related Mohr-Coulomb and Hoek-Brown strength envelopes have been estimated over stresses that are a function of the proposed slope heights. A conservative "Disturbance Factor" (D) of 0.75 - 1.0 has been assumed in deriving the various rock mass strength parameters, indicative of significant disturbance to the rock mass due to production blasting and local stress redistributions resulting from mining activities.

North Pit slopes are expected to consist primarily of andesites within the upper pit walls with rhyolite being more prevalent at lower pit elevations. Contact and hanging wall mudstone units intersect and are expected to impact pit slopes on the west and north walls of the pit. The exposure of these various lithologies is displayed on the northern ultimate pit surface in Figure 16-10.

Figure 16-10: Hanging Wall and Contact Mudstone Lithologies Displayed on North Ultimate Pit

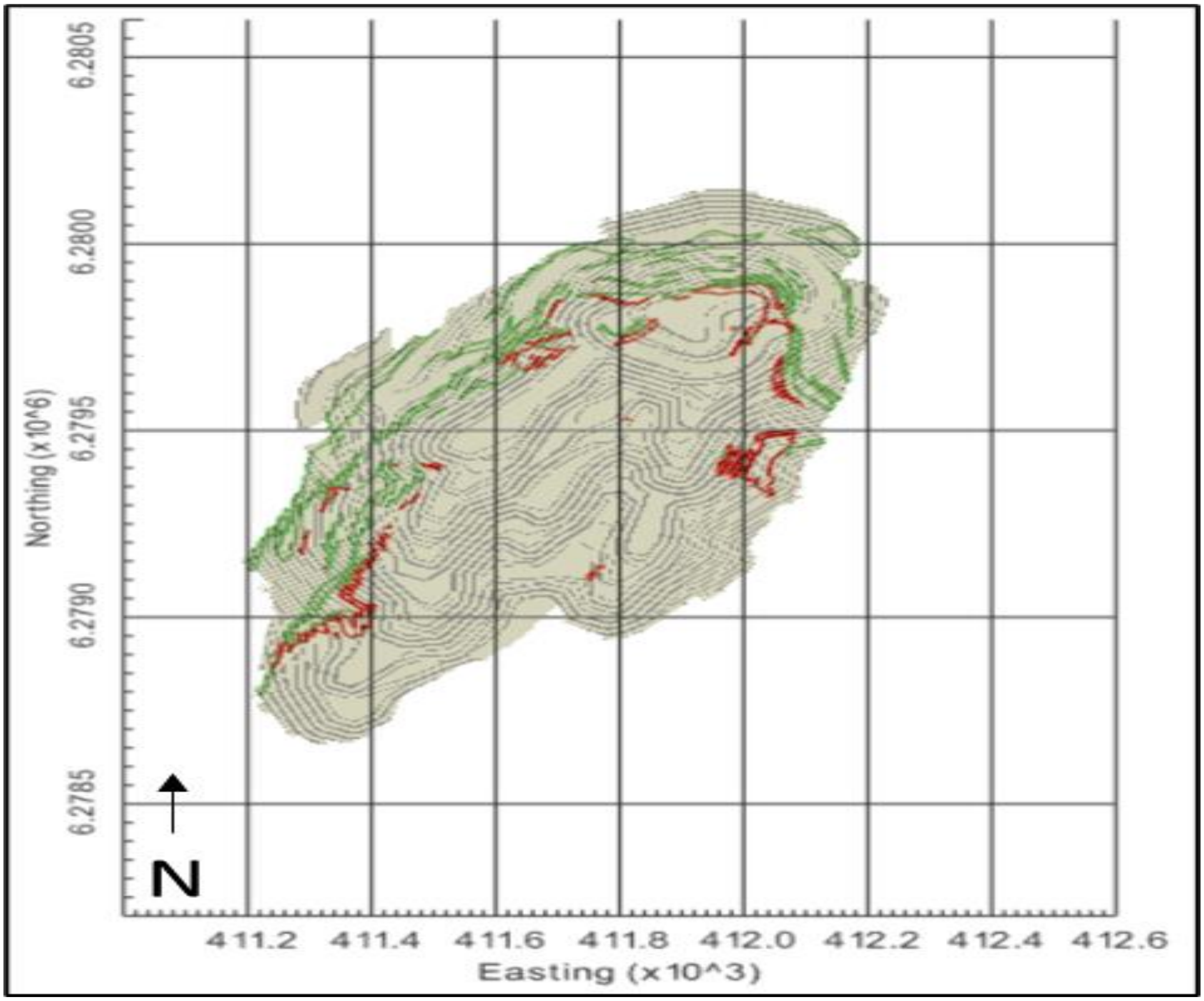
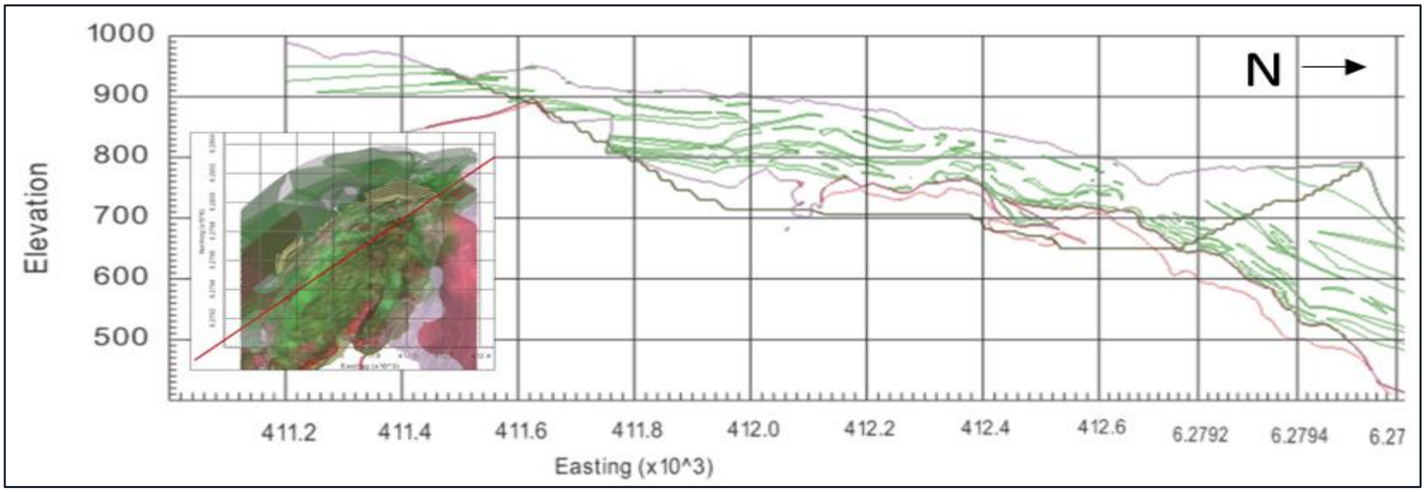
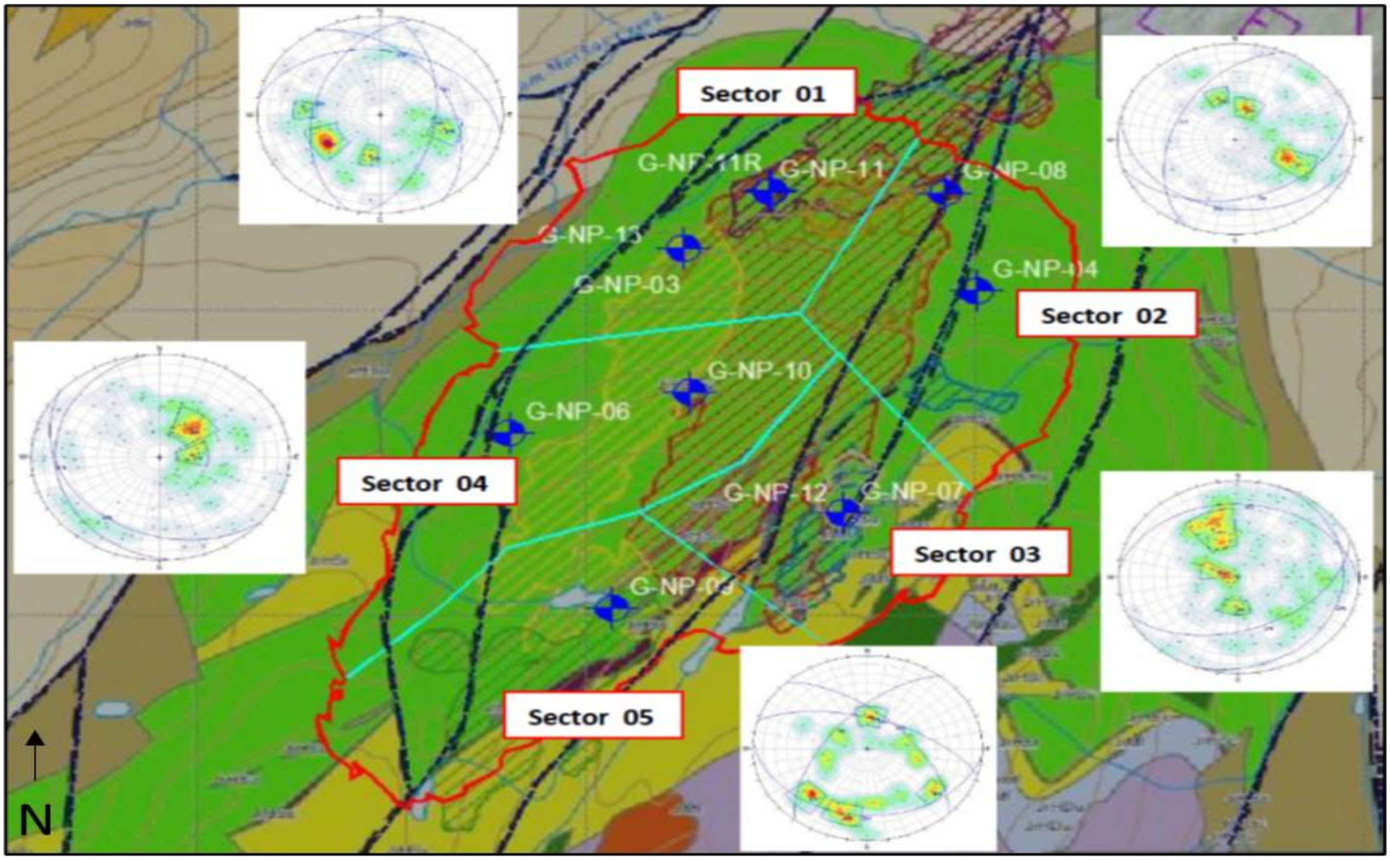


Figure 16-11: Hanging Wall and Contact Mudstone Intercepts Pit Walls



Open pit structural data has been analysed using Dips software with the following conventions: projection using equal area in the lower hemisphere; Dip - Dip direction reporting convention; and equatorial stereonet overlay. The analysis is based on 586 structural data points from 13 oriented boreholes located in the pit area. According to the structural sets identified in each type of lithology and depth, the area has been divided into 6 sectors. The distribution of each sector is shown below.

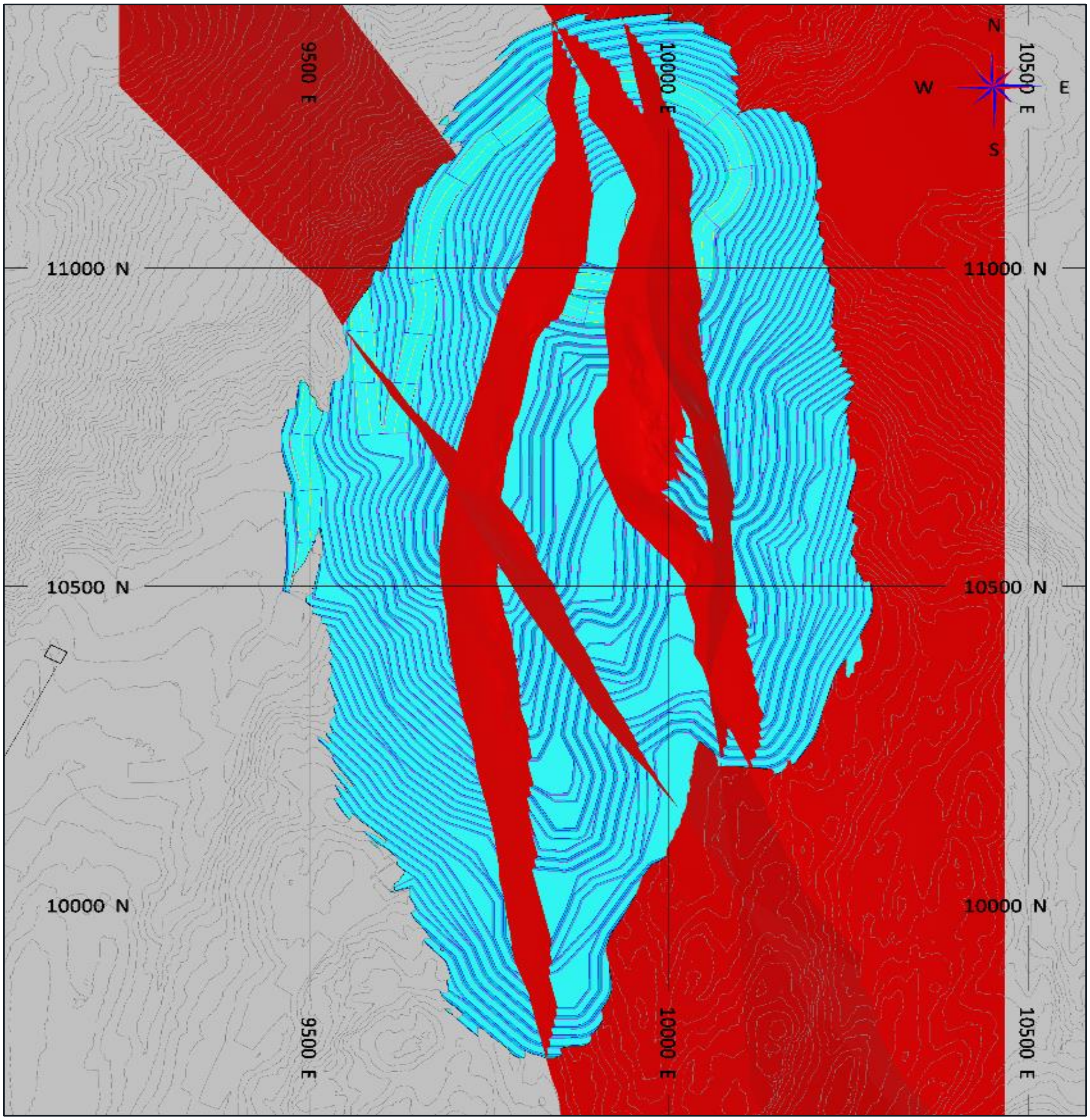
Figure 16-12: **General Structural Analysis by Sector**



Structural Sectors in the Main Pit area – Red line shows the trace of the Projected Open Pit / Stereonets Shows the main joint families of each sector.

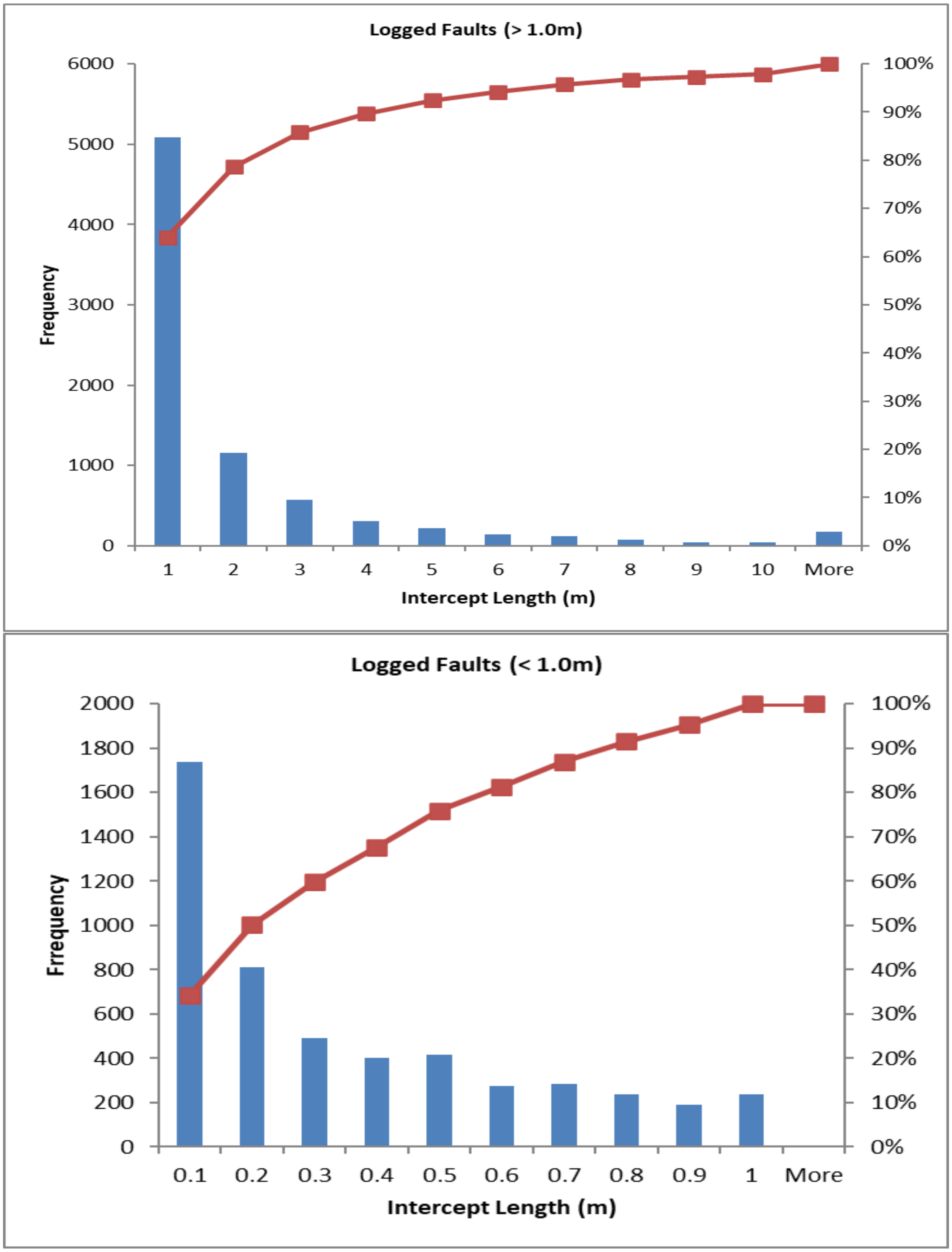
As described in Section 1010 and illustrated below, a number of named faults are known to traverse the project area.

Figure 16-13: North Ultimate Pit with Interpreted Major Fault Structures



Features interpreted as shears, faults, and fault zones/systems have been mapped in outcrops and intersected by drill holes within the proposed pit extents. AGP reviewed the data associated with these features to better understand the frequency and ranges in material conditions. A subset of 2018 to 2020 'fault intercept length' data extracted from total logged core length of 114,634 m is displayed below.

Figure 16-14: Fracture Frequency of 2018 – 2020 Data Subset



Based on current information, several general observations are made regarding the project's mining geotechnics:

- Rock quality varies from good to extremely poor, and is generally related to lithology, and the degree of, and proximity to, local and regional faulting
- Rock quality can change significantly over a very short distance
- There is extensive faulting and contact zones.
- In many parts of the deposit the rock is extremely weak and more soil-like than rock.

Based on the available geotechnical and hydrogeological data, AGP has determined slope design criteria for current PFS-level studies. These may be updated and refined once additional levels of confidence in geotechnical conditions have been established. In AGP's experience the noted criteria are practical estimates of achievable slope configurations; this is by design. If the project is feasible at the noted inter-ramp slope angles, any improvements or further optimization that can be achieved as a result of additional geotechnical study for given scenarios will be value additive. If on the other hand project economics are marginal when these criteria are incorporated, this simply highlights the necessity of determining / confirming the project's geotechnical conditions as soon as practical.

Table 16-7: 2021 PFS Open Pit Slope Design Criteria

Slope Domain	SLOPN Code	Overall Slope (°)	Description
'weak' rock slopes	1	34	includes contact and hanging wall mudstones, fault zones, generally RMR < 40
'competent' rock slopes	2	46	Includes andesites, rhyolites, generally RMR >40

As described below, to allow steeper slope angles in areas with better quality rock and to minimize stripping to the greatest extent possible, AGP divided the pit into slope design sectors based on slope height and dominant geology and geotechnical characteristics. As indicated, the inter-ramp slope recommendations ranged between 34 and 46.

16.3.2.1 Seismicity

Ausenco conducted a seismic hazard assessment (probabilistic and deterministic) for the project site (Ausenco, 2020). Results obtained from the deterministic assessment indicated peak ground acceleration (PGA) of 0.014 g at the project site, with earthquakes with origin in Haida Gwaii active fault, located at a focal distance of 440 km. Probabilistic results corresponded to values reported by the National Building Code of Canada, extrapolated to 5,000- and 10,000-year return period.

16.3.2.2 Geotechnical Model Limitations

The preceding section summarizes information and knowledge gathered to date, primarily by others, along with information collected by AGP during four QP site visits.

This information provides the basis for PFS-level pit slope design and guidelines to assist with mine design, planning, and cost estimating for the project.

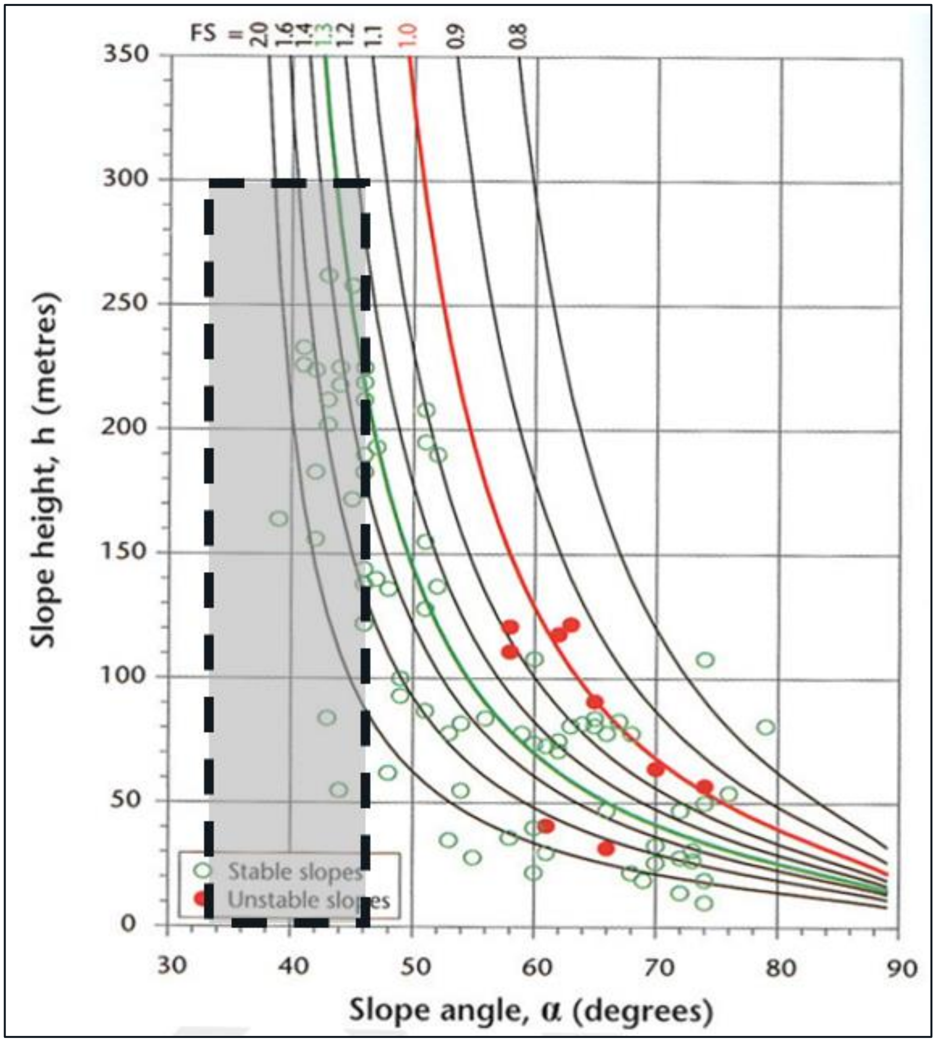
The current geotechnical dataset is considered adequate for PFS-level designs. Where data gaps exist, the engineering geology of the area has been inferred from available data. When quantifying material properties of the rock, ranges of values have been estimated.

Engineering geology interpretations presented in this report should be considered preliminary. Data collected to date may not accurately reflect the rock mass comprising the final open pit walls. Where appropriate, geological features identified should be verified and validated with additional field work and interpretation.

16.3.2.3 Preliminary Open Pit Slope Stability Assessment

The following widely used empirical slope stability chart (Hoek, 1970) demonstrates typical safety factors for a variety of slope configurations. AGP's preliminary guidance for Eskay pit slopes is illustrated.

Figure 16-15: Slope Stability – Height vs Slope Angle vs Factor of Safety (Hoek, 1970)



Preliminary 2D limit equilibrium and FEM analyses were completed for the project by AGP using the 2021 PFS mineral resource interim and ultimate pit shells and the initial pit design guidance described above, to gather insight into inter-ramp and overall slope geotechnics. While currently conceptual in nature, the models have been used to assess and interpret a wide variety of geotechnical and slope stability issues, including variable non-linear and anisotropic rock mass strength criteria, and ground water conditions, as well as the effects of excavation geometry and sequencing.

AGP commonly uses the following approach for target Factor of Safety (FOS) values at the PFS level of study:

- Multi-bench or inter-ramp slopes controlled by discontinuities should achieve a minimum FOS of 1.2.
- Inter-ramp or overall slopes involving shearing through the rock mass and with a low or medium consequence of instability should meet a minimum FOS of 1.3.
- Overall slopes with a high consequence of instability should meet a minimum FOS of 1.5.

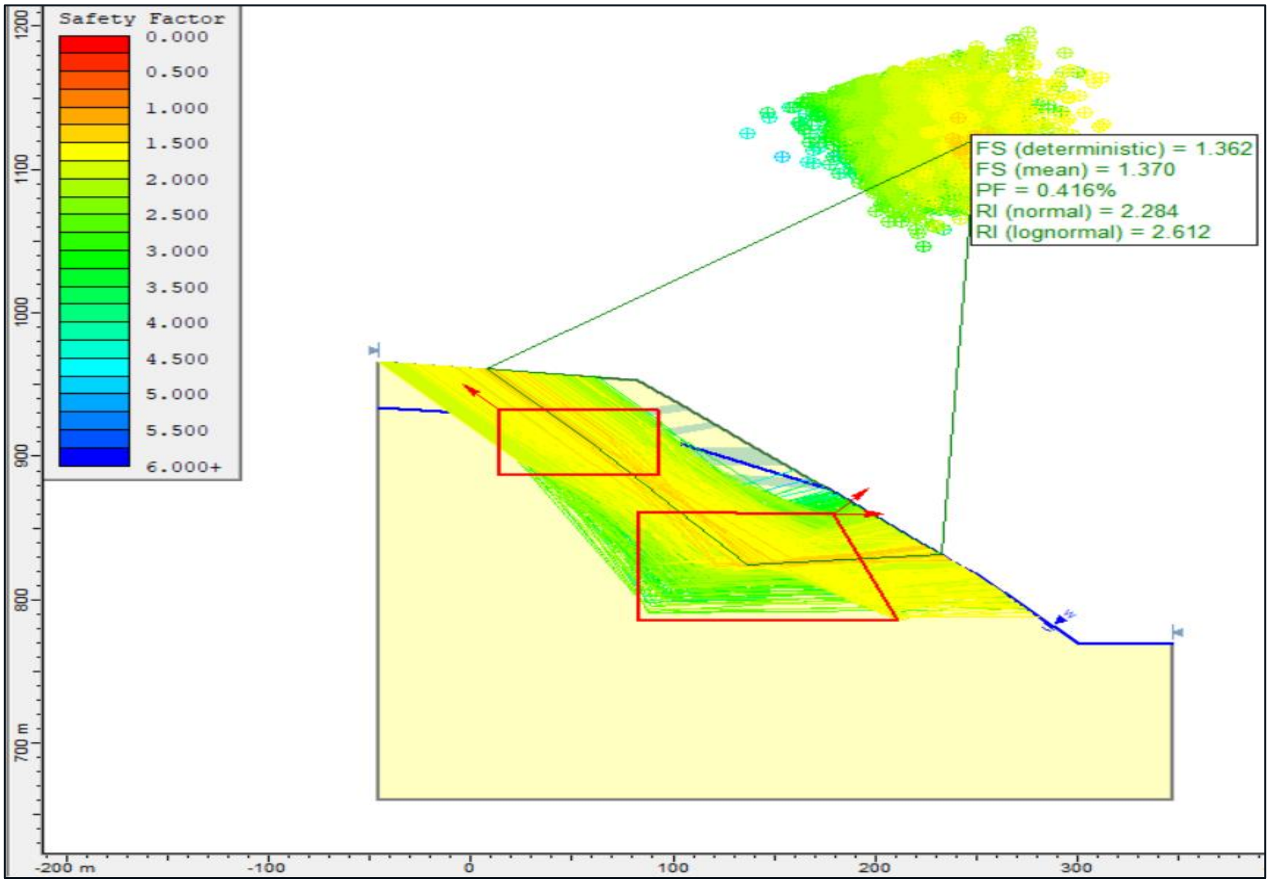
Slope heights ranging from 100 to 300 m with inter-ramp and global slope angles varying from 30 to 45° were analysed under fully to partially saturated conditions. Preliminary analyses indicate the as-designed interim and ultimate ('Phase 3') slopes are predicted to exhibit generally 'stable' conditions for a variety of scenarios, with typical 'minimum' FOS's ranging from ~1.2 to >> 2.0 for inter-ramp and global slopes.

Bench scale slope instabilities have not been assessed for the current study due to insufficient discontinuity and orientation data; bench configurations have included an allowance for reasonable catchment widths to help manage operational challenges that may arise from local bench-scale instabilities.

It is probable that unfavorably oriented geologic structures are present locally within various slope pit sectors, particularly given the size and extents of the pit and the observed variability in discontinuity orientations; it is assumed at present that small bench-scale failures developed along these features can be managed with careful blasting techniques and regular berm maintenance/clearing, wherever access is possible.

On the other hand, both seismic loading and multi-bench-scale to pit-scale structures have the potential to significantly affect overall pit slope stability. The current status and impact of these are both largely unknown. The inclusion of hypothetical adversely oriented faults and bedding planes in the stability analyses indicates potential FOS's less than 1.0, particularly with seismic loading applied. Additional geotechnical investigations are warranted to further determine the location and character of inter-ramp to global-scale conditions and features that may impact stability and mining outcomes.

Figure 16-16: Typical Input and Results from Preliminary 2D Limit Equilibrium Modelling (Sector 4)



Sufficient data has been compiled regarding geotechnical strengths and characteristics of the primary rock types to provide a range of potential pit wall design guidelines. However, numerous assumptions had to be made about the primary controls on rock mass stability, geology, rock mass strength, groundwater pressures, and potential failure mechanisms. As such, the stability models should be considered conceptual in nature. Updated models should be generated and analysed when updates to the mine plan and/or geotechnical domain model become available.

16.3.2.4 Existing Underground Workings

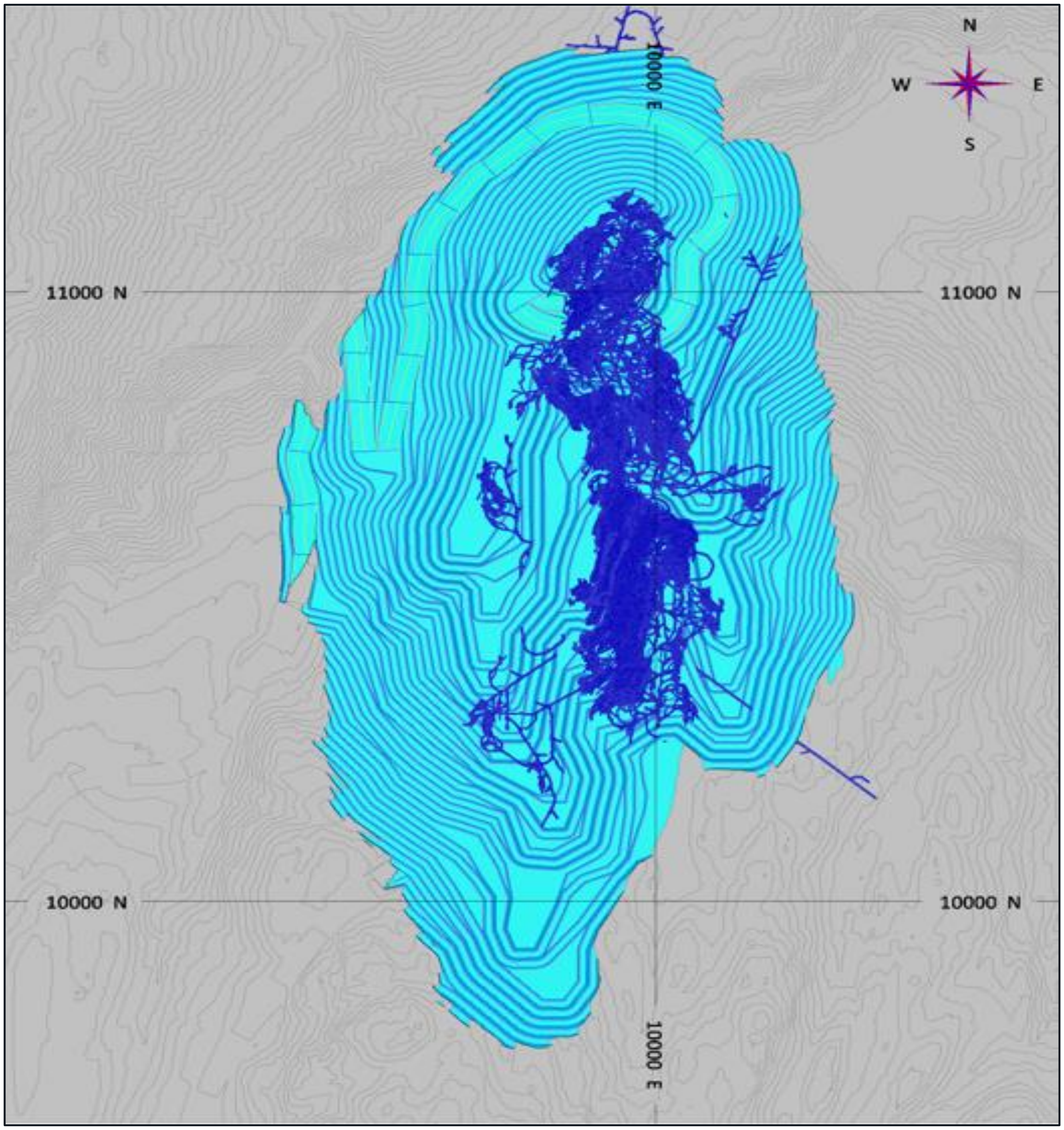
AGP's prior work scope for the 2019 PEA also included a mining geotechnical assessment of the underground mining option. While not pursued over the duration of the PFS study, this work contributed significantly to AGPs conceptualization of ground conditions likely to be encountered during open pit mining. Information and data reviewed for the underground option included previous ground control management plans, ground support recommendations, underground inspection reports, mine plans and previous stope designs etc.

As illustrated in the figure below, the proposed open pit will intersect and mine into the historical underground workings at approximately mid-slope height on the mid to north side of the pit. This will result in increased risks for safely mining in this area and proscriptive plans will need to be developed to adequately mitigate these risks to acceptable levels.

Current best practice for advancing open pit mining operations through existing underground voids is to fill them with either waste or low-grade ore, which removes the void and partially supports the wall rock around the void. If a source of waste rock is available and that will be visibly distinguishable from the mill feed after blasting, dilution can be kept to a minimum, while not tying up feed that could be processed sooner. Failing this, using lower grade feed material to fill the voids is a practical approach.

Although working around known voids will present safety and productivity challenges, a larger concern is the potential for unknown voids. Even with historic mining records and as-built level maps, one must assume that unidentified voids exist. Mining should therefore advance from lower risk areas toward higher risk areas, with probe drilling and perhaps geophysical detection methods.

Figure 16-17: North Ultimate Pit and Existing Underground Workings



16.3.2.5 Data Gap Analysis

A geotechnical data gap analysis has been completed by AGP to determine data requirements to support a FS-level mine design for the proposed open pits.

The available data was evaluated relative to the following considerations:

- Spatial Coverage - ensuring sufficient coverage of rock mass quality and discontinuity orientations of the rock masses in the walls of each major sector of the open pit mine.
- Geological / Geotechnical Domain Coverage - ensuring sufficient characterization of the different geological / geotechnical units (domains) expected to be in and around the open pits.
- Coverage of Major Features - ensuring known faults and other features have been intersected and characterized.
- Orientation Data Bias - ensuring the discontinuity orientation data is sufficiently free of directional bias.
- Orientation Data Quality - ensuring the discontinuity orientation data is of suitable quality.
- Laboratory Strength Testing - ensuring sufficient laboratory strength testing has been completed to characterize the intact rock properties of the different geological units expected at each deposit.

Table 16-8: 2021 PFS Mining Geotechnical Data Gap Analysis

Gap Analysis Criteria	Status	Gaps
Spatial Coverage	Fair	Considerable geotechnical data has been collected to date within most sectors of the open pits; some data gaps remain, particularly for Phase5 north slopes and the potential diversion tunnel.
Geological / Geotech Domain Coverage	Fair	Considerable geological data and geotechnical data currently exists for the project area Limited spatial and geotechnical knowledge exists regarding location and intensity of fault impacted zones, Mudstone intercepts and local characteristics
Coverage of Major Features	Fair	Prelim outcrop and satellite interpretation have been completed; more is required to confirm trends, assess for current unknowns. Initial fault characterization work initiated based on core data, more work required Limited feature continuity and persistence data available
Orientation Data Bias	Fair	Further core orientation data and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Orientation Data Quality	Fair	Further drill core orientation data and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Field and Laboratory Strength Testing	Limited	Limited rock strength testing has been completed to date. UCS, tri-axial, tensile, direct shear and other standard laboratory tests are required to determine / confirm rock strength & deformation parameters, discontinuity strength criteria.

The results of the geotechnical gap analysis indicate several important factors that require additional investigation. For FS designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional higher-quality data. A series of recommended data collection and interpretation tasks are outlined in Section 26 of this report.

16.4 Hydrogeological Considerations

16.4.1 2020 Geotechnical Drilling Program

Packer testing of the 2020 geotechnical boreholes increased the data set of hydraulic conductivity (K) measurements and confirmed elevated K of large faults (e.g., Andesite Creek Fault) identified in previous studies (Golder 1998). Table 16-9 illustrates K variation with depth and lithology and indicates the Hazelton mudstone and andesite are approximately one order of magnitude more conductive than the rhyolite and Bowser mudstone.

Table 16-9: Bedrock Hydraulic Conductivities vs Depths and Lithology

Lithology	Depth (mbgs)	Geomean K (m/s)
Hazelton Mudstone	0 - 50	3.6E-06
	50 - 100	5.8E-07
Andesite	0 - 50	2.1E-06
	50 - 100	1.4E-06
	100 - 200	1.3E-06
Rhyolite	0 - 50	6.0E-07
	50 - 100	2.8E-07
	100 - 200	2.7E-07
Bowser Mudstone	0 - 50	6.3E-07

Overburden deposits have limited thickness and are confined primarily in the valleys of the main creeks in the mining area (Andesite and Argillite Creeks) Groundwater depths are highly variable (artesian to 60 m), reflective of the bedrock environment and partially dewatered historic underground workings.

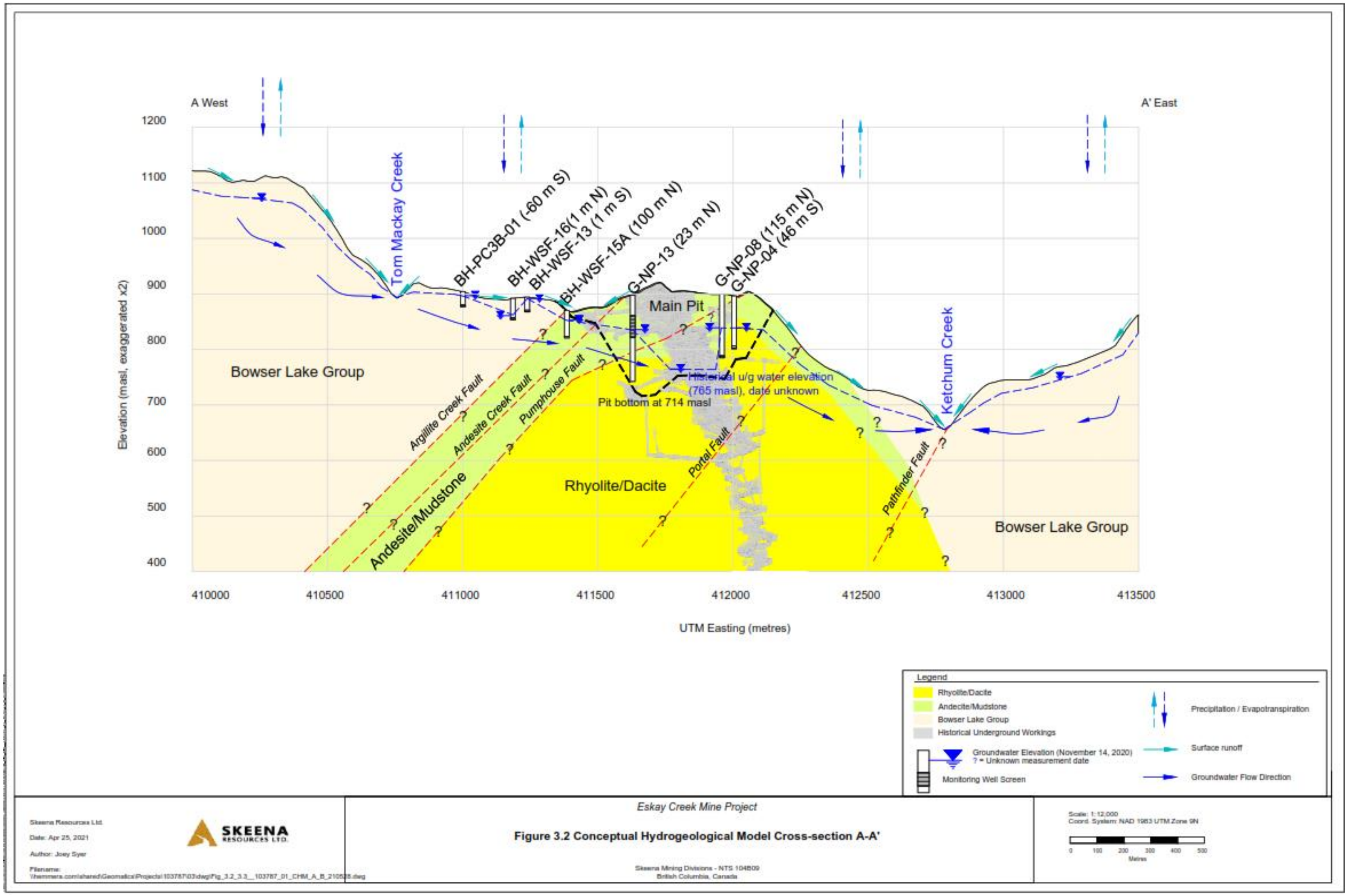
The groundwater levels and K data were used to develop a conceptual model (Figure 16-18) and three-dimensional numerical groundwater model (Groundwater Vistas with add-on module MODFLOW-SURFACT) to predict potential inflows to the open pits.

16.4.2 Groundwater Modelling

A three-dimensional (3D) numerical baseline model was developed that aligns with the updated conceptual hydrogeological model. Recharge zones were assigned based on topographic elevation equivalent to 4% to 17% of the mean annual precipitation (1,938 mm/yr). The model was successfully calibrated to match three targets, including the measured groundwater levels in piezometers, the observed and estimated low flows in Tom MacKay Creek and Eskay Creek, and the measured maintenance pumping rates from the historic underground mine workings. The baseline model was simulated steady state, representing the long-term average groundwater flow conditions.

Average groundwater flow into the ultimate Main Pit for the Base Case is estimated at approximately 818 gpm (or 53 L/s) and up to 2,099 gpm (135 L/s) due to uncertainties in geology and recharge. These predictions do not include precipitation or the expanded Main Pit (Phase 5), which will extend through and to the north of Tom McKay Creek. During operations of the former Eskay Creek Mine, mine dewatering rates from 2000–2004 varied seasonally from a June peak of 82–107 L/s to an April low of 13–38 L/s. Groundwater flow quantity into the South Pit is estimated to be small (up to 10 gpm or 1 L/s) because of its elevation and depth to groundwater, see Figure 16-18.

Figure 16-18: Deposit Geology and Estimated Hydraulic Conductivity



16.4.3 Potential Groundwater Risks and Opportunities Based on Current Mine Plan

The highly fractured nature of the andesite and mudstone lithologies are positive for dewatering but imply potentially larger volumes of contact water to be managed. The historic underground workings are thought to act as a sump and partially dewater the surrounding bedrock and could be used to accelerate pit dewatering.

The groundwater removed from the underground workings shows some elevated zinc concentrations prior to discharge whereas the groundwater quality from monitoring wells is generally good with circum-neutral pH and low metal concentrations with only localized mining impacts indicated. Groundwater, if diverted before entering the pit, could be used in the process and reduce potential water treatment costs.

Diversion of Tom McKay to allow Phase 5 expansion to the north will potentially increase groundwater inflow to the Main Pit. Estimation of these inflows together with pit dewatering modelling will be undertaken in the feasibility study. The 3D model will also be used to support permitting by estimating the changes in groundwater flows and solute transport to the receiving environment.

16.5 Pit Shell Development

The open pit ultimate size and phasing requirements were determined with various input parameters including estimates of the expected mining, processing and general and administrative (G&A) costs, as well as metallurgical recoveries, pit slopes and reasonable long-term metal price assumptions. AGP worked together with Skeena personnel to select appropriate operating cost parameters for the proposed Eskay Creek open pit. The mining costs are estimates based on cost estimates for equipment from vendors and previous studies completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various dump locations. Process costs and a portion of the G&A costs were provided by Ausenco and Skeena based on preliminary costing results.

The parameters used are shown in Table 16-10. The net value calculations are in United States dollars (US$) unless otherwise noted. Costs and revenues are converted to Canadian dollars for use in pit shell determination. The mining cost estimates are based on the use of 144 t trucks using an approximate WRSF configuration to determine incremental hauls for mineralized material and waste. The smelting terms and recovery assumptions are based on creating 20 - 45 g/t gold bulk concentrates.

Table 16-10: Pit Shell Parameter Assumptions

Description	Units	Value	Gold Value	Silver Value
Exchange rates				
CAD	US$ =	1.26		
Resource model				
Block classification used		M+I		
Block model height	m	4		
Mining bench height	m	8		
Metal prices				
Price	$/oz		1475	20
Royalty	%		2%	2%

Smelting, refining, transportation terms				
Payable for 20 g/t Au concentrate	%		72.88	71.25
Payable for 25 g/t Au concentrate	%		75.5	73
Payable for 35 g/t Au concentrate	%		80.75	76.5
Payable for 45 g/t Au concentrate	%		86	80
Minimum deduction	unit, g/dmt		0	0
Participation (on profits)	%		100	100
Bulk concentrate treatment charge	$/dmt	0		
Refining	$/oz		0	0
Concentrate moisture	%	12		
Transit losses	%	0.5		
Concentrate transportation cost	C$/wmt	141.60		
Metallurgical Information				
Recovery [1]	A constant	-0.3112 * (Au con g/t) + 96.753		
	B constant	-0.0075 * (Au con g/t) -1.0716		
	%		A*(1-EXP(B*Au))	0.93 * Au rec
Power cost				
Cost of power	C$/Kwhr	$0.05		
Fuel cost				
Diesel fuel cost to site	C$/ l	$1.03		
Mining cost [2]			NAG	PAG
Waste base rate - 880 elevation	C$/t		3.02	3.71
Incremental rate - above	C$/t/4m bench		($0.007)	($0.007)
Incremental rate - below	C$/t/4m bench		$0.016	$0.016
Mill feed base rate - 880 elevation	C$/t		2.43	2.43
Incremental rate - above	C$/t/4m bench		$0.008	$0.008
Incremental rate - below	C$/t/4m bench		$0.013	$0.013
Processing [3]				
Processing cost	C$/t mill feed	$22.80		
Water treatment	C$/t mill feed	$1.70		
Total processing cost	C$/t mill feed	$24.50		
General and administrative cost				
G&A cost	C$/t mill feed	$6.06		
Total process and G&A				
Process + G&A	C$/t mill feed	$30.56		

Note: [1] maximum NSR value used from 20–45 g/t Au concentrate grades, but if no Fe or S values available 25 g/t Au concentrate used with Au rec = 92%, Ag rec = 97%. [2] mining costs based on using 144 t haul trucks. [3] process costs based on 2.5 Mt/a dry throughput

Wall slopes for pit optimization were based on review of available historical underground data and observations as outlined in Section 16.3 from the 2020 field program. Allowances were made for ramps in the slopes to determine an overall angle for use in the L–G routine. The overall slope angle values are shown in Table 16-11. Slopes were flattened as required due to inclusion of haulage ramps.

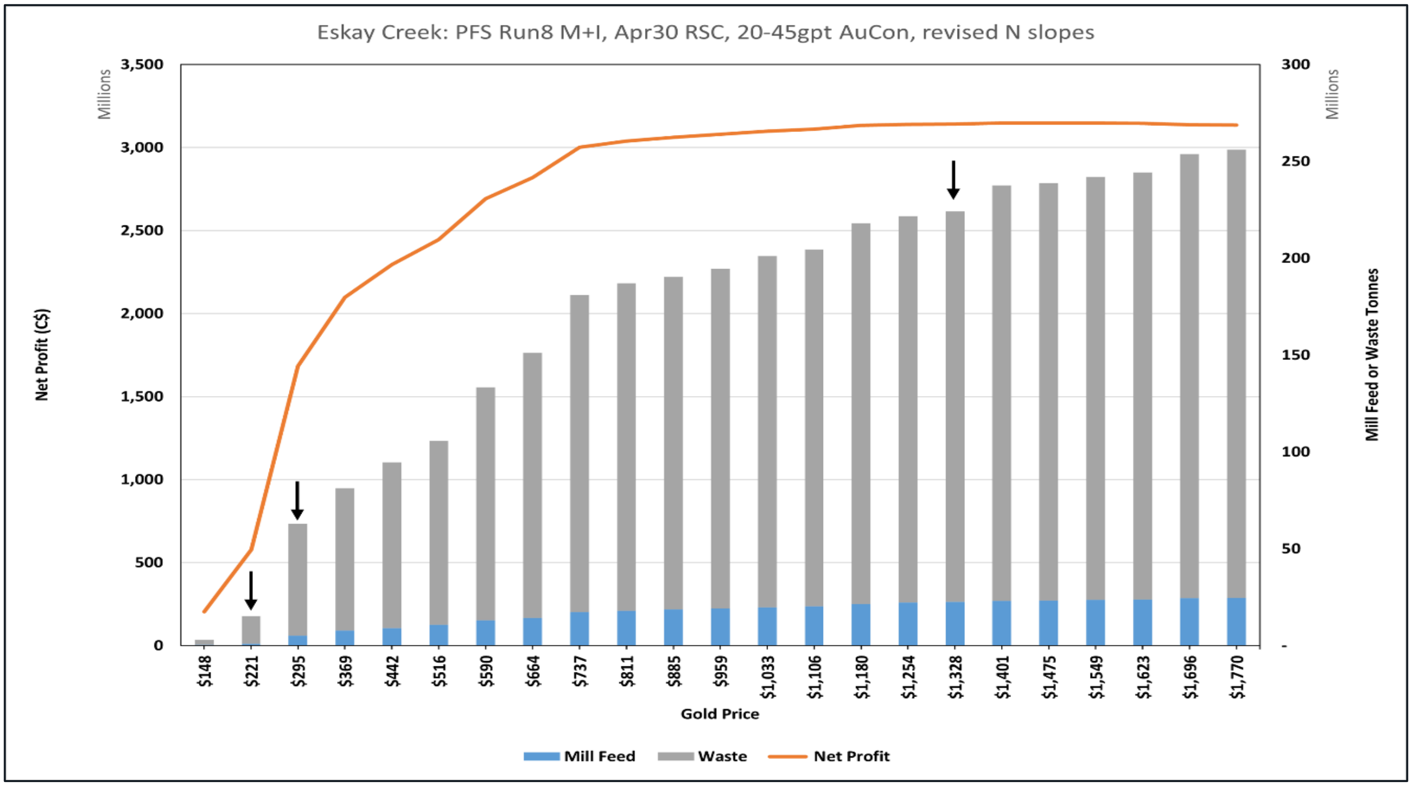
Nested L–G pit shells were generated to examine sensitivity to the gold and silver prices with a target of US$1,450/oz Au and US$20.00/oz Ag. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Undiluted Indicated and Inferred material was used in the analysis. The net smelter return (NSR) was varied by applying revenue factors of 0.10 to 1.20 at 0.05 increments, to generate a set of nested L–G shells. The chosen set of revenue factors result in an equivalent gold price varying from US$148/oz up to US$1,770/oz. All other parameters were fixed. The resulting nested pit shells assist in visualizing natural breakpoints in the deposit and selecting shells to act as design guidance for phase design. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell using US$1,450/oz Au and US$20.00/oz Ag. No creek restrictions were used to restrict the pit shells near Tom Mackay Creek and Ketchum Creek. Mill feed material/waste tonnages and potential net profit were plotted against gold price and are displayed in Figure 16-19.

Figure 16-19 contained several break points in the pit shells. These were used as a guide for sequencing pit phase designs. With each incremental the increase in the waste tonnage, and to a lesser degree the mill tonnage, the undiscounted net profit also increased. In the case of the first break point shown at US$221/oz Au, the cumulative waste tonnage is 14.2 Mt, with a corresponding mill feed tonnage of 999 kt or a strip ratio of 14.3:1. The net profit also increased beyond this point, showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 18% of the net value of a $1,450/oz pit but with only 7% of the waste of the larger pit shell.

Table 16-11: Pit Shell Slopes

Slope Domain	SLOPN Code	Overall Slope (º)	Description
Weak slopes	1	34	
Competent slopes	2	46	
Weak slopes at north end of pit	3	29.3	One 30.2 m-wide ramp, 100m slope height
Competent slopes at north end of pit	4	32.5	One 30.2 m-wide ramp, 50m slope height

Figure 16-19: Eskay Creek Potential Profit vs. Price by Pit Shell



Note: Figure prepared by AGP, 2021.

The second break point was at US$295/oz Au. The incremental waste tonnage from the first break point is 43.4 Mt, with a corresponding increase in mill feed tonnage of 4.2 Mt or a strip ratio of 10.4:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price. This pit shell was used for the pit design of Phase 2 and also influenced phase 1. There are significant waste tonnages in the next higher pit prices to achieve the next increases in profit. The cumulative value of the first two break points was 53% of the US$1,450/oz Au pit shell but with only 27% of the waste movement of the larger pit required. This pit shell ran significantly further north than the first break point and was located closer to the east side of the deposit.

The final pit shell selected represented the ultimate pit at US$1,328/oz Au. This resulted in a substantial jump in the waste tonnage from the second break point to the third break point by 158 Mt with a gain of 18.3 Mt of feed material for an incremental strip ratio of 8.6:1. The net profit continues to increase beyond the third break point, although at a flatter rate than in earlier breakpoints. The cumulative value of the first three break points was 100% of the US$1,450/oz Au pit shell with 94% of the waste movement of the larger pit required. Limited potential pit value was available beyond this pit shell to cover schedule discounting another phase.

An additional pit shell could potentially be included at US$ 664 or US$737 Au. However, access would become more difficult as shells ran the length of the deposit and backfill areas would likely not be available as early in the schedule. Preliminary schedules also indicated that bench advance would be a primary constraint to achieve the desired mill throughput rates, so narrow phases were minimized so that more efficient mining could be possible.

16.6 Dilution

The open pit resource model was provided as an undiluted percentage type model, such that the grades from the wireframes were reported into separate percentage parcels of ore and waste in each block. The provided feed percentage values exclude underground workings and all material within 0.2 m of their original solids. These underground solids were viewed on several plan views with ORE% values and the workings appear to have been properly adjusted in a consistent manner. As the mine workings were mostly backfilled, they were included in the waste percentage.

To account for mining dilution, AGP modelled contact dilution into the in-situ resource blocks. To determine the amount of dilution, and the grade of the dilution, the size of the block in the model was examined. The block size within the model was 9 x 9 m in plan view, and 4 m high. Mining would be completed on 8 m lifts for waste and 4 m lifts for mill feed, if required, and the equipment selected is capable of mining in that manner.

The percentage of dilution is calculated for each contact side using an assumed 1.25 m contact dilution distance. This dilution skin thickness was selected by considering the spatial nature of the mineralization, proposed grade control methods, GPS-assisted digging accuracy, and blast heave.

If one side of a mineralized block above cut-off is in contact with a waste block, then it is estimated that dilution of 13.9% (1.25 m/9 m) by volume would result. If two sides are contacting, it would rise to 27.8%. Three sides would be 41.7%, and four sides 55.6%. Four sides represent an isolated block of mill feed.

All mineralized blocks in the resource model contain grade values; however, the material outside the mineralized shapes have no grade estimates and have been treated as though the gold and silver grades are zero for dilution purposes. The NSR value per tonne that was stored to the block model previously was used as the grade for cut-off application. As the NSR is inclusive of all revenues and royalties, applying a C$30.56/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks. This cut-off grade value represents the preliminary process and site G&A costs.

Using this marginal cut-off grade, the first step is to identify the mill feed and waste blocks in the model. The second step is to add dilution mass and metal into the mill feed blocks from the neighbouring waste blocks. The third step is to remove the dilution mass from the contact waste blocks to achieve a mass balance.

AGP has an in-house routine that applies the above three dilution steps to define new items called DDEN, DORE%, DWAS%, as well as the grade items (DAU, DAG, DPB, DZN, DCU, DAS, DHG, DSB, DSPCT and DFE). The default waste blocks would receive DORE%=0.

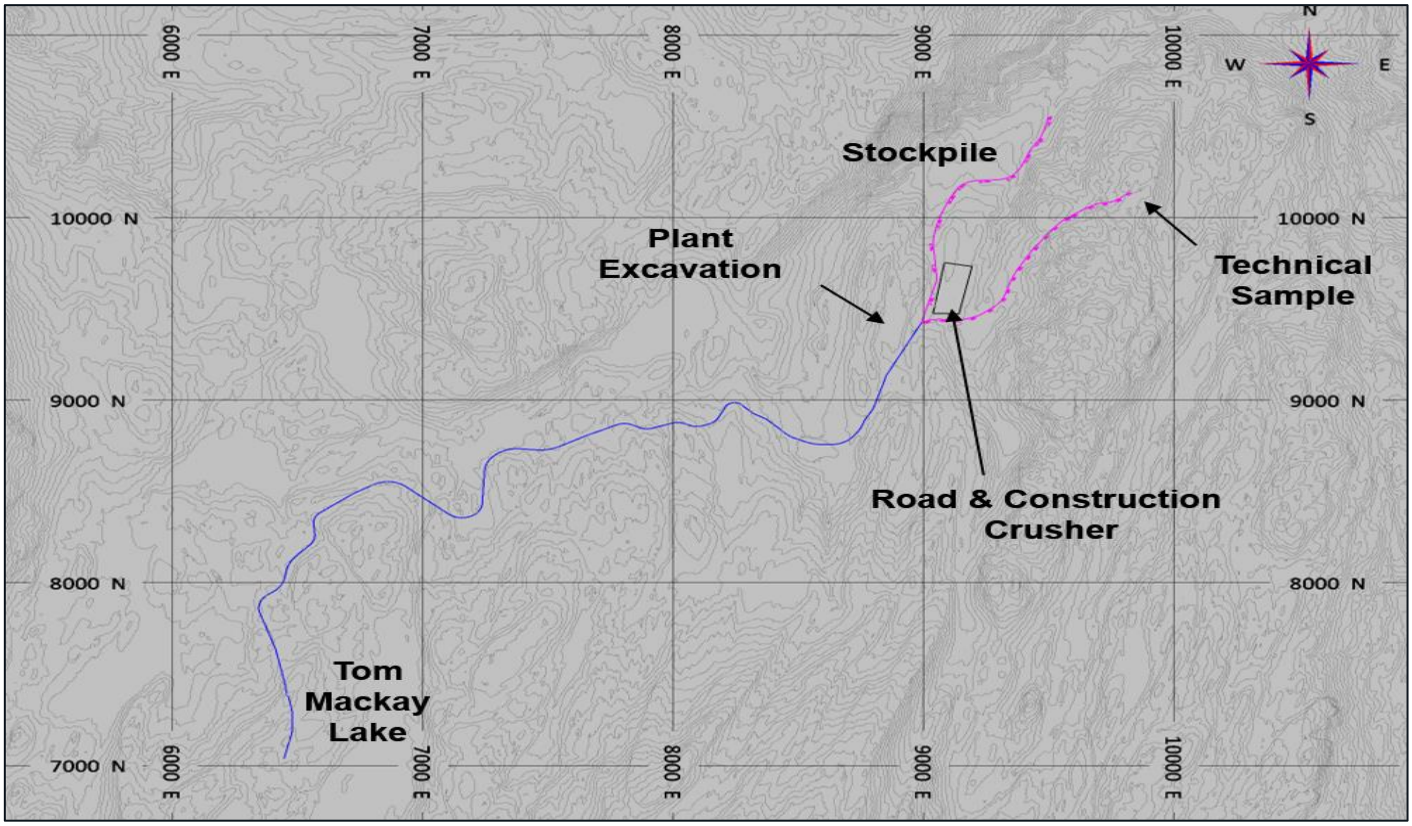
In this manner, the contact diluted blocks were included in the tonnage and grade calculation of mill feed tonnes. The mill feed tonnage report was then run with the block model DORE% item to report out the diluted tonnes and grade.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 21.1% more tonnes and 16.6% lower gold grade than the in-situ feed summary. The grade dilution percentage was lower than the feed tonnage percentage since the mineralized waste blocks included some grade. The average grade of the dilution material was 0.16 g/t Au and 3.65 g/t Ag. AGP considers these dilution percentages to be reasonable considering the expected seasonal working conditions as well as mining through underground workings.

16.7 Pre-Production

Mine development activities will occur at site during the three years of pre-production. Road construction will be the initial primary activity with NAG waste being sourced from a technical sample phase in the north pit as well as the plant site excavation. All PAG waste is intended to be submerged in the Tom Mackay Lake, so a road will need to be established to it from the technical sample phase. A road will also be required between the technical sample pit and the stockpile location near the future crusher. The approximate initial road locations are shown in Figure 16-20. The initial roads will be established in year -3 of the mine schedule so that the technical sample can be taken. Years -2 and -1 of the production schedule will be used to establish the upper mining benches and stockpile material sufficient to ramp up the process plant to a throughput rate of 2.9 Mtpa by the end of Year 1.

Figure 16-20: Initial Roads for Pre-Production



Note: Figure prepared by AGP,2021

16.8 Pit Designs

Pit designs were developed for the north and south pit areas. The north pit will consist of five main phases, while the south pit will only contain a single small phase. The pit optimization shells used to determine the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters outlined in Table 16-12 were applied to pit designs. Based on the analysis of recent field samples, it was recommended to use two groups of lithologies to define separate areas of weak and competent slopes. The location of the various lithologies in the north ultimate pit is shown in Figure 16-21.

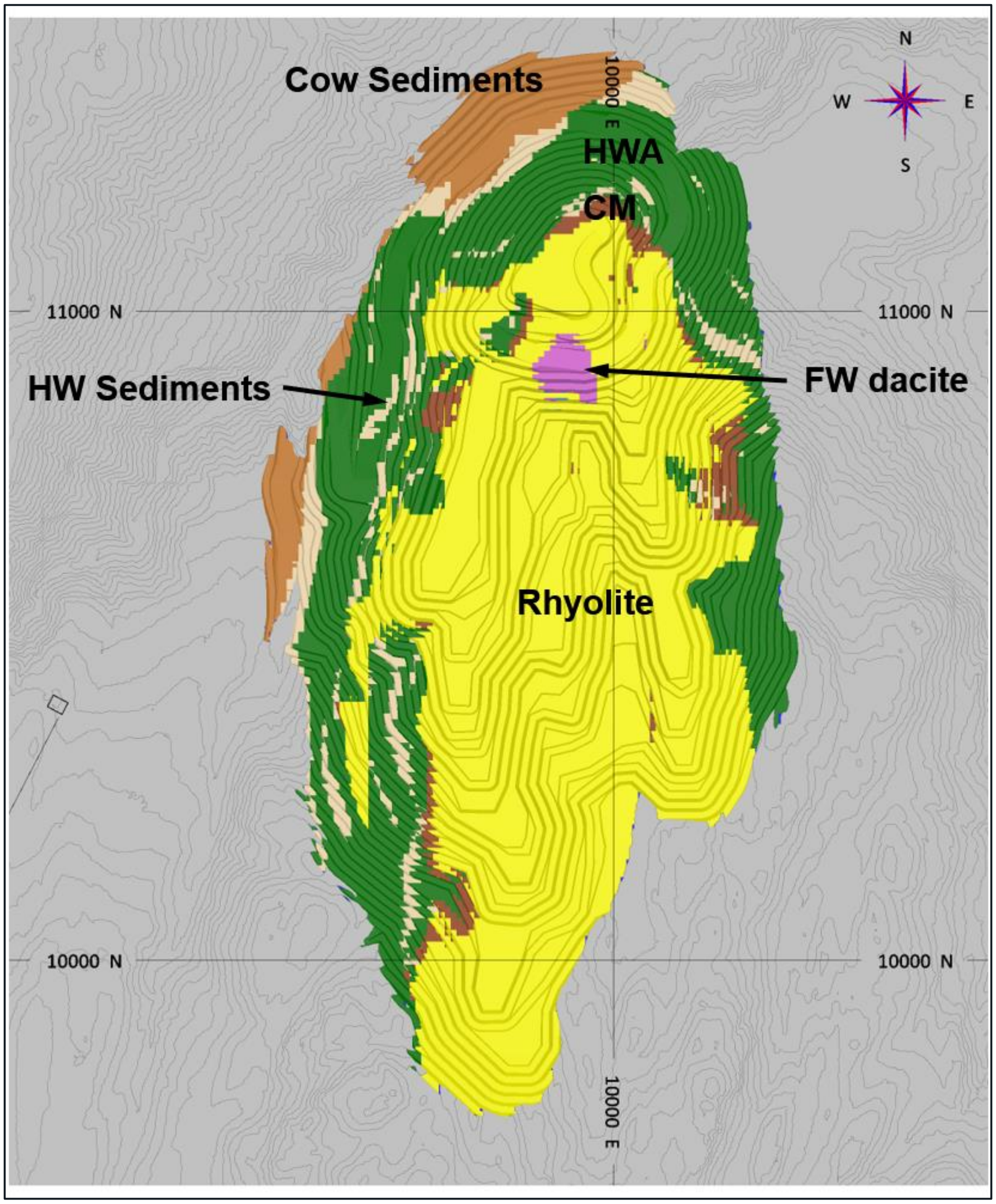
Table 16-12: Geotechnical Parameters for Pit Design

Lithology	SLOPE Domain Code	Inter-Ramp Angle (degrees)	Bench Face Angle (degrees)	Height Between Berms (m)	Catch Bench Width (m)
Weak slopes - Contact Mudstone, HW Andesite, HW mudstone	1	34	70	8	8.95
Competent slopes - Cow Sediments, Rhyolite, FW sediments, FW dacite	2	46	70	16	9.63

Note: 8 m bench heights during mining

Equipment sizing for ramps and working benches is based on the use of 144 t rigid-frame haul trucks. The operating width used for the truck is 6.9 m. This means that single lane access is 23.3 m (twice the operating width plus berm and ditch) and double lane widths are 30.2 m (three times the operating width plus berm and ditch). Ramp gradients are 10% in the pit and WRSF for uphill gradients. Working benches were designed for 35–40 m minimum mining width on pushbacks. As the haul road grades exceed 5%, runaway lanes or retardation barriers will need to be incorporated into final execution designs as the project progresses to more detailed stages.

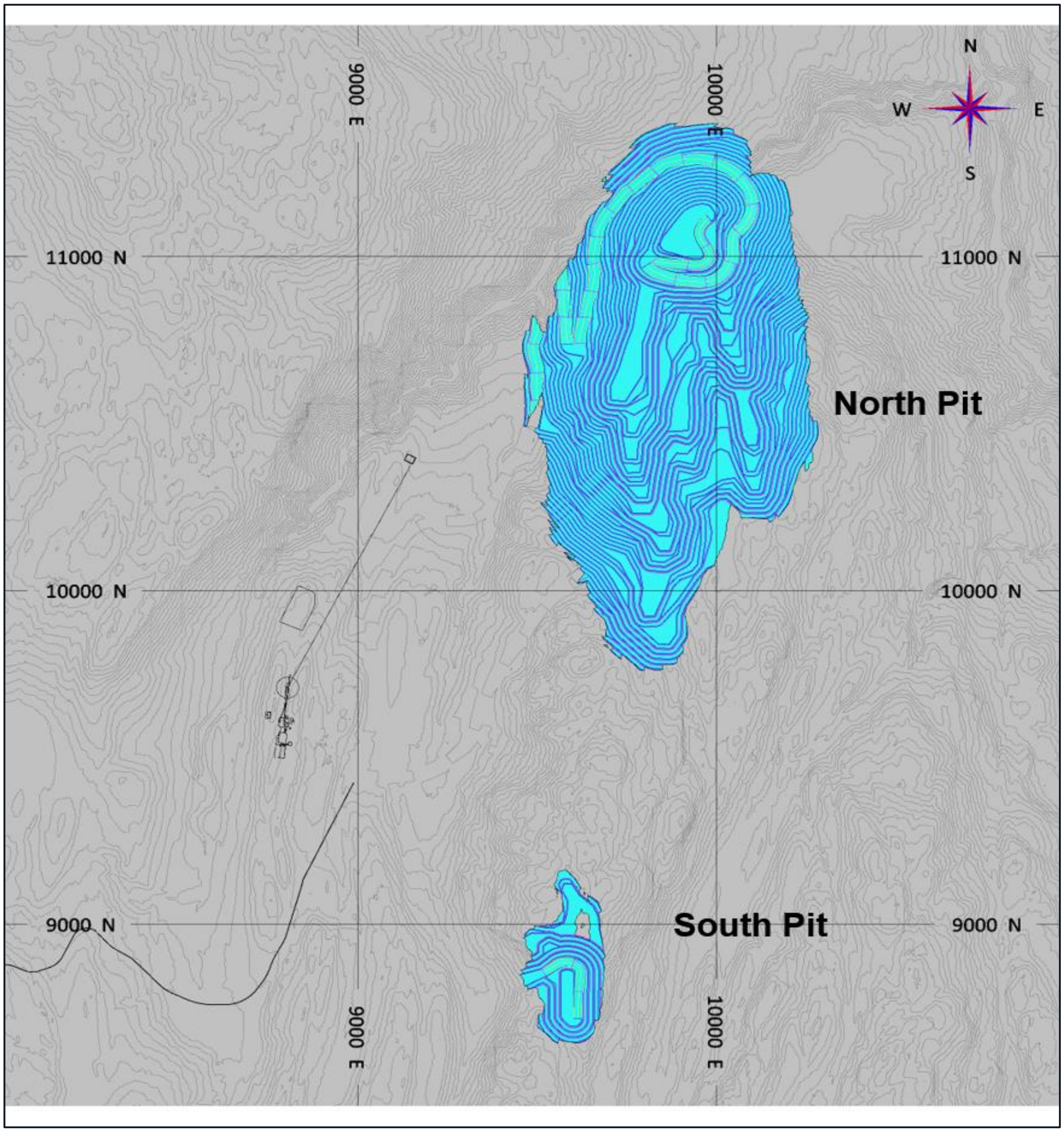
Figure 16-21: Rock Types in Ultimate North Pit



Note: Figure prepared by AGP, 2021

The north and south pits are displayed in Figure 16-22. The south pit is significantly smaller than the north pit and is likely to be mined near the end of the mine schedule.

Figure 16-22: Proposed Eskay Creek North and South Pits



Note: Figure prepared by AGP, 2021.

Tonnes and grade for the final pit designs are reported in Table 16-13 using the diluted tonnes and grade from the model and a mining recovery of 98% to account for additional mineralized material losses. Positive marginal block values from the pit optimization run were used to determine mill feed material blocks.

Table 16-13: Final Design – Phases, Tonnages, and Grades

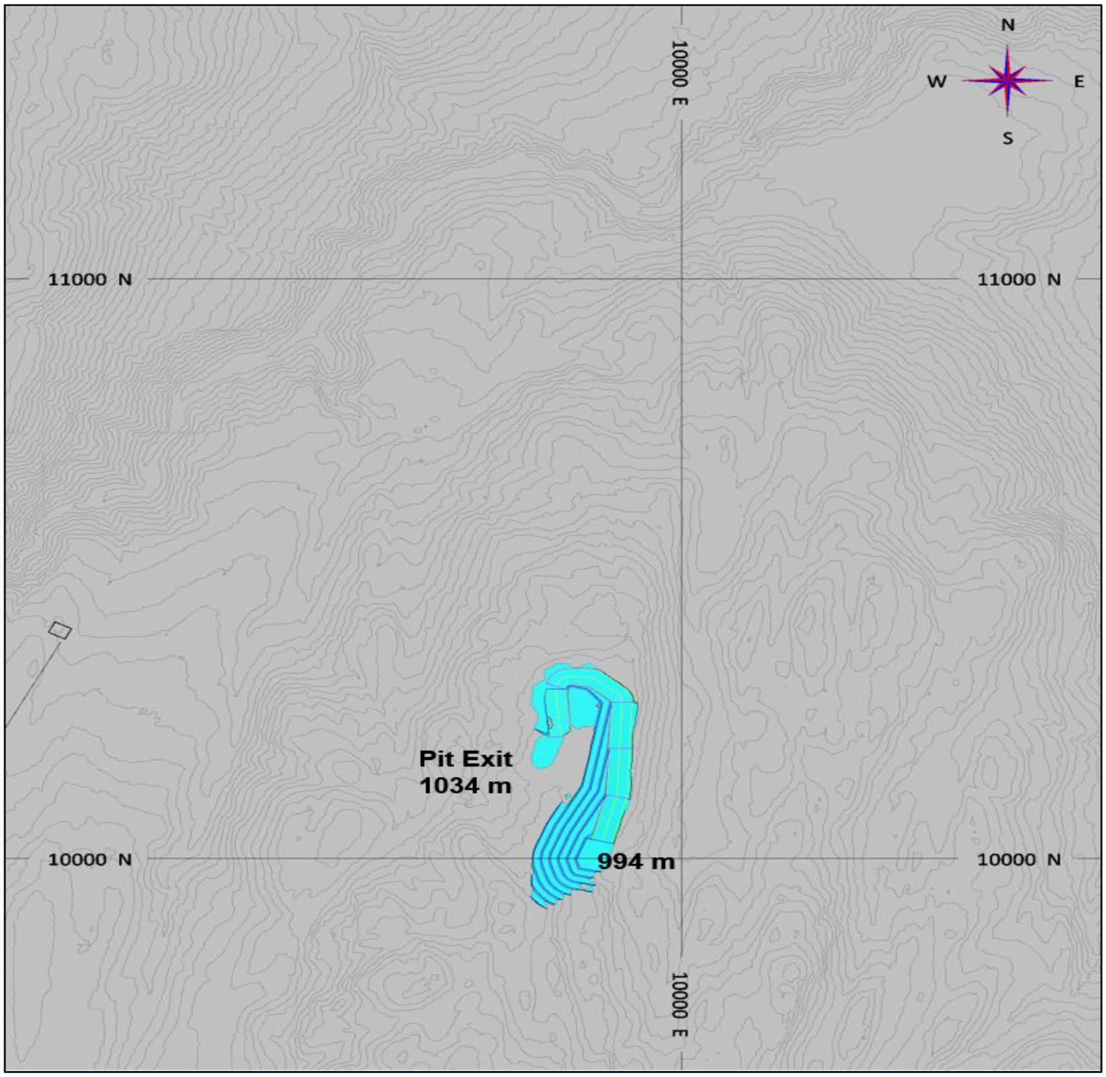
Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	As (ppm)	Hg (ppm)	Sb (ppm)	S (%)	Fe (%)	Waste (Mt)	Total (Mt)	Strip Ratio
North Phase TS	0.03	4.80	55	0.04	0.08	0.018	4,911	67	431	1.4	1.9	1.2	1.2	43.0
North Phase 1	3.58	3.92	76	0.10	0.18	0.020	2,156	169	1,678	1.6	1.6	22.4	26.0	6.3
North Phase 2E	0.28	1.40	68	0.03	0.08	0.008	385	17	7,528	1.7	2.6	2.4	2.7	8.5
North Phase 2	7.54	4.30	156	0.57	0.92	0.100	644	84	1,950	0.9	1.3	74.9	82.4	9.9
North Phase 3	13.06	2.96	70	0.58	0.90	0.076	412	17	611	0.9	1.3	105.6	118.6	8.1
South Phase 1	1.93	1.82	59	0.08	0.11	0.012	953	5	245	0.7	1.1	5.2	7.1	2.7
Total	26.42	3.37	94	0.47	0.74	0.070	758	56	1,185	1.0	1.3	211.6	238.0	8.0

The phase designs are described in further detail in the following sub-sections.

16.8.1 North Phase TS (Technical Sample)

A technical sample will be mined in Year-3 so that process performance of the mill can be evaluated with a larger sample than drill hole samples. This phase has been located near the top of the north pit on the east side of the ridge. Very few shallow targets were available due to the plunging nature of the orebody away from topography. Phase bench elevations range from 1050 m down to 994 m. The mill feed sample is available on the bottom two benches of this phase. The north phase TS design is shown in Figure 16-23.

Figure 16-23: Proposed North Phase TS

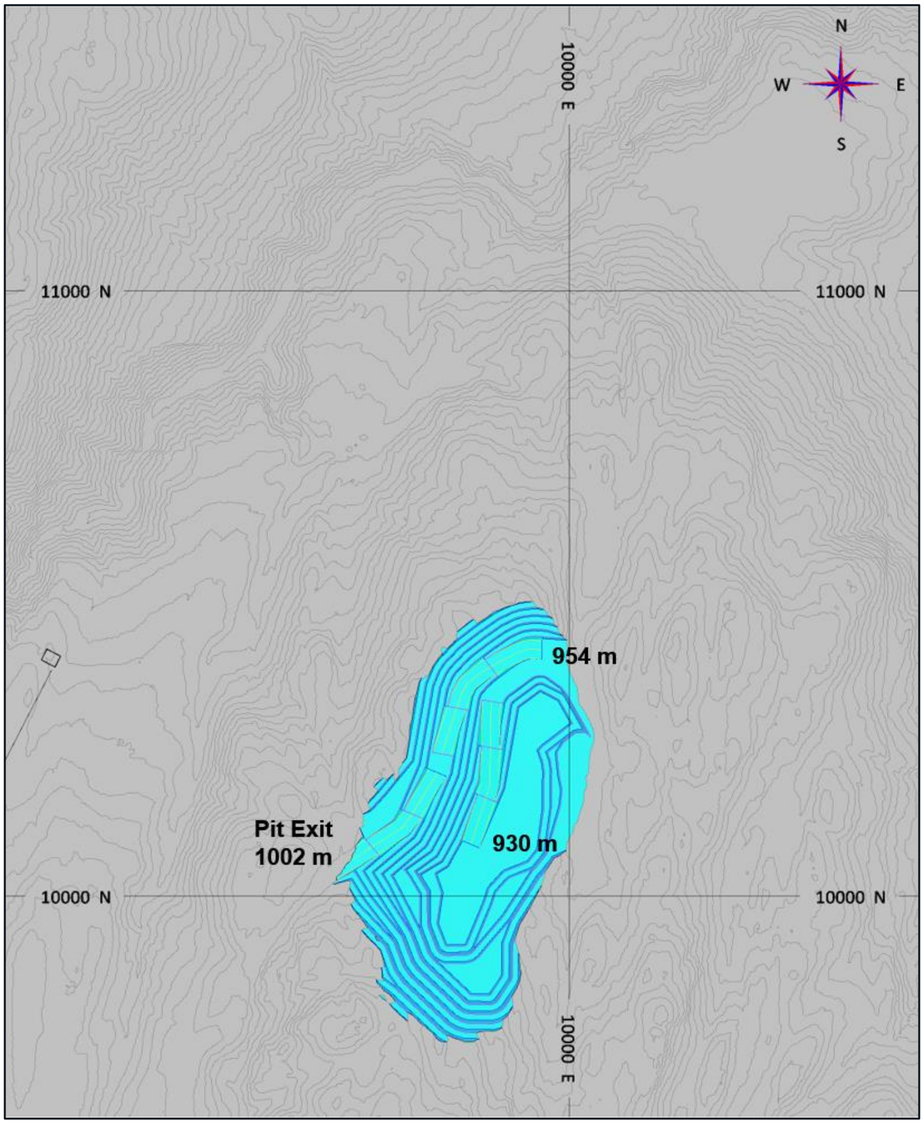


Note: Figure prepared by AGP, 2021.

16.8.2 North Phase 1

Phase 1 will start being mined in Year-3 with the technical sample, with the upper four benches to be available as quarry material for construction purposes. This phase begins mining at the upper elevations of the north pit and targets a shallow high-grade zone for mill feed. Phase bench elevations will range from 1066 masl down to 930 masl. All waste and mineralized material accesses will be on the west side of the phase, where the WRSFs and mill feed crusher will be located. The north phase 1 design is shown in Figure 16-24.

Figure 16-24: Proposed North Phase 1

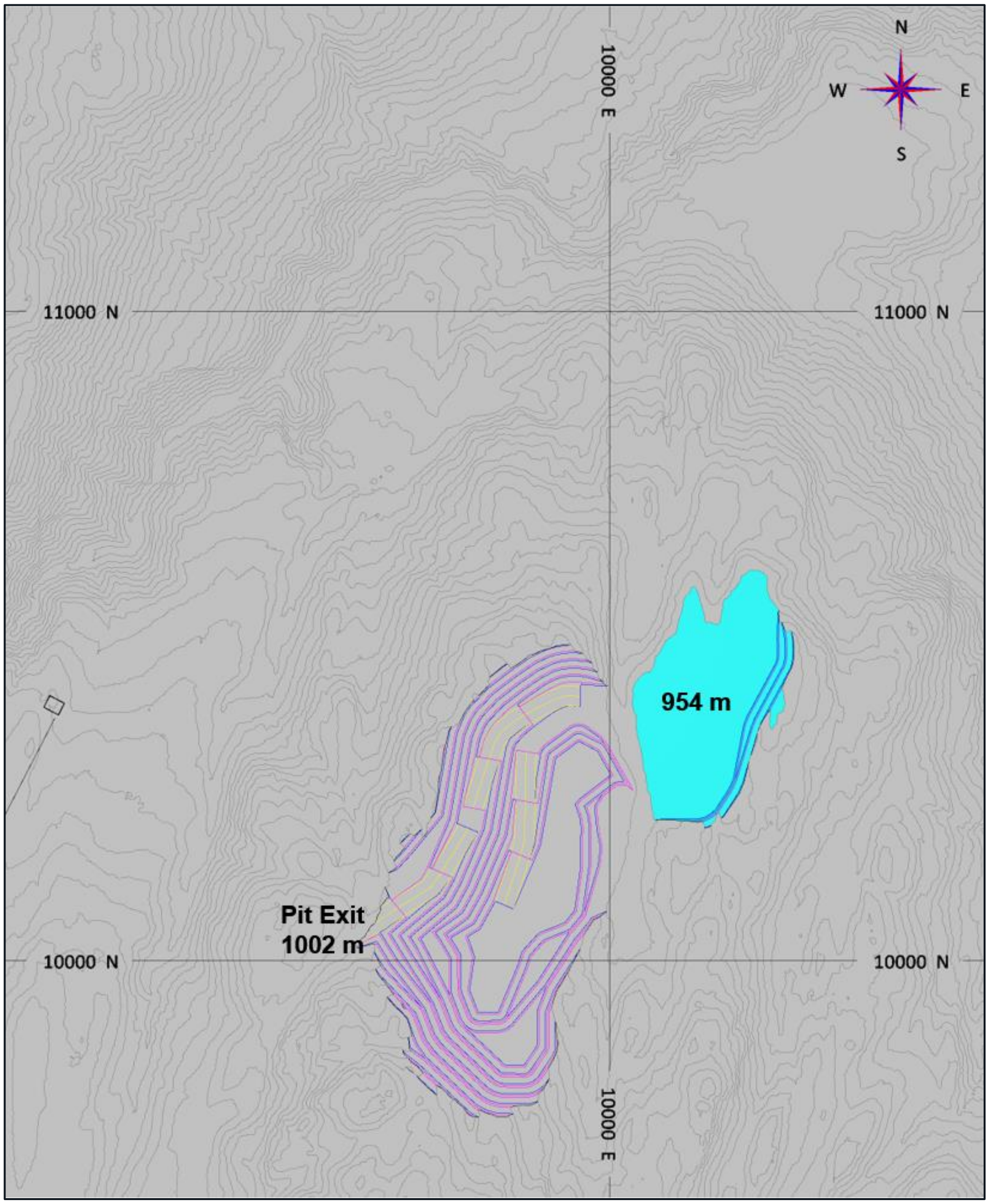


Note: Figure prepared by AGP, 2021.

16.8.3 North Phase 2E

Phase 2E will also be accessed from the west side of the pit via the phase 1 access ramp. This phase should be mined before mining Phase 2 to elevations below 954 m. Phase bench elevations will range from 994 masl down to 954 masl. The north phase 2E design is shown in Figure 16-25.

Figure 16-25: Proposed North Phase 2E

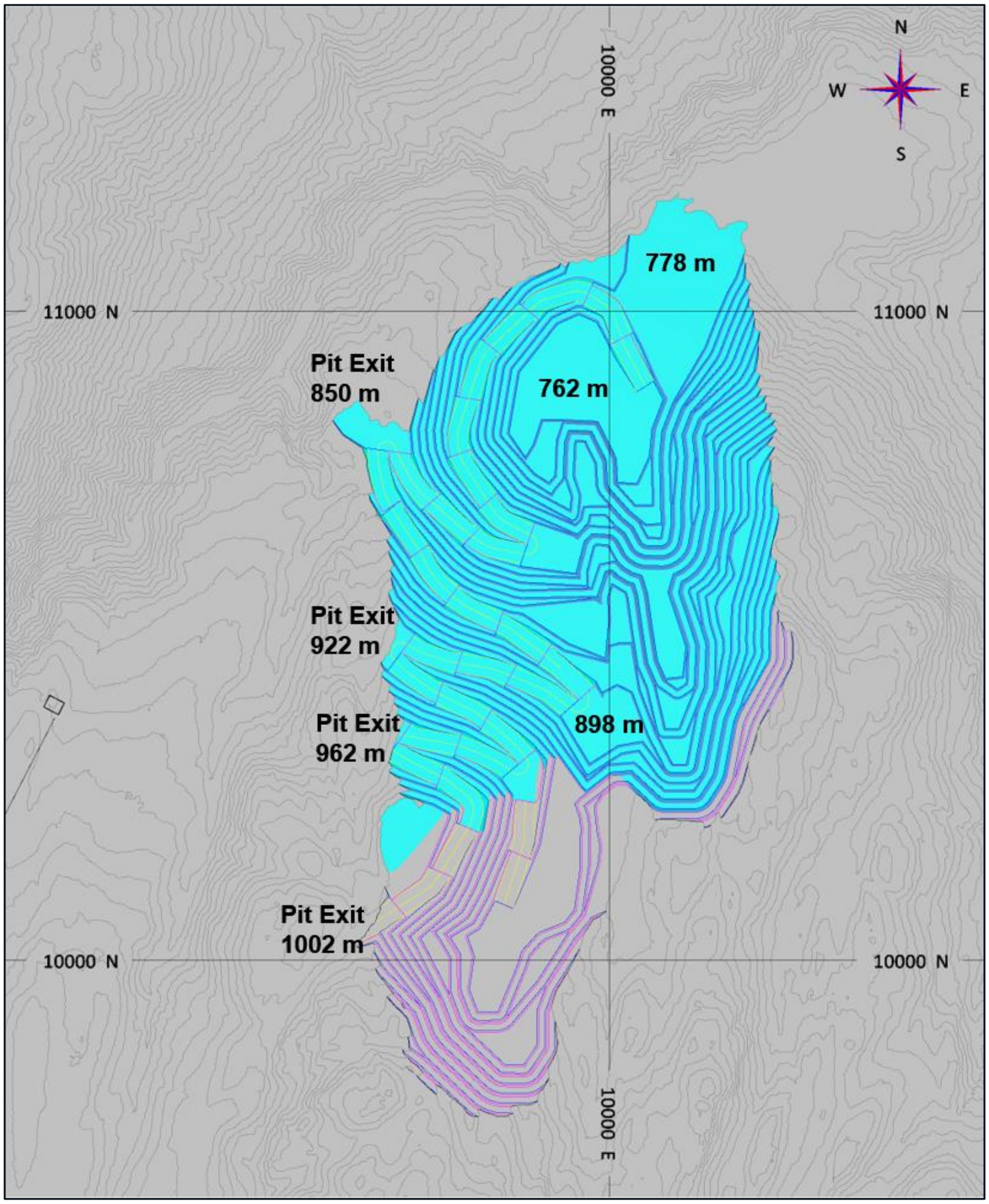


Note: Figure prepared by AGP, 2021.

16.8.4 North Phase 2

Phase 2 will also be accessed from the west side of the pit. As the phase advances down benches to the north of phases 1 and 2E, haul road accesses will be left in place along the west side so that they may be use by later phases. Phase bench elevations will range from 1010 masl down to 762 masl. The north phase 2 design is shown in Figure 16-26.

Figure 16-26: Proposed North Phase 2

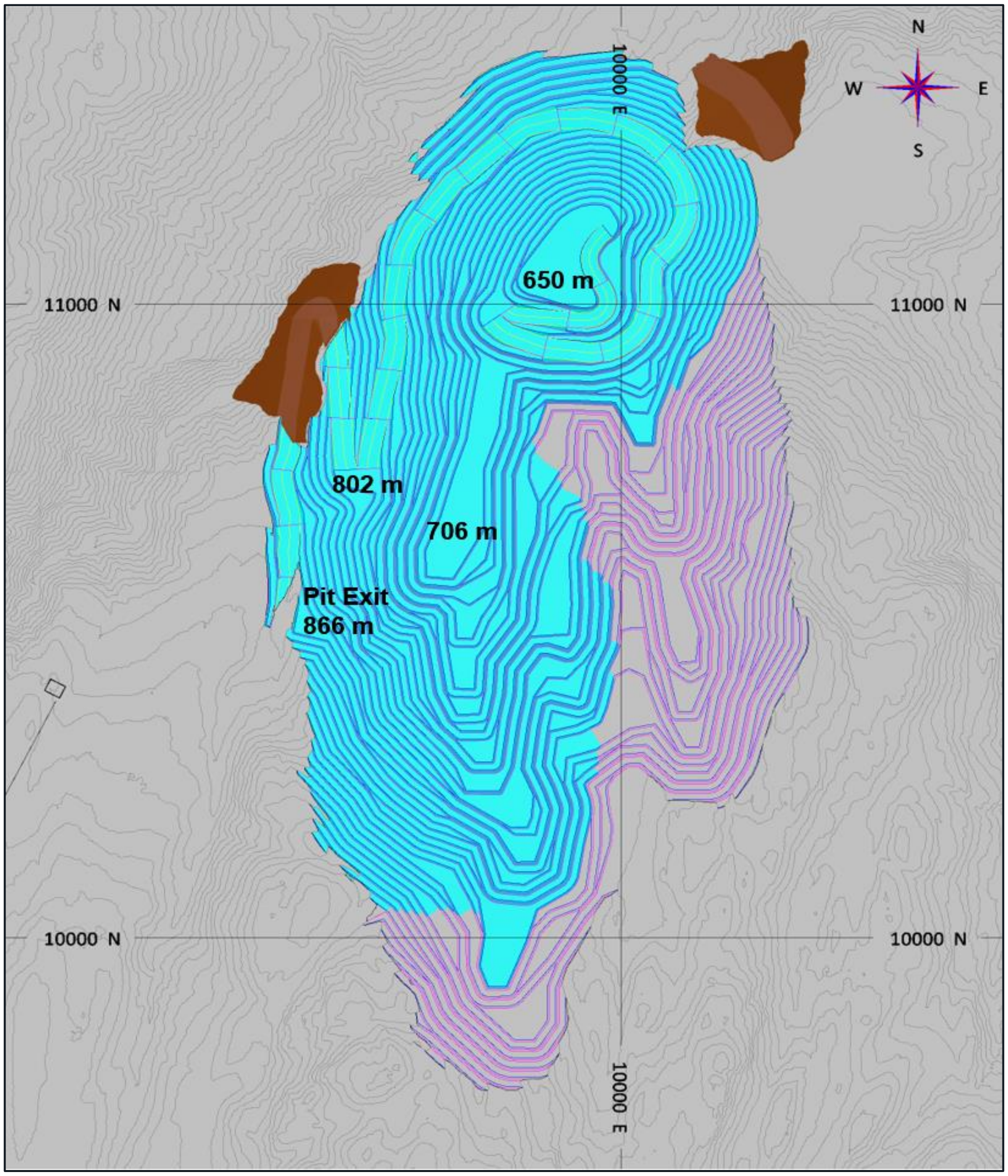


Note: Figure prepared by AGP, 2021.

16.8.5 North Phase 3

Phase 3 is the final north phase and extends across Tom Mackay Creek. This design includes a switchback at 826 m elevation that is created with waste rock across the creek. The access on the north side of the creek is also created with waste rocks at the northeast corner of the pit at 778 m elevation. The pit exit at 866 m elevation pit is where the mined material will leave the pit near the crusher. Phase bench elevations will range from 994 masl down to 650 masl. The phase 3 design is shown in Figure 16-27.

Figure 16-27: Proposed North Phase 3

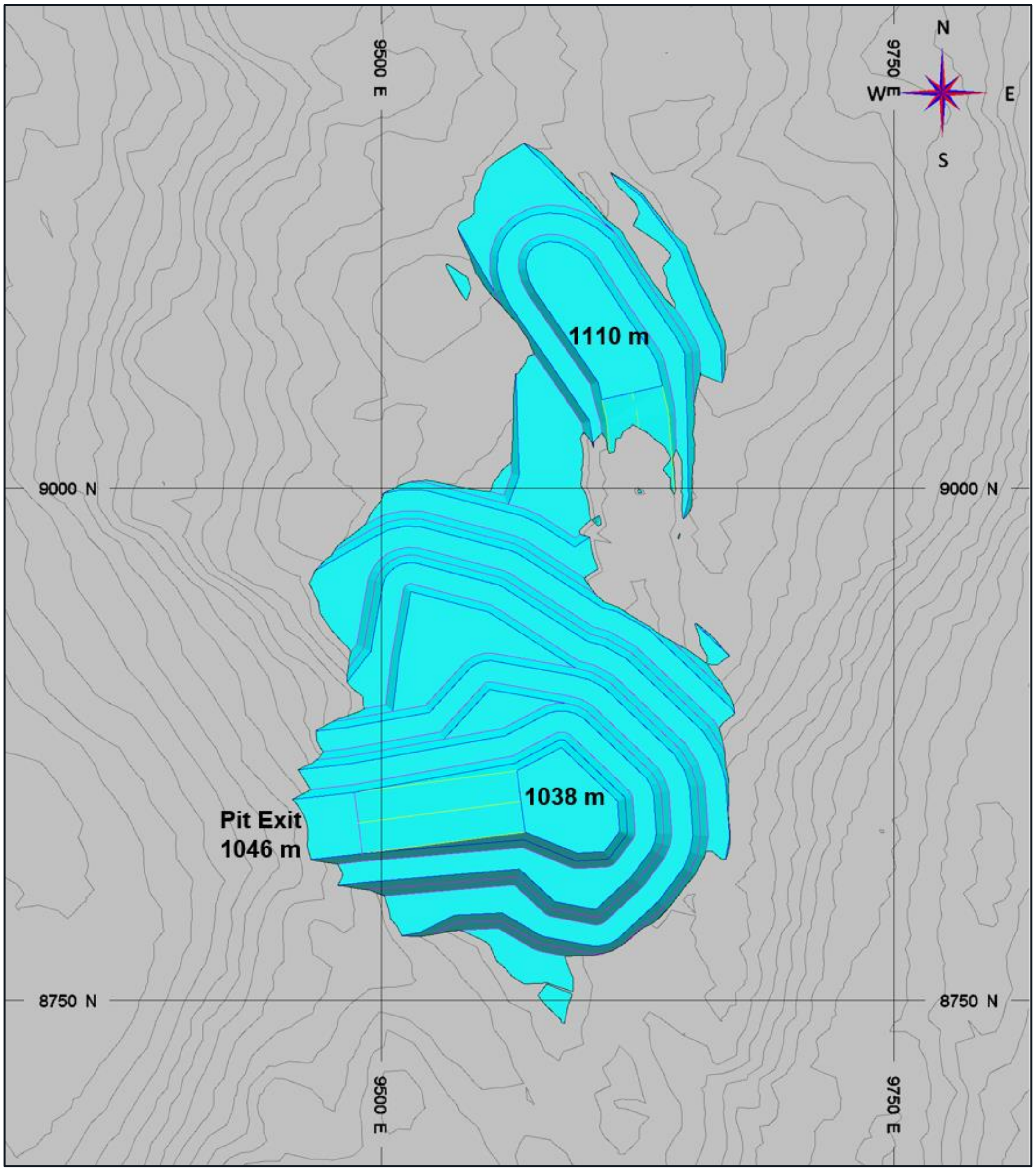


Note: Figure prepared by AGP, 2021.

16.8.6 South Phase 1

There will only be a single small phase in the south pit. Phase bench elevations will range from 1138 masl down to 1018 masl. This phase will be mined at the highest elevations of any other pit phase, and it is likely to be accessed from the top of the Waste Dump West (WDW) (largest NAG waste rock storage facility) near the end of mining. The south pit design is shown in Figure 16-28.

Figure 16-28: Proposed South Phase 1



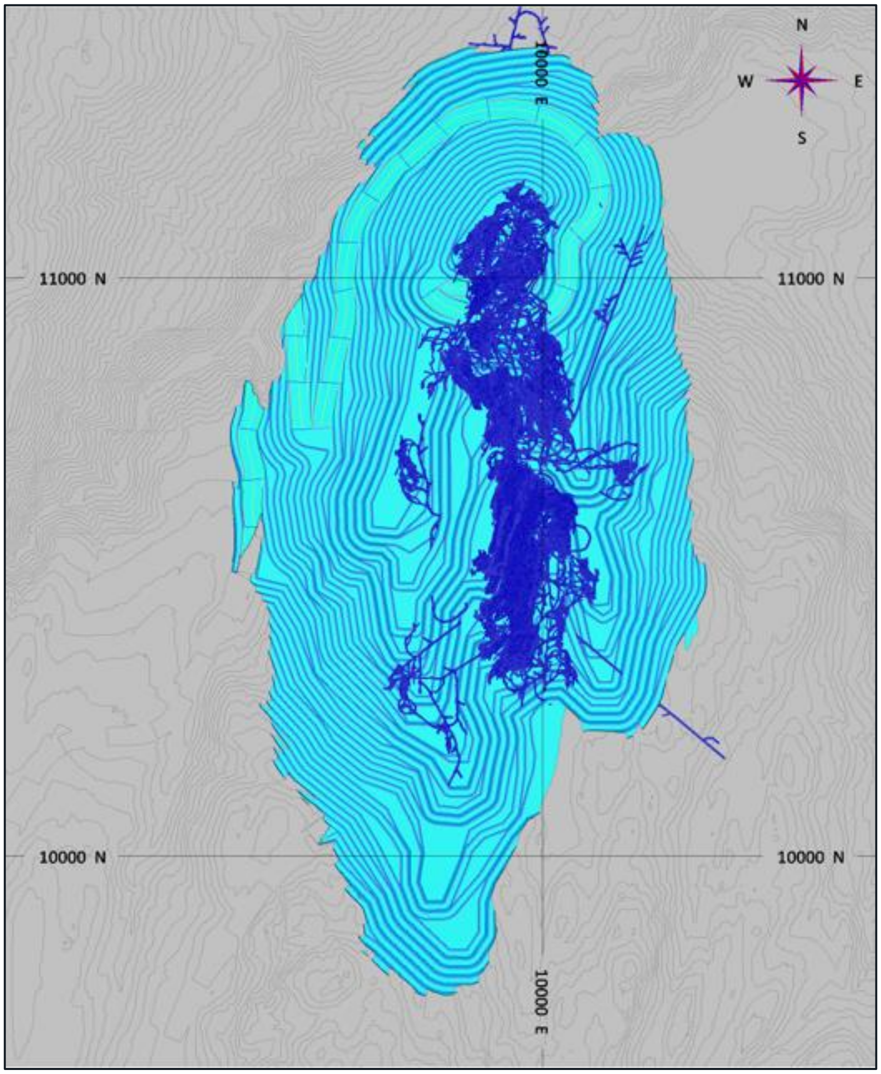
Note: Figure prepared by AGP, 2021.

16.9 Working Around Underground Voids

Best practice for advancing open pit mining operations through existing underground voids is to fill them with either waste or mill feed, which removes the void and supports the wall rock around the void.

Although working around known voids will present some safety and productivity challenges, the bigger concern is the unknown voids. It is anticipated that the RC grade control drilling program will provide additional information regarding the location of the voids in advance of mining equipment being present. Additional support hours have been included in the cost estimate to compensate for the extra time required working around and preparing the old mine workings. The expected issue will be drifts as opposed to stopes, as the stopes were backfilled with cemented material for stability. The location of the old workings is noted in the north pit and shown in dark blue in Figure 16-29.

Figure 16-29: Location of Historic Underground Workings



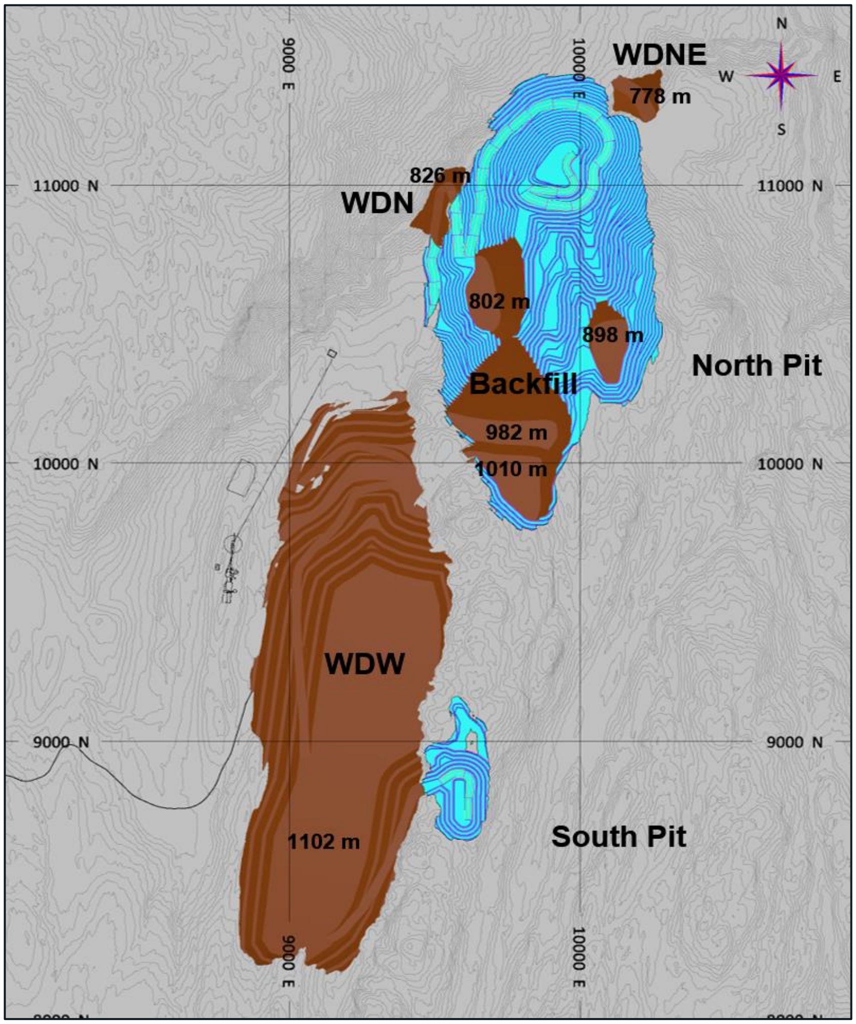
Note: Figure prepared by AGP, 2021.

16.10 Waste Rock Storage Facility Design

Various rock types are present in the material mined within the final pits. The key difference since the PEA study was the segregation of PAG and NAG waste rock. Based on recent test work, the only lithologies considered as NAG were hangingwall andesite and HW sediments. The remainder of the waste rock was considered PAG and will be sent to the Tom Mackay Lake storage facility to be submersed below water. NAG and PAG waste material contained in the ultimate pits are 161 Mt and 50 Mt, respectively. The total amount of waste within the mine plan is 212 Mt. This split in material will be determined by blast hole sampling and from the RC grade control drilling.

The largest NAG WRSF is labelled WDW. It is located to the immediate west of the north and south pits. WDN and WDNE (Waste Dump Northeast) are two small NAG WRSF's which are used to establish access to mining areas in phase 3. The remainder of the NAG waste will be placed into the mined-out north pit as backfill. These NAG waste storage area locations are displayed in Figure 16-30. The projected storage capacities are shown in Table 16-14.

Figure 16-30: Planned Waste Storage Areas



Note: Figure prepared by AGP, 2021.

Table 16-14: **WRSF Parameters**

Parameter	Units	WDW	WDN	WDNE	North Pit Backfill	TSF Embankment	TSF
Waste storage capacity	Mm3	63	0.6	0.5	12	1	24
Maximum elevation	masl	1102	845	778	1010	1114	1112

The WRSF design used a swell factor of 1.30. For the WDW facility, the lift height will be 20 m. Assuming a 37° face slope, the overall slope will be 26.5° with 13.6 m berm widths. A 37° face slope was also used for the in-pit backfill WRSFs.

The WRSFs will be actively reclaimed as they are developed. Dozers will re-slope as the facilities are advanced to allow revegetation to occur as soon as possible. Drainage ditches will need to be in place along the west side of the WDW facility, so water does not flow directly into Tom Mackay Creek.

The TSF embankment will be constructed with NAG waste, while all PAG material will be sent to the TSF and submersed below water. The intent is that PAG waste is dumped across the Tom Mackay facility as a causeway, followed by use of a dragline to retreat and transfer the portion of causeway material above water into the void between causeways and below the water level. This process is repeated over the extent of the storage facility so that a series of causeways are built and retreated in succession in a manner that minimizes exposure time of PAG waste rock to air. For extended dragline downtime periods, dozers will be used to push PAG material below the water surface from the causeways in controlled conditions.

Drainage from the WDW will be directed to the settling pond to the west of the pit and treated as required.

16.10.1 WDW Stability Analysis

Critical sections through the WDW were selected to develop the overall waste rock storage facility geometry, i.e., bench heights, bench widths, overall WDW slope and overall height. Stability of the WDW was assessed using the limit-equilibrium modelling software Slope/W, (Geostudio, 2018). Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than 1.5 FOS for static and 1.1 FOS for pseudostatic. The WDW stability analyses exceeded both static and pseudo-static under standard of practice guidelines.

16.10.2 WDW Water Management

A surface water management plan for the waste rock storage facility was developed based on the expansion of the facility over the life of mine. The plan includes managing both non-contact and contact water as the facility increase in size from 0 ha to 111 ha.

The waste rock storage facility is located west of the open pits in two parallel drainages that are oriented in a northeast direction. The eastern drainage includes Argillite Creek and the watershed that includes the WDW is approximately 240 ha. GoldSim computer model was utilized to develop the runoff from the WDW and the watershed above the WDW for average year, WDW (contact water) for 1:200 year event and 1:475 year event for non-contact water upstream of the WDW.

The water management plan includes a contact water pond located northwest of the WDW, contact water diversion channels, and non-contact water rock drain underneath the WDW. The contact water diversion channels discharge into the Contact Water Pond and the rock drain discharges into Tom MacKay Creek.

The contact surface water channels are design to convey up to the 1:200 year storm event to the contact water pond. These channels consist of both permanent and temporary channels. The permanent channels are located along the exterior of the WDW in zones that are final, and the temporary channels are located along benches that will be covered during development of the WDW. The channel consists of compacted subgrade, riprap, and grout in high velocity zones.

The contract water pond is constrained by the open pit, haul road and stockpile and has a design storage capacity of approximately 50k m^3. The pond is designed to remove any coarse grain suspended sediments transported from surface runoff before discharging into Tom MacKay Creek. The retention time, for average annual flow, is approximately 5 days. In addition, a turbidity fence, near the outlet, will be installed to improve the management of finer suspended sediments.

For non-contact water and seepage through the WDW, a flow-through rock drain has been designed to pass underneath the WDW. In the past few decades, the practice of conveying surface flow has gained acceptance for flows as high as 30 m/s. The peak flow from the 1:475 year event is approximately 4.5 m^3/s.

The rock drain has been designed with a factor of safety of four for the predicted flow rate for the 1:200 year event. The rock drain considers the removal of material less than 300 mm with a geotextile wrap and a transition zone above the drain to prevent fines migration into the drain. In addition, an inlet sacrificial rock drain zone will be installed to capture coarse grain material, organic material, etc. This zone will be replaced as needed to maintain flow into the drain. Snow will be removed around the inlet to maintain good flow at the entrance into the rock drain. At the toe of the WDW, the rock drain will transition into two (2) HDPE solid wall pipes to convey flow to Tom MacKay Creek due to mine facilities constraints and the diversion of Tom MacKay Creek in year 5 through a tunnel.

16.11 Mine Schedule

The mine schedule plans to deliver 26.4 Mt of mill feed grading 3.37 g/t gold and 94.4 g/t silver over a mine life of ten years. Waste tonnage totalling 212 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is 8.0:1. The detailed planned mine schedule is shown in Table 16-15 and Table 16-16, as well as by phase in Table 16-17 and Figure 16-31. Figure 16-32 and Figure 16-33 show the variation of the proposed mill feed over the life of mine by mill feed type, grade, and contained ounces.

The mine schedule assumes a maximum of 2.9 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. A maximum descent rate of eight benches per year per phase was applied to account for grade control, snow removal and filling of the previous underground workings.

The current mine life includes three years of pre-stripping and ten years of mining. Mill feed is stockpiled during the pre-production years. Three stockpiles were used for this schedule where:

- LG = material between marginal NSR cut-off and 1.5 g/t Au
- MG = material between 1.5 and 3.0 g/t Au, and
- HG = material above 3 g/t Au

A total stockpile capacity of approximately 1.5 Mt was used due to limited storage space. If space is found to be too restrictive during operations, LG stockpiles may need to be placed on selected benches of the waste facilities. The stockpiled mill feed, together with pit phasing, will be used to ensure mill feed is available during periods of poor weather. High precipitation will also necessitate in-pit sumps and surface ditches around the pits.

When mining starts, various infrastructure items will require development and construction activities. Significant activities near the pit will include construction of the process plant, crusher, conveyor between plant and crusher, TSF embankments and establishing proper roads to the mill feed crusher and waste destinations. Operationally, ditching and drains will need to be established near roads and infrastructure facilities.

The TSF embankments were scheduled to be constructed in Years -1 to Year 3. A total of 966,000 cubic metres were scheduled to be sourced from the mining areas.

Year -3 has the technical sample being mined as well as the upper benches of Phase 1 of the north pit. In this period, a total of 1.9 Mt of waste material will be moved as the project ramps up. The final bench elevations for the technical sample and phase 1 in Year -3 were 994 m and 1042 m, respectively.

Years -2 and -1 mining bring an additional total material movement of 1.4 Mt and 9.0 Mt moved, respectively. At the end of pre-production, the crusher stockpile will contain 560 kt of mill feed material grading 1.75 g/t Au and 73 g/t Ag in anticipation of plant commissioning and operation. Phases 1 and 2 end up on 994 m bench while the WDW waste facility reaches the 982 m lift.

Table 16-15: PFS Mine Schedule (stockpiles and material movement)

Description		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
Mining summary	NAG Waste (Mt)	1.5	1.0	7.0	13.0	24.6	25.1	21.7	18.3	20.1	15.9	8.5	4.5	0.0	161
	PAG Waste (Mt)	0.4	0.4	1.5	3.3	4.3	3.6	4.5	6.2	5.3	6.6	5.6	6.4	2.3	50
	Mined Waste (Mt)	1.9	1.4	8.5	16.3	28.8	28.7	26.2	24.5	25.4	22.5	14.1	10.9	2.3	212
	Mined Ore (Mt)	0.0	0.0	0.5	2.9	2.4	1.9	3.1	3.2	3.0	2.6	2.9	3.1	0.9	26.4
	Au (g/t)	4.80	1.24	1.60	4.33	3.07	5.29	4.03	4.00	2.67	2.73	3.06	2.60	1.36	3.37
	Ag (g/t)	55	136	72	78	100	159	156	115	54	66	51	101	53	94
	Sb (ppm)	431	162	300	2091	2477	2054	2034	1041	450	473	363	551	292	1185
	Hg (ppm)	66.70	8.40	26.38	196.01	80.03	121.60	70.74	29.67	14.13	14.35	10.75	12.93	7.76	56.07
	As (ppm)	4911	444	264	2572	493	667	632	640	276	394	698	516	452	758
	Pb (%)	0.04	0.04	0.13	0.10	0.20	0.35	0.66	0.51	0.15	0.69	0.71	0.90	0.29	0.47
	Zn (%)	0.08	0.08	0.21	0.18	0.37	0.60	1.03	0.82	0.27	1.07	1.07	1.34	0.43	0.74
	Fe (%)	1.92	1.46	1.41	1.62	1.75	1.59	1.30	1.34	1.96	1.51	0.87	0.35	1.02	1.33
	S (%)	1.37	0.86	1.26	1.66	1.23	1.06	0.89	0.96	1.48	1.15	0.60	0.20	0.74	1.01
	Mined Total (Mt)	2	1	9	19	31	31	29	28	28	25	17	14	3	238
Processed Material	Mill Feed (Mt)	0.0	0.0	0.0	2.0	2.9	2.9	2.9	2.7	2.7	2.7	2.7	2.7	2.2	26.4
	Au (g/t)	0.00	0.00	0.00	5.00	3.46	3.83	4.23	4.51	2.83	2.64	3.17	2.83	1.14	3.37
	Ag (g/t)	0	0	0	97	94	119	165	132	58	65	53	114	29	94
	Sb (ppm)	0	0	0	2484	2330	1534	2130	1155	471	483	373	585	296	1185
	Hg (ppm)	0.00	0.00	0.00	219.45	104.39	93.13	74.00	32.23	14.82	14.78	10.93	12.83	10.20	56.07
	As (ppm)	0	0	0	2657	1148	542	650	677	281	398	716	491	445	758
	Pb (%)	0.00	0.00	0.00	0.09	0.19	0.27	0.67	0.56	0.16	0.66	0.72	0.93	0.38	0.47
	Zn (%)	0.00	0.00	0.00	0.17	0.34	0.47	1.04	0.90	0.28	1.03	1.08	1.37	0.58	0.74
	Cu (%)	0.00	0.00	0.00	0.02	0.04	0.06	0.11	0.10	0.04	0.10	0.07	0.11	0.03	0.06972
	Fe (%)	0.00	0.00	0.00	1.67	1.67	1.59	1.30	1.32	1.89	1.53	0.86	0.36	1.16	1.33
	S (%)	0.00	0.00	0.00	1.69	1.34	1.12	0.90	0.96	1.47	1.14	0.59	0.20	0.79	1.01

Table 16-16: PFS Mine Schedule (stockpiles and material movement)

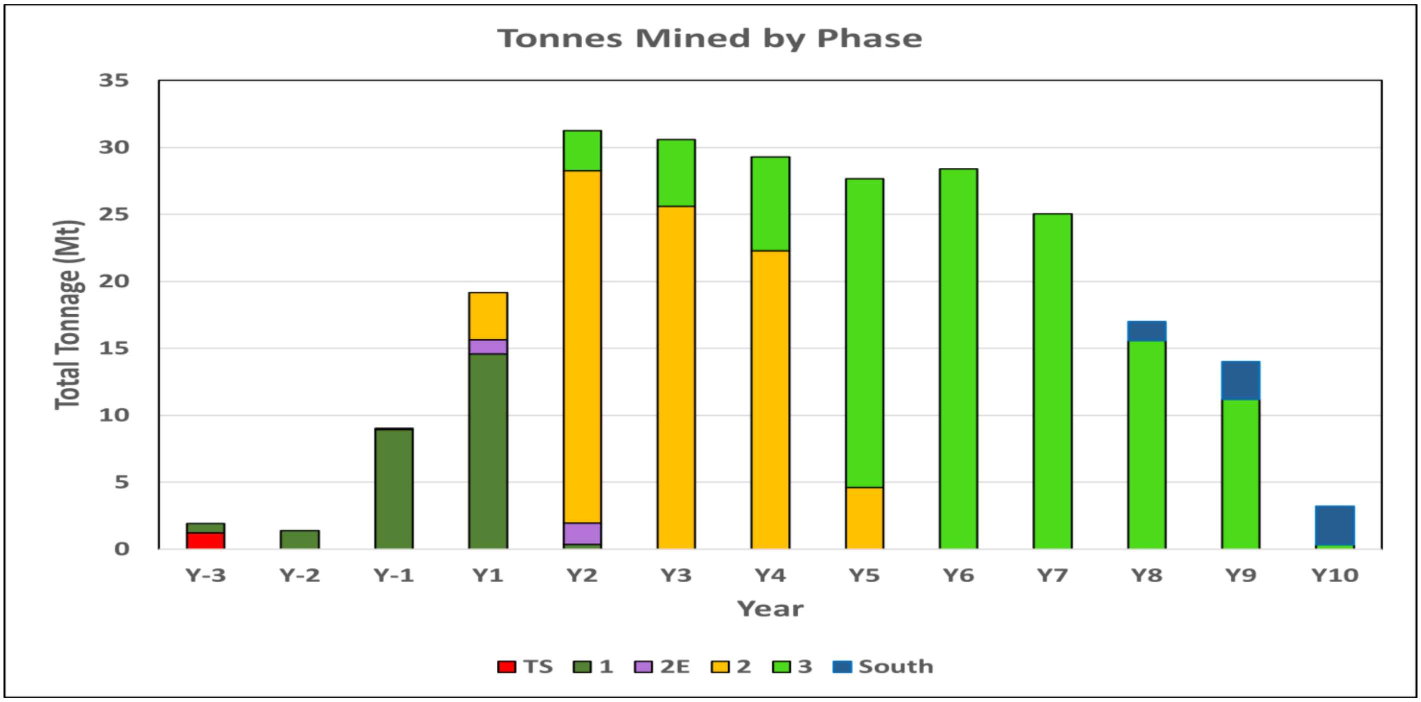
Description		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
Stockpile Balance	LG (kt), 0-1.5 g/t Au	7	14	360	1124	978	2	185	642	903	787	940	1333	0	
	Au (g/t)	1.04	0.90	0.94	1.00	0.95	0.58	0.81	0.94	0.98	1.00	1.02	1.00	0.81	
	Ag (g/t)	64.4	67.8	56.7	43.9	41.1	12.8	19.1	19.6	17.4	14.8	13.5	13.6	9.1	
	Sb (ppm)	170	139	215	247	507	91	500	413	361	312	291	298	100	
	Hg (ppm)	22	13	24	43	37	8	19	16	13	12	11	12	72	
	As (ppm)	1,025	798	217	246	297	243	336	388	349	328	339	442	670	
	Pb (%)	0.076	0.054	0.128	0.145	0.120	0.121	0.555	0.314	0.247	0.262	0.323	0.448	0.031	
	Zn (%)	0.094	0.072	0.203	0.229	0.202	0.230	0.802	0.486	0.389	0.405	0.491	0.689	0.060	
	Cu (%)	0.010	0.006	0.017	0.018	0.017	0.010	0.048	0.039	0.033	0.032	0.031	0.035	0.005	
	Fe (%)	1.808	1.630	1.421	1.405	1.604	1.270	1.368	1.402	1.792	1.764	1.642	1.250	1.868	
	S (%)	0.767	0.726	1.206	1.330	1.240	0.720	0.886	0.951	1.134	1.152	1.099	0.828	1.749	
	MG (kt), 1.5-3 g/t Au	6	9	126	116	1	0	0	7	1	2	2	2	0	
	Au (g/t)	2.35	2.22	2.08	2.17	1.81	0.00	0.00	1.64	1.57	1.59	1.59	1.59	0.00	
	Ag (g/t)	14.6	110.8	96.7	43.4	20.4	0.0	0.0	24.9	1.9	3.1	3.1	3.1	0.0	
	Sb (ppm)	256	262	388	967	1,045	0	0	593	599	417	417	417	0	
	Hg (ppm)	38	29	28	173	8	0	0	27	7	6	6	6	0	
	As (ppm)	4,100	2,774	506	717	401	0	0	1,342	314	302	302	302	0	
	Pb (%)	0.017	0.027	0.117	0.054	0.089	0.000	0.000	0.230	0.256	0.260	0.260	0.260	0.000	
	Zn (%)	0.043	0.057	0.212	0.119	0.144	0.000	0.000	0.331	0.400	0.409	0.409	0.409	0.000	
	Cu (%)	0.008	0.008	0.022	0.015	0.015	0.000	0.000	0.038	0.050	0.051	0.051	0.051	0.000	
	Fe (%)	1.930	1.755	1.414	1.651	3.065	0.000	0.000	3.569	2.134	2.091	2.091	2.091	0.000	
	S (%)	1.346	1.284	1.284	1.784	2.486	0.000	0.000	2.997	2.334	2.204	2.204	2.204	0.000	
	HG (kt), > 3 g/t Au	15	16	74	223	0	0	0	0	0	0	0	0	0	
	Au (g/t)	7.33	7.20	5.14	9.81	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Ag (g/t)	66.5	68.9	109.4	83.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
	Sb (ppm)	609	596	589	3,958	0	0	0	0	0	0	0	0	0	
	Hg (ppm)	96	94	45	348	0	0	0	0	0	0	0	0	0	

Description		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Total
	As (ppm)	6,870	6,623	1,839	9,303	0	0	0	0	0	0	0	0	0	
	Pb (%)	0.037	0.039	0.095	0.046	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
	Zn (%)	0.085	0.094	0.188	0.103	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
	Cu (%)	0.025	0.024	0.023	0.014	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
	Fe (%)	1.960	1.947	1.527	1.769	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
	S (%)	1.629	1.623	1.442	2.007	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
Total Stockpile Reclaim	(Mt)	0.0	0.0	0.0	0.8	1.1	1.0	0.0	0.0	0.0	0.4	0.0	0.0	1.3	4.7
Total Material Movement	(Mt)	1.9	1.4	9.0	20.0	32.4	31.6	29.3	27.7	28.4	25.4	17.0	14.0	4.6	242.7

Table 16-17: Tonnes Mined by Phase

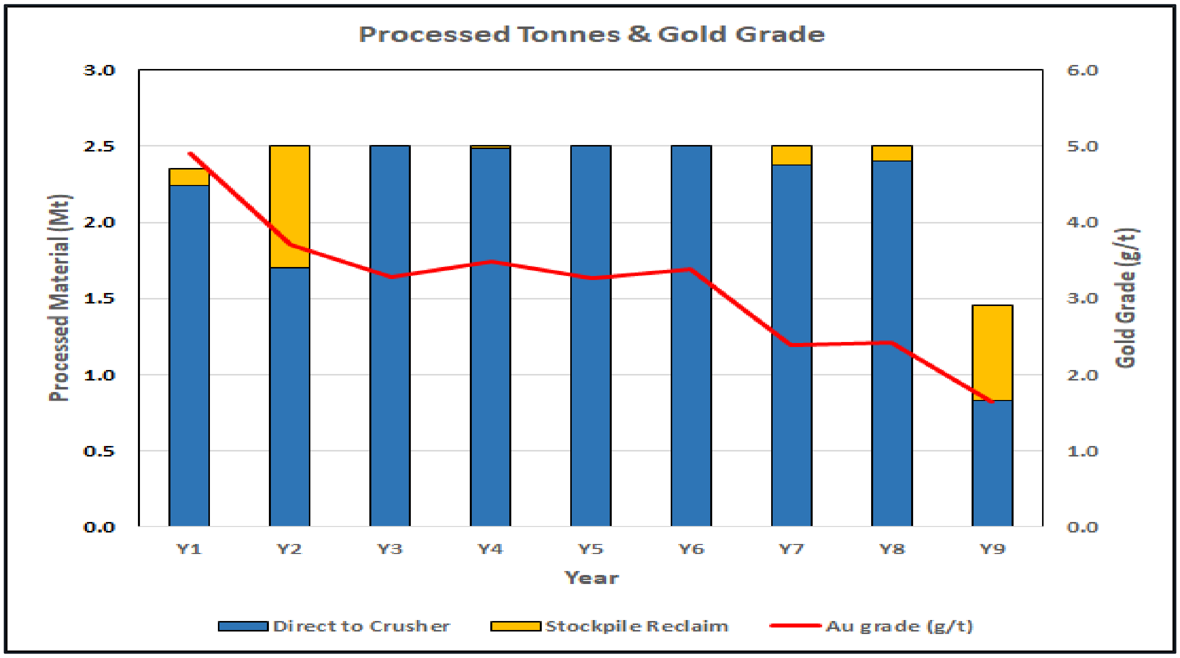
Phase	Total Tonnage (Mt)													Total (Mt)
	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	
TS	1.2													1.2
1	0.7	1.4	8.9	14.6	0.3									26.0
2E				1.1	1.6									2.7
2			0.1	3.5	26.3	25.6	22.3	4.6						82.4
3					3.0	5.0	7.0	23.1	28.4	25.0	15.6	11.2	0.3	118.6
South											1.4	2.8	2.9	7.1
Total	1.9	1.4	9.0	19.2	31.3	30.6	29.3	27.7	28.4	25.0	17.0	14.0	3.2	238.0

Figure 16-31: Tonnes Mined by Phase



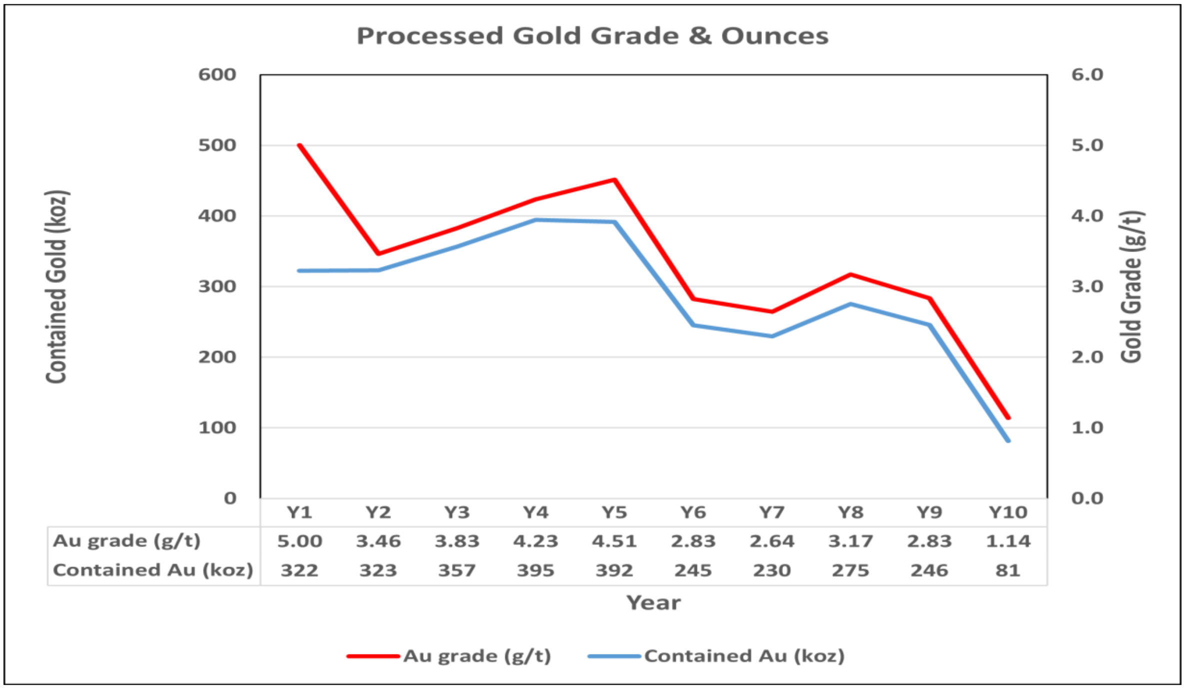
Note: Figure prepared by AGP, 2021.

Figure 16-32: Planned Life of Mine Mill Feed Tonnes and Ounces



Note: Figure prepared by AGP, 2021.

Figure 16-33: Process Grade and Contained Ounces of Gold



Note: Figure prepared by AGP, 2021.

Table 16-18 displays a summary of the resource classifications for the mill feed.

Year 1 production assumes the plant will require twelve months to achieve a full production rate of 2.9 Mtpa. The first six months were assumed to steadily increase up to 80% of full production capacity, while the remainder of the first year steadily increases up to 100% of production capacity. Subsequent months will be at 100% of nameplate capacity in the mill. This plant ramp-up schedule results in the full Year 1 production of 2.0 Mt. Mill feed will be from stockpile or direct feed from Phase 1 and Phase 2E. This period has the initial mining in Phase 2E on the east side of the pit, with Phases 1 and 2 continuing to be active. All NAG waste will be directed to the WDW facility and the TSF embankments.

Year 2 production will be at the full 2.9 Mt of mill feed. Phases 1 and 2E mining will be completed to final levels of 930 masl and 954 masl respectively. Phases 2 continues mining while phase 3 is started. Phases 2 and 3 will be mined down to the levels of 898 masl, and 962 masl respectively. All NAG waste will be directed to the WDW facility and the TSF embankments.

Year 3 production will see Phase 2 as the dominant phase of mining in this period, driving to a depth of 850 masl. Phases 3 is the only other active phase and advances down to level 930masl. All NAG waste will be directed to the WDW facility and the TSF embankments.

Year 4 production will again have Phases 2 and 3 as the only active phases with their final levels being 786 masl and 914 masl respectively. This is the final year where the plant operates at 2.9 Mtpa due changing ore properties and increased hardness in the remaining years. From this period forward, no further NAG waste material is required to be sent to the TSF embankments. A small portion of NAG waste material will be directed back into the pit as backfill as space allows on the east side of 898 m bench. The ultimate pit limit has been reached in this area of the pit. The remainder of the NAG waste will be sent to the WDW facility up to 1062 masl lift.

Year 5 will see the plant production lowered to 2.7 Mtpa and remain at this rate for the remainder of the mine life. Phases 2 and 3 are the only active mining phases and will be advanced down to 762 masl and 850 masl levels respectively. Phase 2 mining is completed in this period. Most of the NAG waste will be directed to the WDW facility up to 1082 masl elevation, but approximately 13% of the NAG waste will be directed to a backfill dump at the 1010 m elevation.

In advance of mine production in Year 6, a water diversion tunnel will need to be excavated so that water can be diverted around the ultimate pit. Detailed designs have not yet been developed; however, a selected alignment is displayed in Figure 16-47 which shows the tunnel upstream of the WDN facility and downstream of the WDNE facility along Tom Mackay Creek. Additional geotechnical information is planned to be obtained during the 2021 field season so that a design basis can be better established. During operation Tom Mackay creek will be diverted through this tunnel, completely avoiding contact with pit activities.

Year 6 will have Phase 3 as the only active phase, and it will be advanced down to 794 masl. NAG waste will be sent to the WDW facility up to the final elevation of 1102 masl and to the backfill at 1010 masl elevation. NAG waste will also be sent to a road switchback which extends across Tom Mackay Creek and is referred to as the WDN facility. This switchback will be used as part of the final road to the bottom of Phase 3. A NAG waste dump facility called WDNE will also be created across Tom Mackay Creek to the northeast of the ultimate pit and will be used for accessing upper benches on the north side of the creek.

Year 7 will have Phase 3 continuing to be mined and advanced down to 754 masl. NAG waste material will continue to extend the backfill dump at 1022 masl, with the remainder being directed to the WDW facility.

Year 8 will have mining continue in Phase 3 and initiate in the south pit phase. Phase 3 will be advanced down to 722 masl while the south pit will be advanced down to 1106 masl. NAG waste material will be directed to a new backfill elevation at 982 masl as well as the WDW facility.

Year 9 will have mining continue in Phase 3 and the South Pit Phase down to 658 masl and 1066 masl, respectively. NAG waste material will be directed to a new backfill elevation at 802 masl as well as the WDW facility.

Year 10 will be the final mining period with mining being completed in both Phase 3 and the South Pit. Phase 3 will have mining completed on the 650 masl level while the South Pit will be advanced down to 1018 masl.

The mine schedule was completed monthly for the years -3 to year 1, quarterly for years 2 and 3, and annually for the remainder of the schedule. The mine is scheduled to deliver 26.4 Mt of mill feed grading 3.37 g/t Au and 94 g/t Ag. NAG waste totalling 161 Mt will be stored in the WDW, WDN, and WDNE waste facilities external to the ultimate pit, as well as back into the mined-out pit areas as backfill. PAG waste totalling 50 Mt will also be directed to the tailings storage facility at Tom Mackay Lake and submersed below water. The overall strip ratio is 8.0:1.

Table 16-18: Resource Summary of Scheduled Material

Resource Class	Mill Feed	Grade		Contained Ounces	
Measured	13.5	4.25	124	1.85	53.7
Indicated	12.9	2.46	64	1.02	26.5
Total	**26.4**	**3.37**	**94**	**2.87**	**80.2**

16.12 Mine Plan Sequence

Anticipated end-of-year positions for the open pits are shown in Figure 16-34 to Figure 16-46.

Mining will be initiated in the north pit and will continue throughout the schedule, while the south pit will only be active in years 8 to 10.

16.13 Mining Equipment Selection

The mining equipment selected to meet the required production schedule is conventional mining equipment, with additional support equipment for snow removal and surface ditching maintenance.

Drilling will be completed with down the hole hammer (DTH) drills with140 and 229 mm bits. This will provide the capability to drill patterns for either 4 m or 8 m bench heights. The smaller drill will be the primary drill in the preproduction period and as larger productive benches developed relegated to pre-shear, drain holes and back up drilling duties.

Preproduction mining will be completed with 11.5 m^3 loaders and 91 t rigid body trucks. This smaller fleet is better suited to the lower production tonnage requirements and narrower working conditions. With full production starting in Year 1, the primary loading units will be 22 m^3 hydraulic shovels. Additional loading will be completed by a 23 m^3 loader. The smaller loaders will shift to working at the primary crusher and site maintenance roles (snow removal, etc.). It is expected that one of the 11.5 m^3 loaders will be at the primary crusher full time. The main production haulage trucks will be conventional 144 t rigid body trucks from Year 1 onwards.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management. Snowplows and additional graders were included in the fleet. In addition, smaller road maintenance equipment is included to keep drainage ditches open and sedimentation ponds functional.

Within the planned pit, an additional large backhoe will assist the mill feed preparation. It will be responsible for cleaning hanging wall and footwall material around the old, cemented stopes from the underground mining. While capable of loading the 144 t trucks if required, it is not scheduled to do so because of the extended loading time necessary. The backhoe/truck combination is not as efficient as the proposed primary loading units.

The proposed equipment requirements for the LOMP are included in Section 21.

16.14 Grade Control

Grade control will be completed with a separate fleet of RC drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60º.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m, with sampling over 6 m. These drill holes will be used to find undiscovered veinlets or pockets of mineralization.

The grade control holes will serve two purposes:

- Definition of the mill feed grade and contacts;
- Location of previous underground infrastructure prior to blasthole rigs drilling.

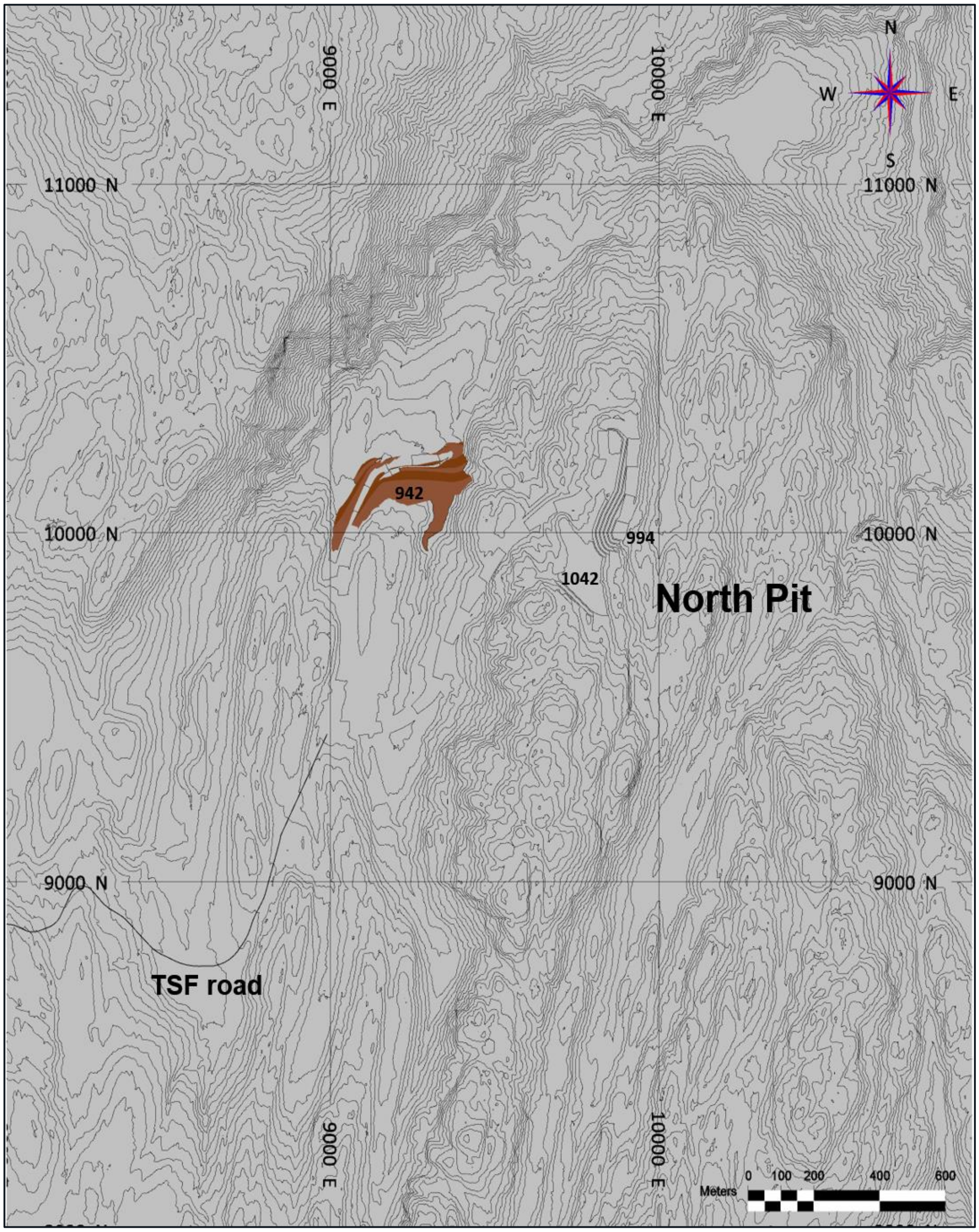
Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model. Blasthole sampling will also be part of the grade control program initially to determine the best method for Eskay Creek.

16.15 End of Period Plans

Images of the end of period positions in the pit of the waste dumps and pits are shown in Figure 16-34 to Figure 16-46.

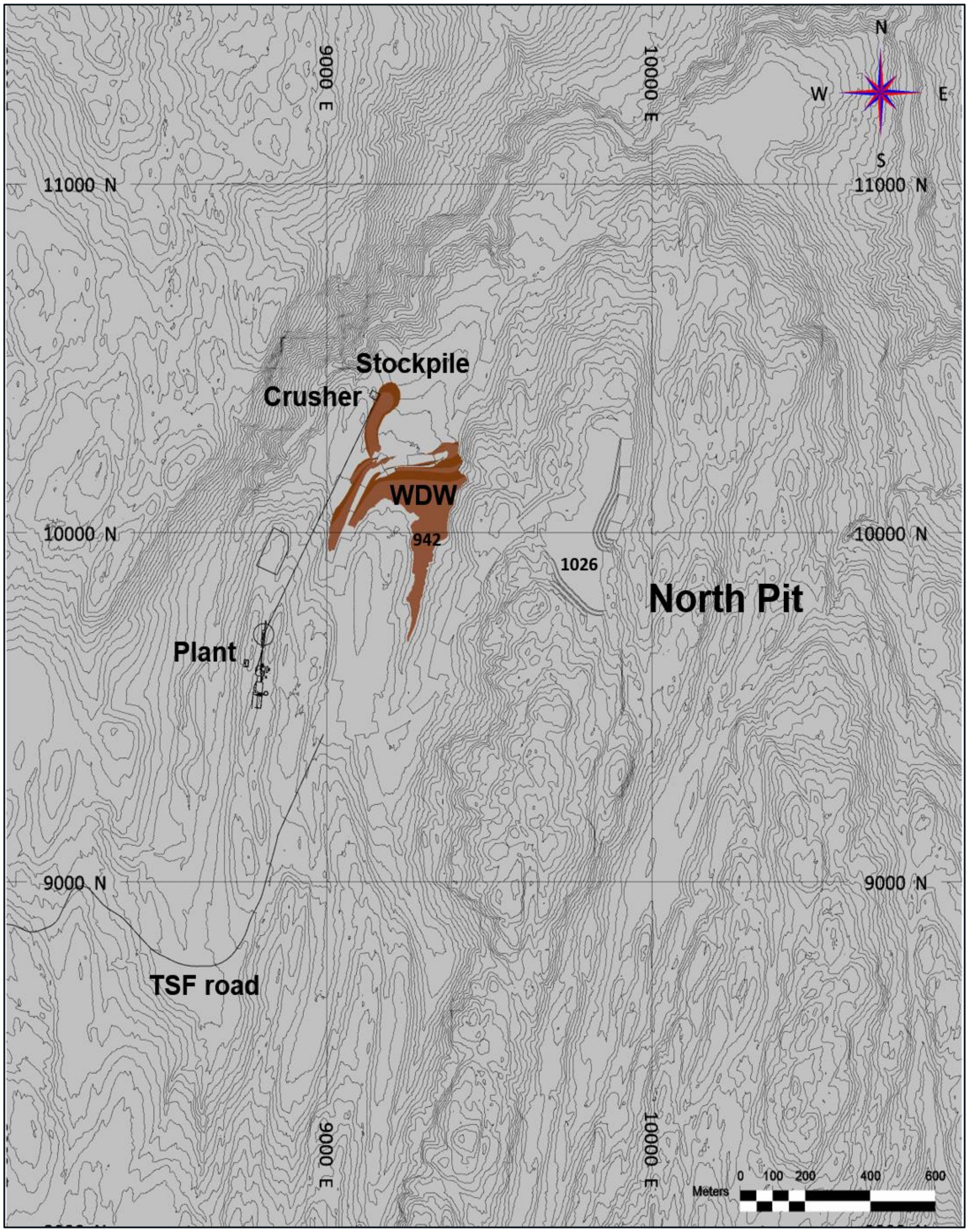
The location of the proposed Diversion Tunnel is shown in Figure 16-47.

Figure 16-34: End of Preproduction Period – Year-3



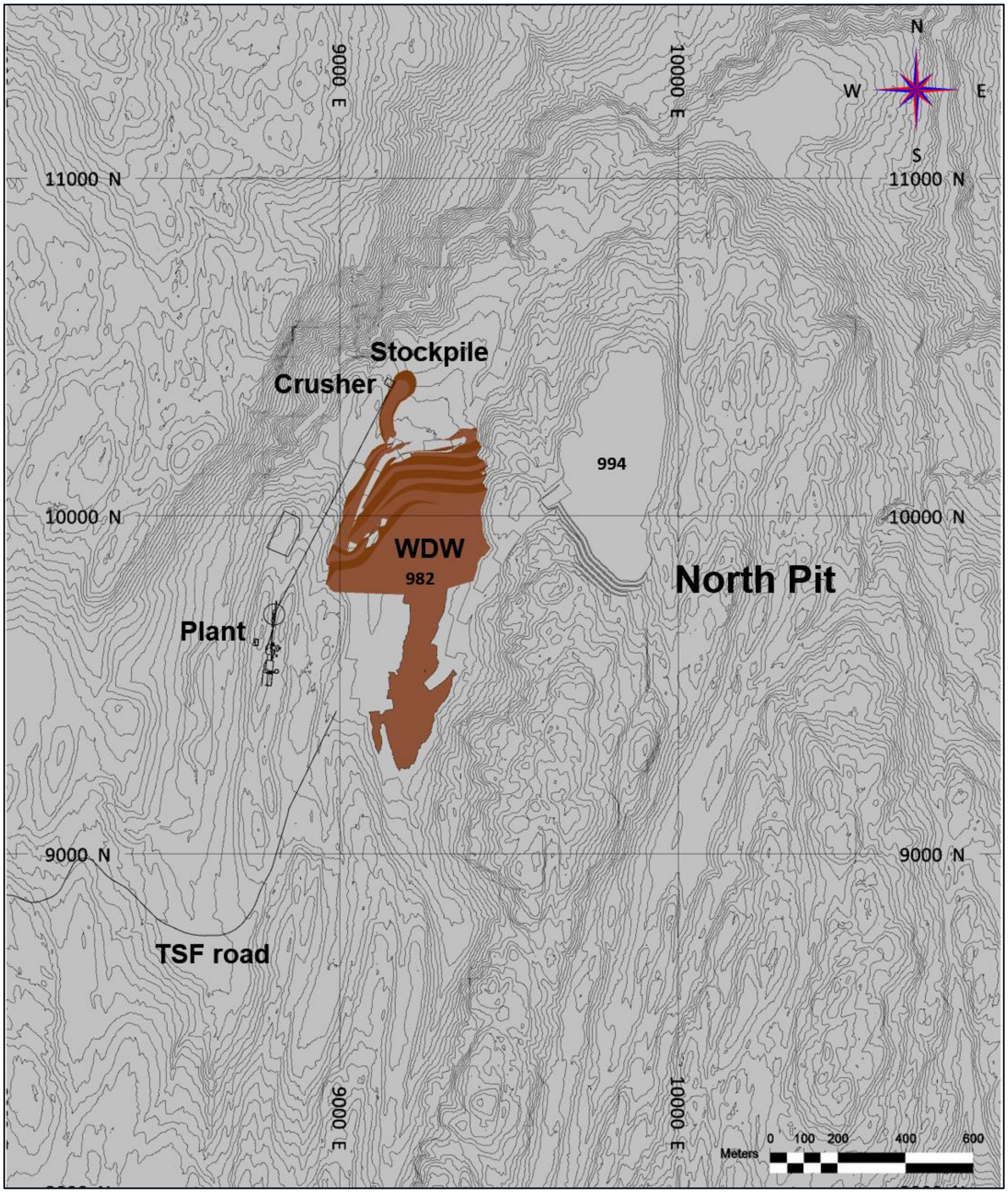
Note: Figure prepared by AGP, 2021.

Figure 16-35: End of Preproduction Period – Year-2



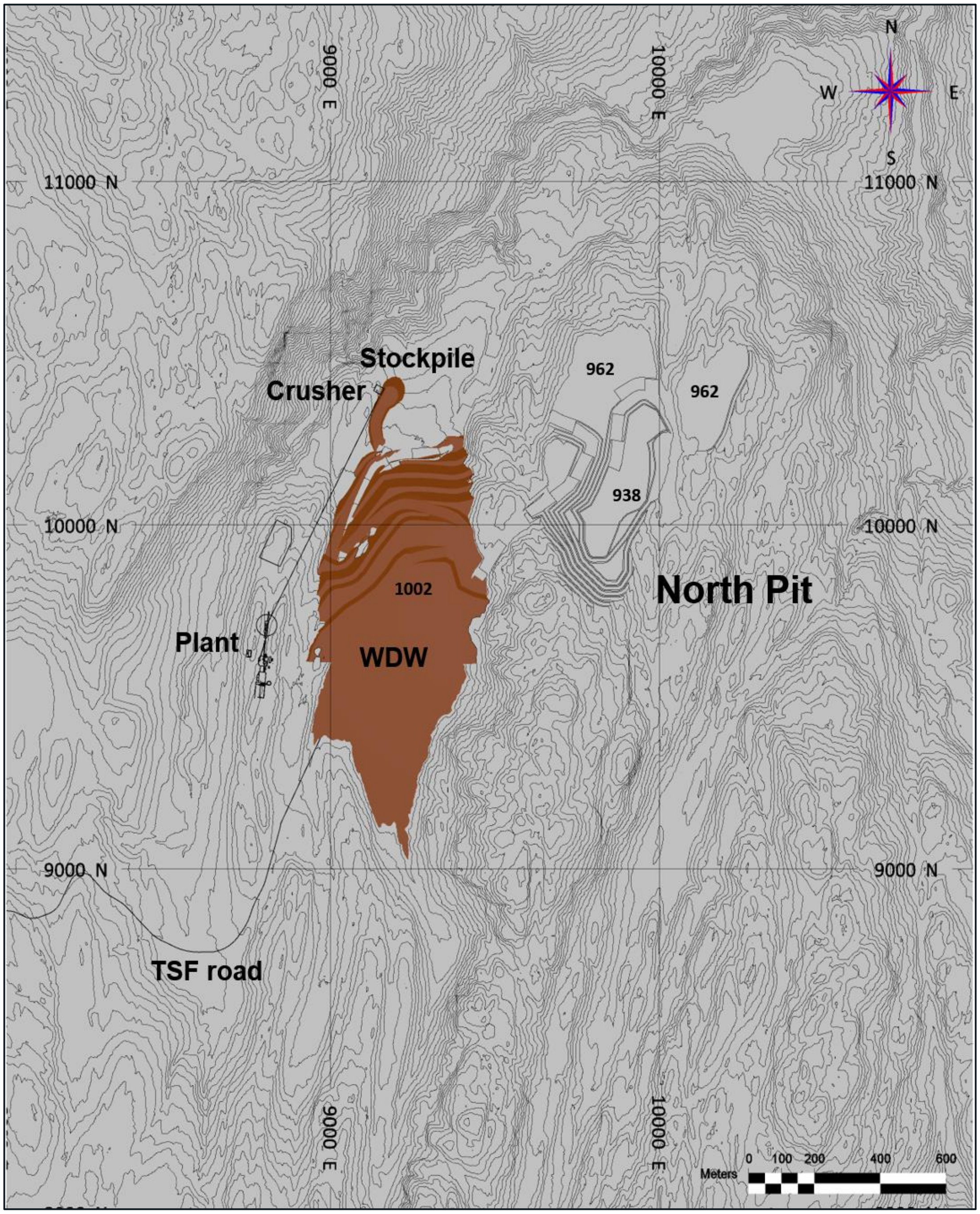
Note: Figure prepared by AGP, 2021.

Figure 16-36: End of Preproduction period – Year-1



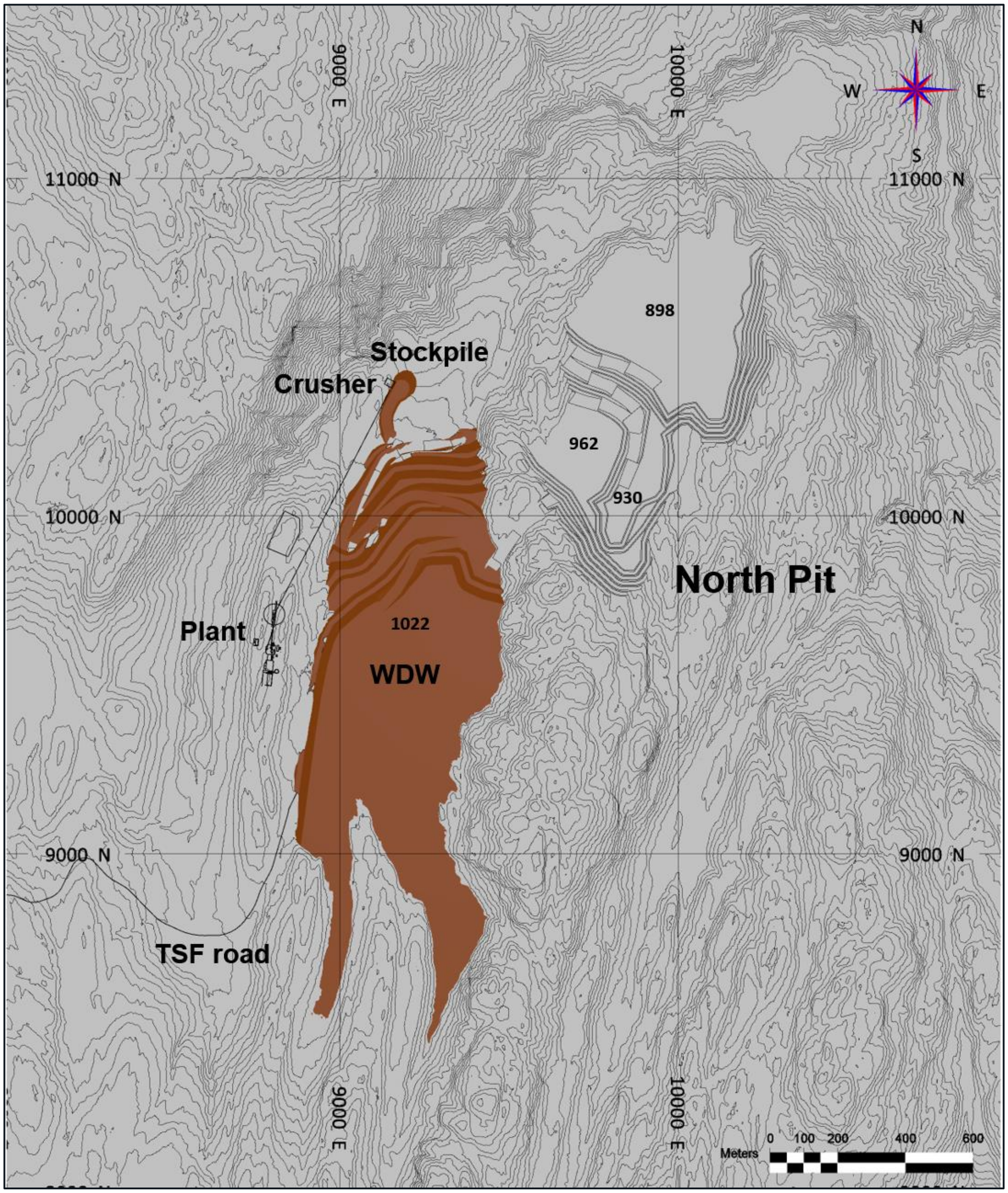
Note: Figure prepared by AGP, 2021.

Figure 16-37: End of Year 1



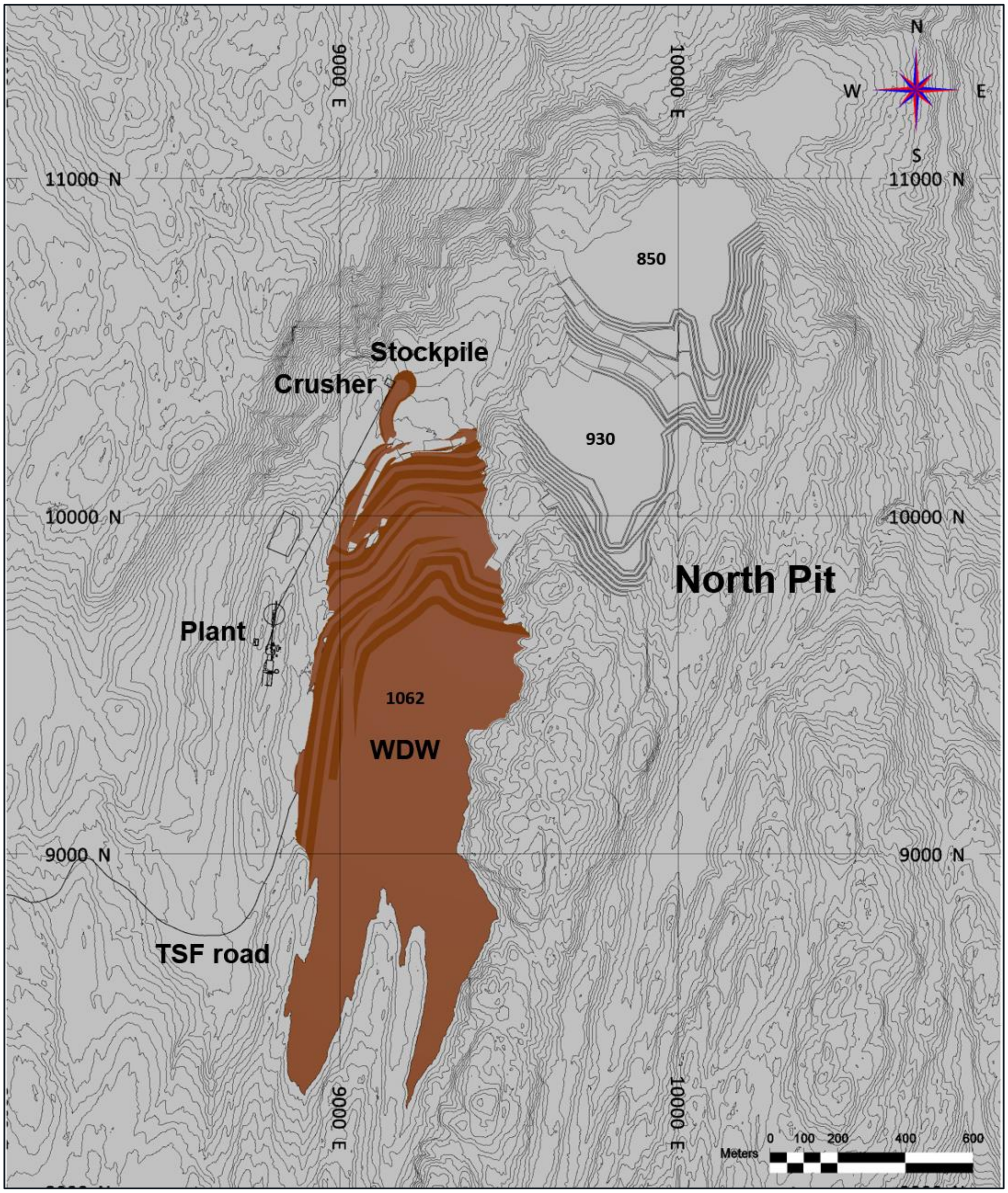
Note: Figure prepared by AGP, 2021.

Figure 16-38: **End of Year 2**



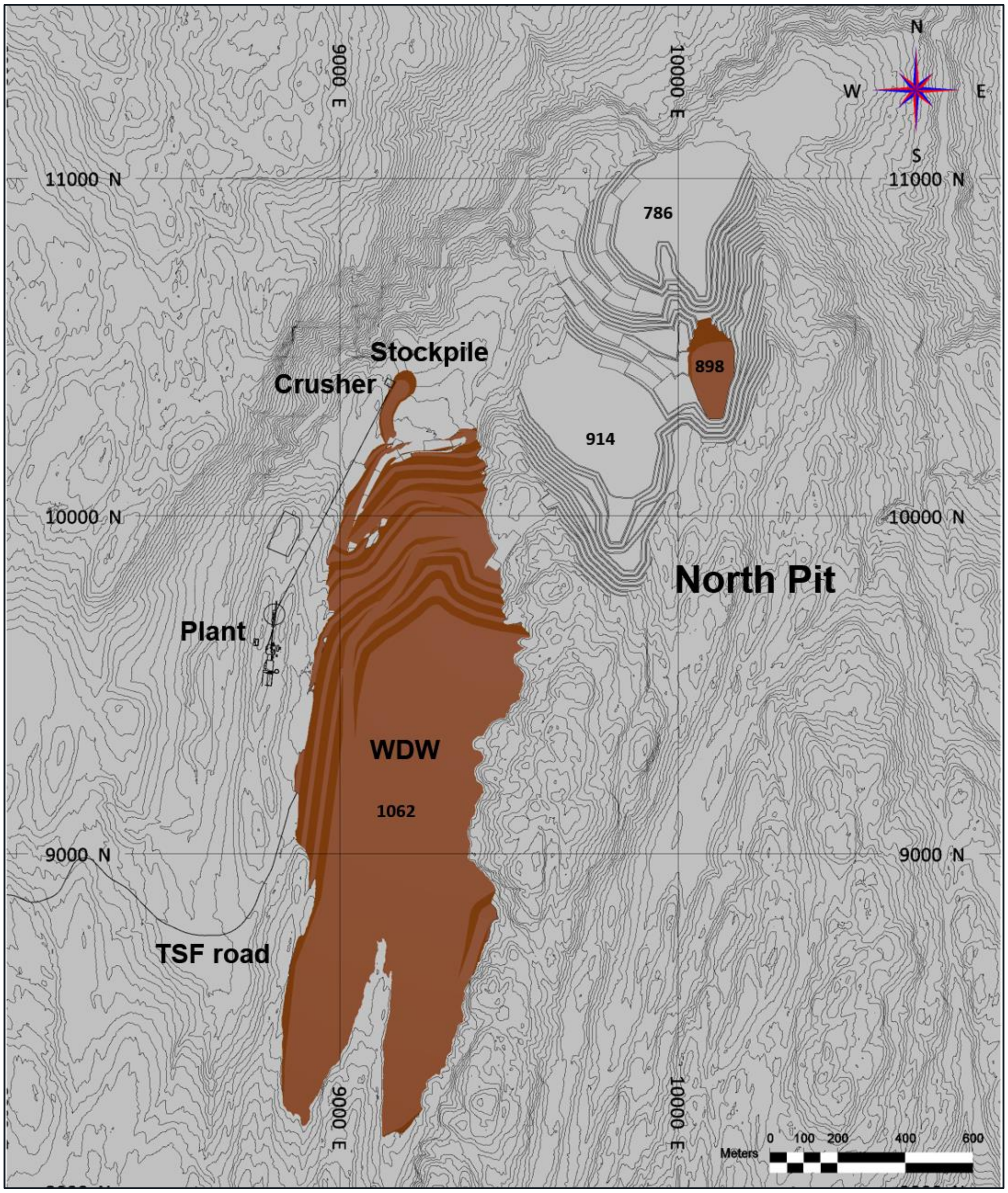
Note: Figure prepared by AGP, 2021.

Figure 16-39: End of Year 3



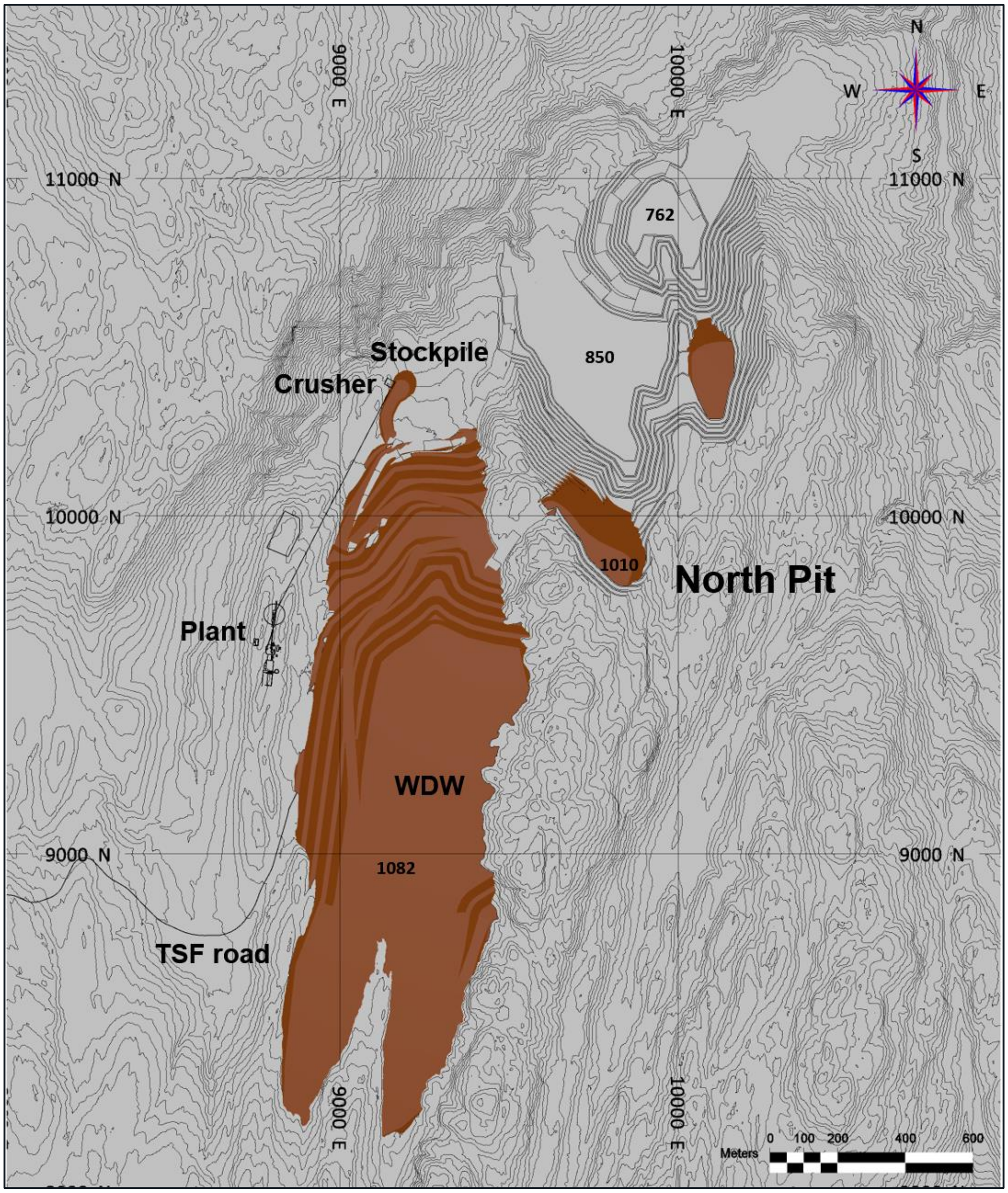
Note: Figure prepared by AGP, 2021.

Figure 16-40: End of Year 4



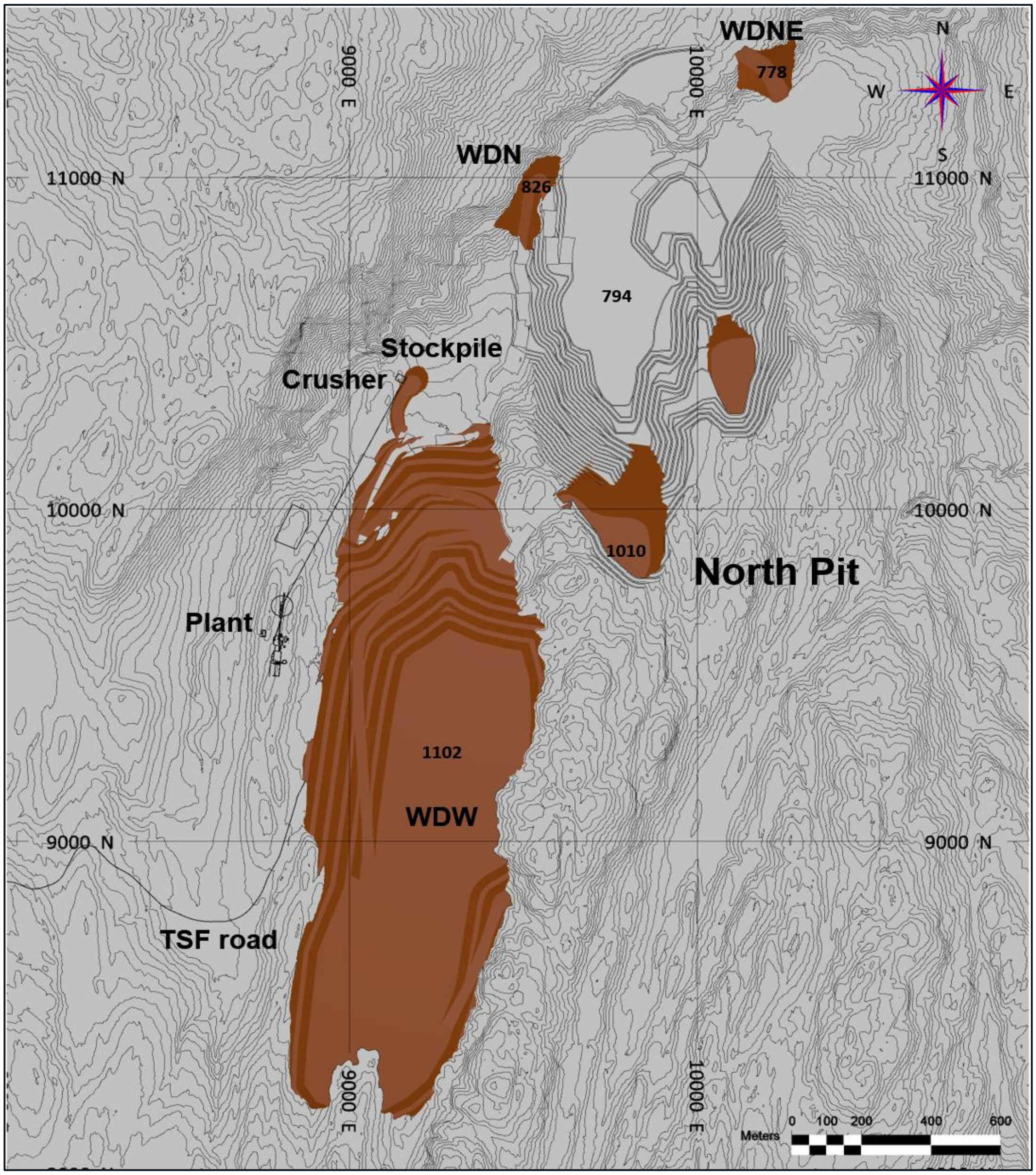
Note: Figure prepared by AGP, 2021.

Figure 16-41: End of Year 5



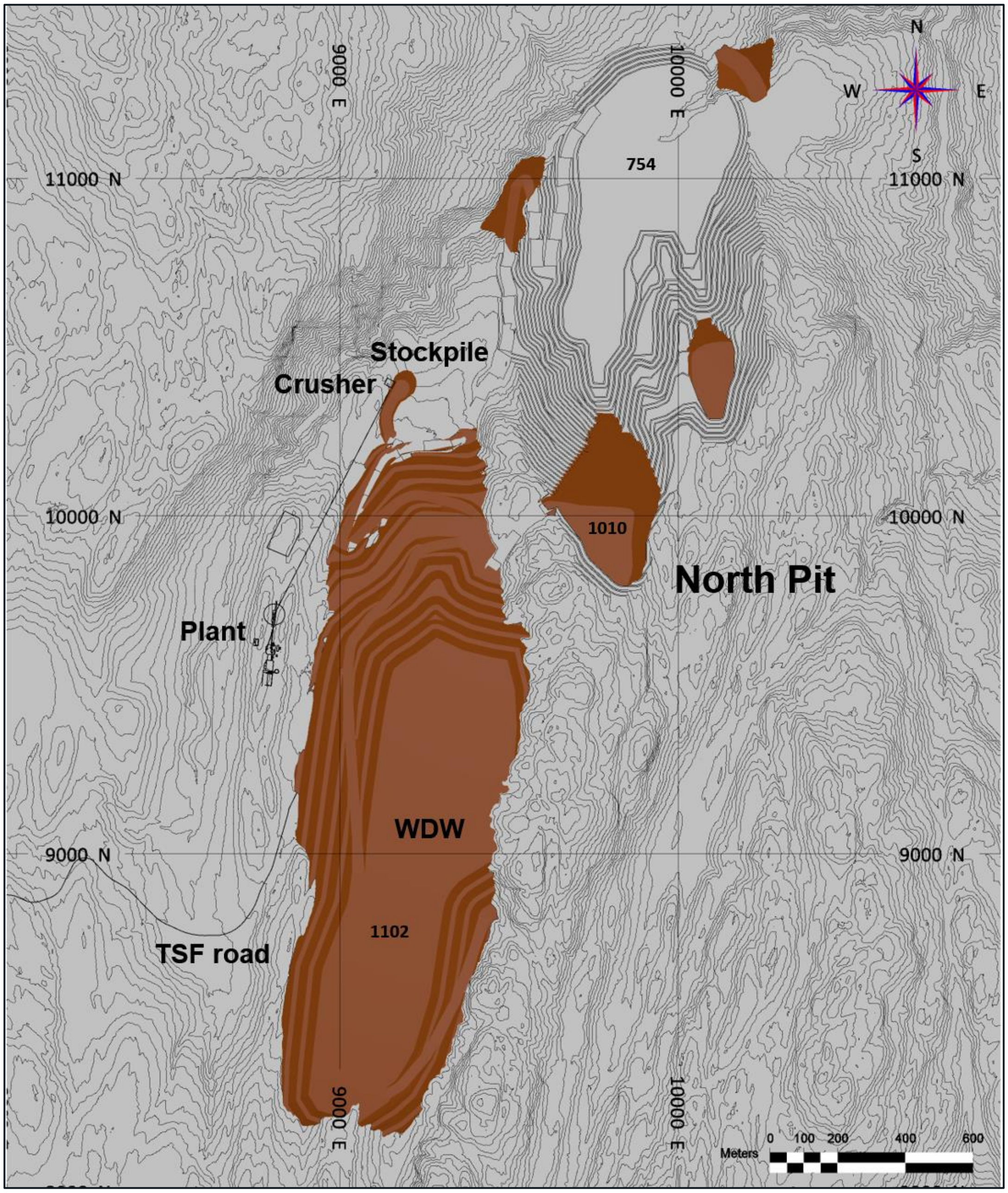
Note: Figure prepared by AGP, 2021.

Figure 16-42: End of Year 6



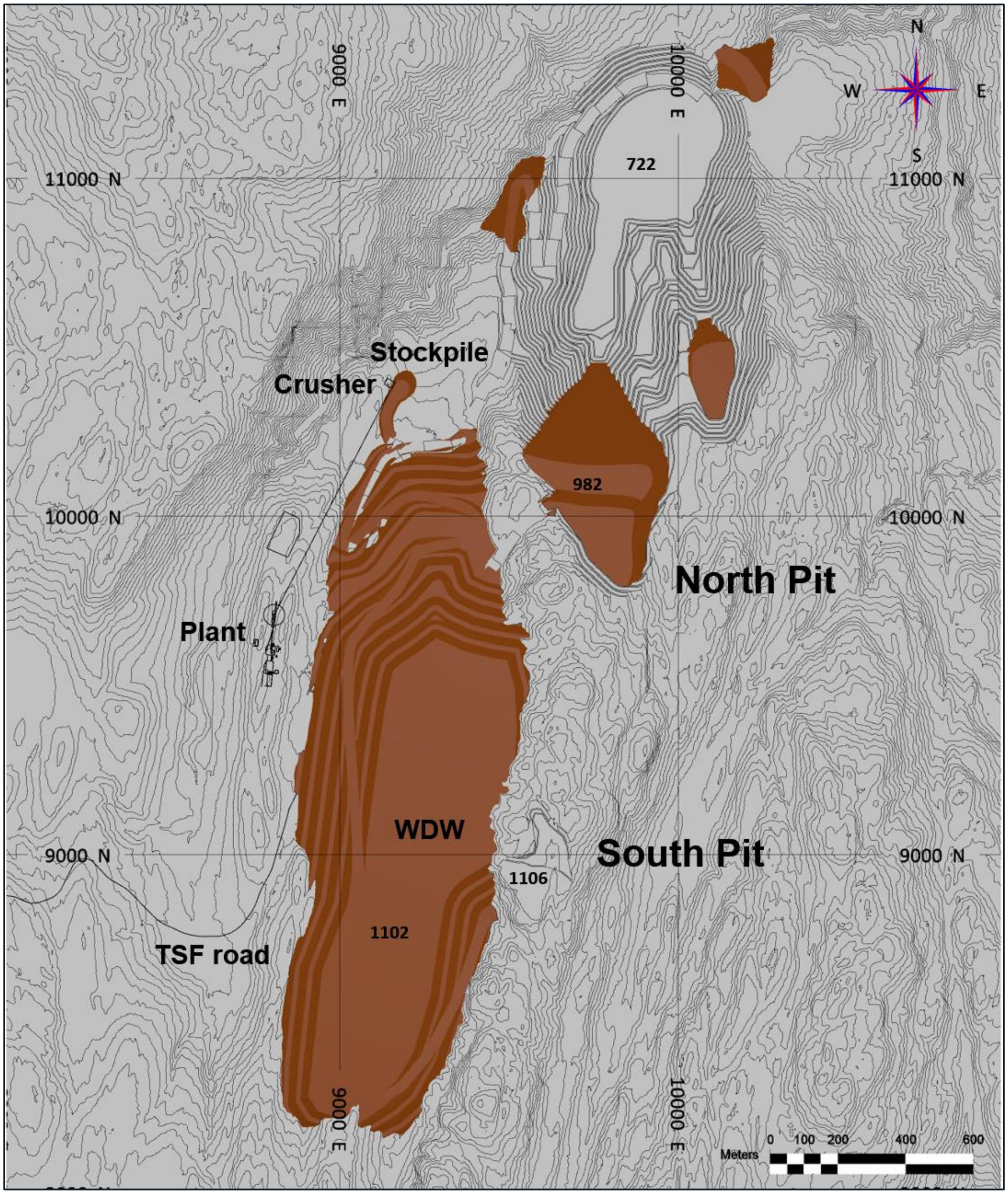
Note: Figure prepared by AGP, 2021.

Figure 16-43: End of Year 7



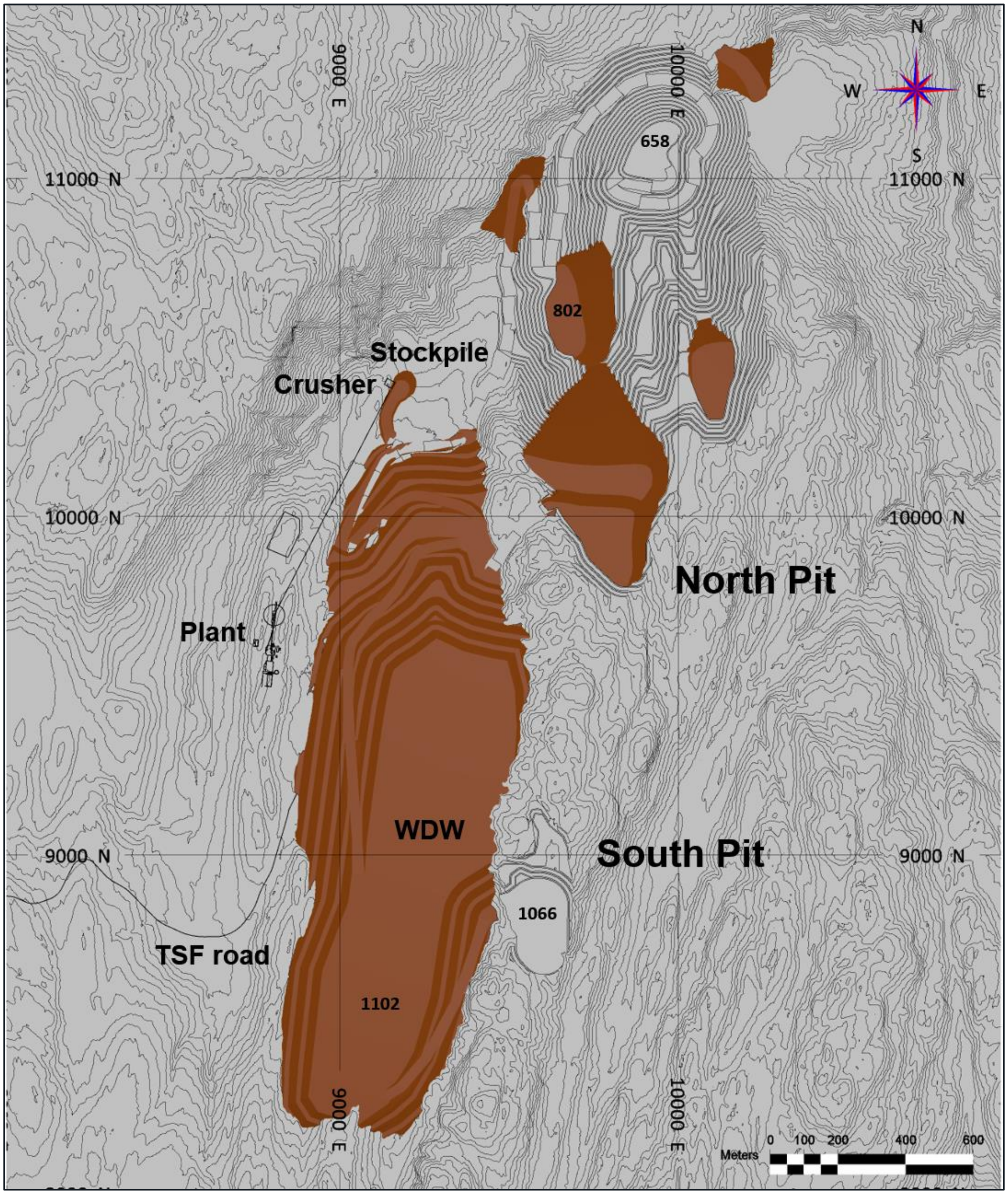
Note: Figure prepared by AGP, 2021.

Figure 16-44: **End of Year 8**



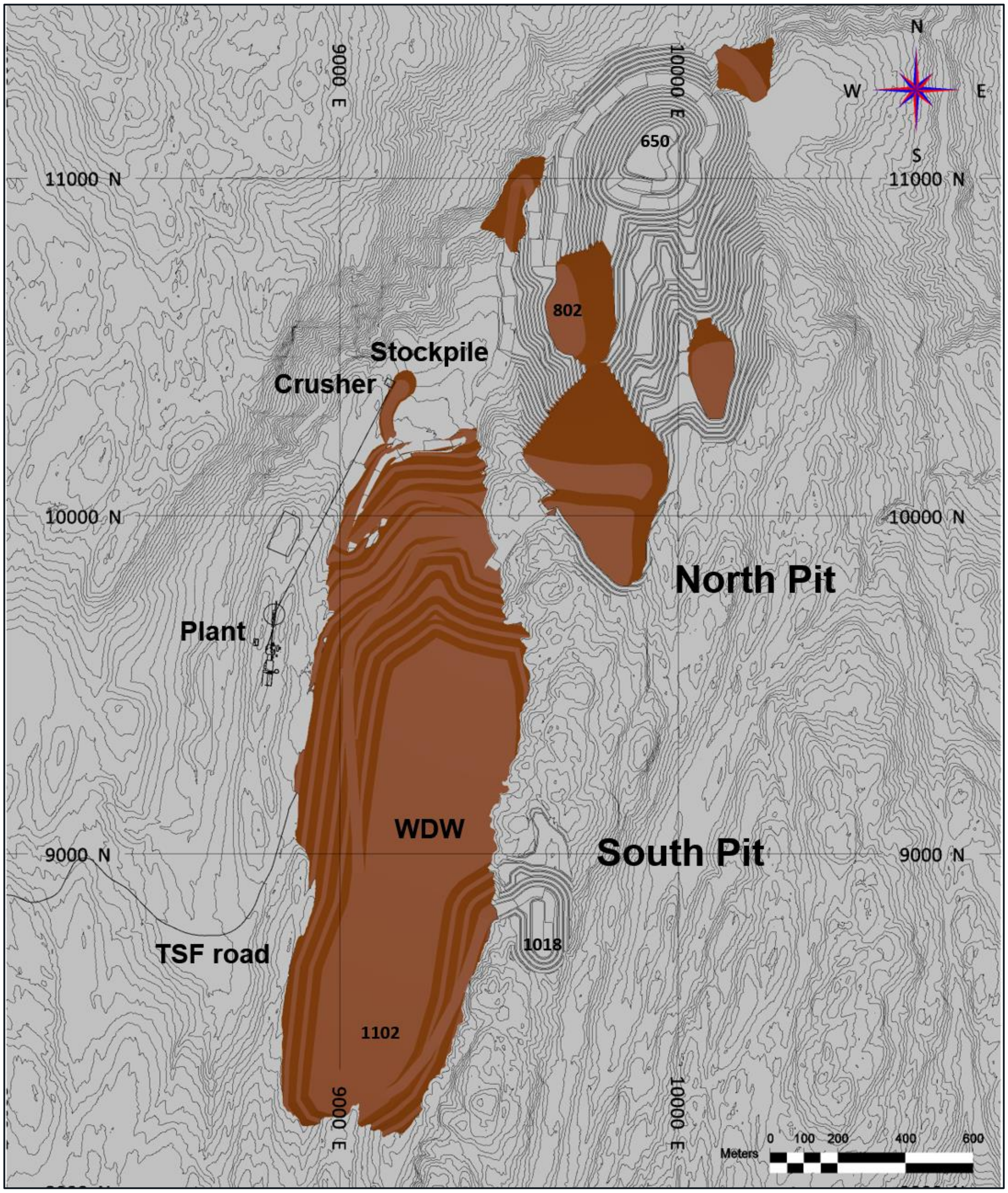
Note: Figure prepared by AGP, 2021.

Figure 16-45: End of Year 9



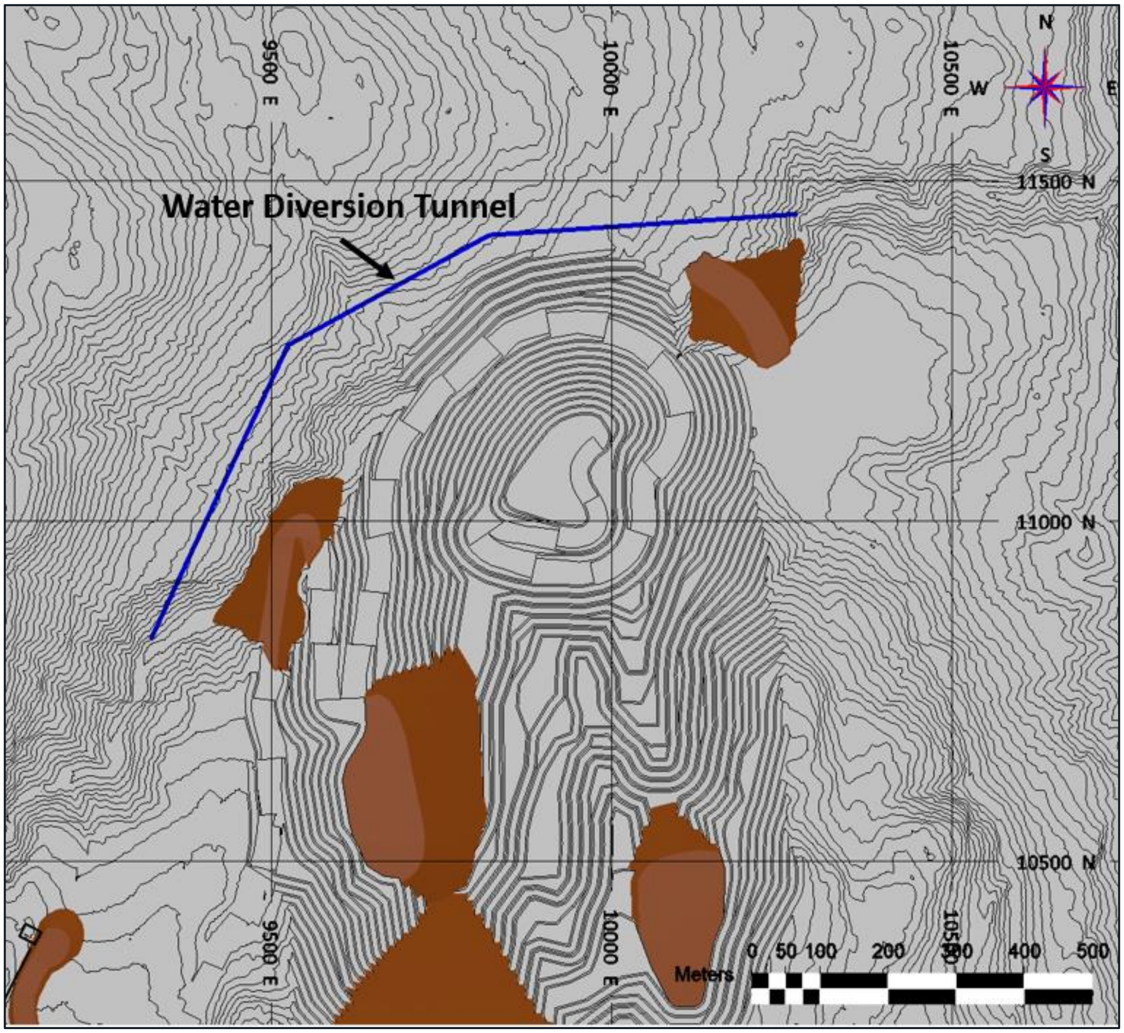
Note: Figure prepared by AGP, 2021.

Figure 16-46: End of Year 10 (End of Mining)



Note: Figure prepared by AGP, 2021.

Figure 16-47: Diversion Tunnel Alignment



Note: Figure prepared by AGP, 2021.

17 RECOVERY METHODS

The plant will process material at a nominal rate of 2.9 Mt/y for Years 1 to 4 and 2.7 Mt/y for the remaining years with an average head grade of 3.2 g/t Au and 94 g/t Ag. The plant is designed to operate two shifts per day, 365 d/y with an overall plant availability of 92%. The process plant feed will be supplied from the Eskay Creek open pit mine and the process plant will produce gold concentrate to be sold to refineries.

The resulting design criteria from the metallurgical test work described in Section 13 used to design the process facility is described in this section.

The processing plant will consist of the following:

- Single stage crushing circuit (jaw), fed from the open pit mine;
- Coarse ore stockpile with reclaim system, fed from an overland conveyor;
- Primary grinding including a SAG mill, pebble crusher (installed at Year 4), and ball mill in closed circuit with hydrocyclones;
- Rougher flotation with conventional concentrate regrind and two stages of cleaning;
- Slimes classification via two stages of hydrocycloning, fed from the rougher flotation tails;
- Secondary grinding and scavenger flotation, fed from the slimes circuit underflow;
- Fines flotation and two stages of cleaning, fed from the slimes circuit overflow;
- Concentrate thickening, storage and filtration;
- Concentrate load-out by way of front-end loader filling concentrate transportation;
- Final tailings pumping to the tailings storage facility (TSF).

17.1 Throughput Overview

As discussed in Section 13, comminution testing concluded that a higher throughput rate can be achieved depending on the zone being processed. The breakage data determined for each year of the mine plan is based on the upper quartile of the breakage properties for each ore zone. The ore becomes harder and more competent after the first four years of operation. Due to the increased difficulty in processing, the plant will process material at a nominal rate of 2.9 Mt/y for Years 1 to 4 then reduce to 2.7 Mt/y in year 5 onward as the material increases in hardness and competency.

Table 17-1: Yearly Comminution Characteristics

	CWi (kWh/t)	RWi (kWh/t)	BWi (kWh/t)	Axb
Year 1	19.1	14.9	16.7	54.1
Year 2	19.2	14.9	17.5	40.8
Year 3	20.8	14.9	16.6	38.0
Year 4	20.9	15.0	16.0	33.5
Year 5	17.9	16.4	18.2	33.6
Year 6	17.6	16.4	18.3	33.9
Year 7	18.0	16.4	18.0	33.4
Year 8	18.3	18.2	20.4	33.4
Year 9	20.4	20.9	20.8	32.4
Year 10	18.8	19.8	22.3	33.4

17.2 Plant Design

17.2.1 Process Design Criteria

The key criteria selected for the plant design are:

- Nominal base plant treatment rate of 7.9 kt/d (Year 1-4) and 7.4 kt/d (Year 5 onwards);
- Design availability of 92%, which equates to 8,059 operating hours per year, with standby equipment in critical areas; and
- Sufficient plant design flexibility for treatment of all ore types.

The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery. An overview of the plant design criteria are listed in Table 17-2. Comminution parameters are provided in Table 17-3.

Table 17-2: Eskay Creek Process Design Criteria - Overview

Description	Units	Years 1 - 4	Year 5+
Ore Throughput	Mt/y	2.9	2.7
Average feed grade, Au	g/t	4.1	2.9
Average feed grade, Ag	g/t	120.8	76.5
Average Concentrate grade, Au	g/t	50.7	43.5
Average Recovery to concentrate, mass	% Plant feed	7.0	5.5
Recovery to concentrate, Au	% Plant feed	85-88	68-89
ROM specific gravity	SG	2.9	
Operating Schedule			
Crusher availability	%	70	
Plant availability	%	92	
Filter plant availability	%	85	
Daily throughput - average	kt/d	7.9	7.4
Plant capacity, nominal @ 92% availability	t/h	360	335

Table 17-3: Comminution Design Criteria

Description	Units	Year 1 – 4	Year 5+
Crushing (Single Stage)			
Primary crusher	type	Jaw Crusher	
Coarse ore stockpile residence time - live	h	8	
Crushing circuit feed, F_{100}	mm	800	
Bond Crusher Work Index (CWi)	kWh/t	18.6	
A x b	-	36.7	
Grinding			
Circuit type	-	SAG Mill, Ball Mill	SAG Mill, Pebble Crusher, Ball Mill
Bond Rod Mill Work Index (RWi)	kWh/t	15.8	
Bond Ball Mill Work Index (BWi)	kWh/t	17.9	
Feed particle size, F_{80}	mm	97	
Product particle size, P_{80}	µm	100	
Pebble crushing rate, nominal	% New feed	N/A	24

Design criteria for the flotation plant were determined from the testwork conducted by BaseMet (described in Section 13) and is summarized in Table 17-4.

Table 17-4: Flotation Plant Design Criteria

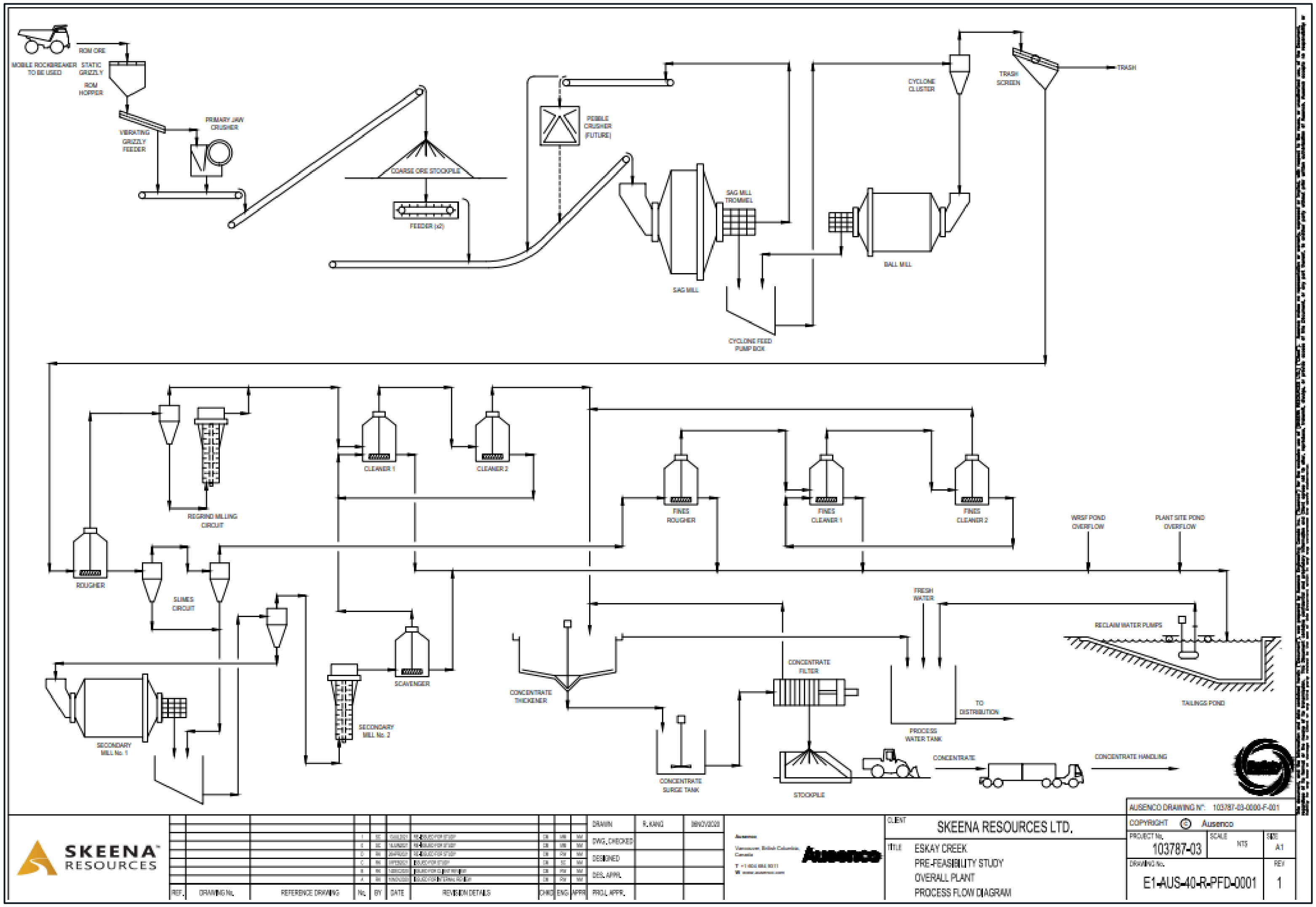
Description	Units	Year 1 – 4	Year 5+
Feed rate	t/h	360	335
Roughers			
Cell type	-	Direct Flotation Reactor (DFR)	
Concentrate grade, Au	g/t	13	
Recovery to concentrate, mass	% Plant feed	11.2	
Regrind Mill			
Feed rate, nominal	t/h	40	37
Feed size P_{80}	µm	85	
Discharge size P_{80}	µm	15	
Specific grinding energy (SGE)	kWh/t	52.1	
Secondary Mill 1			
Type	-	Ball mill	
Feed rate, nominal	t/h	221	206
Feed size, P_{80}	µm	150	
Discharge size, P_{80}	µm	50	
SGE	kWh/t	13.1	
Secondary Mill 2			
Type	-	High Intensity Grind (HIG) mill	
Feed rate, nominal	t/h	221	206
Feed size, P_{80}	µm	50	
Discharge size, P_{80}	µm	30	
SGE	kWh/t	14.1	
Scavengers			
Cell type	-	DFR	
Concentrate grade, Au	g/t	13.4	
Recovery to concentrate, mass	% Plant feed	7.5	
Stage recovery, Au	% Plant feed	37.0	
Cleaners		2 Stages	
Cell type	-	DFR	
Second stage concentrate grade, Au	g/t	31.4	
Recovery to concentrate, mass	% Plant feed	7.8	
Stage recovery, Au	% Plant feed	69.8	
Slimes Roughers			
Cell type	-	DFR	
Concentrate grade, Au	g/t	4.0	

Description	Units	Year 1 – 4	Year 5+
Recovery to concentrate, mass	% Plant feed	3.7	
Stage recovery, Au	% Plant feed	4.1	
Slimes Cleaners		2 Stages	
Cell type	-	DFR	
Second stage concentrate grade, Au	g/t	18.0	
Recovery to concentrate, mass	% Plant feed	0.3	
Stage recovery, Au	% Plant feed	2.6	
Concentrate Dewatering			
Unit area thickening rate (design)	t/ m^2.h	0.3	
Thickener underflow density	% w/w	55	
Filtration cake	t/m^2.h	0.3	
Filter cake moisture	% w/w	12	

17.2.2 Process Flowsheet

The simplified process flowsheet is shown in Figure 17-1.

Figure 17-1: **Simplified Process Flowsheet**



Note: Figure prepared by Ausenco, 17 July 2021

The Eskay Creek flowsheet will incorporate the major process equipment listed in **Table** 17-5.

Table 17-5: Major Process Equipment

Area	Type		Specifications
Primary Crushing	Primary crusher	Model/Type	C120 jaw crusher (or equivalent)
Grinding	SAG mill	No of mills	1
		Size	7.3 m diameter (inside shell) x 4.3 m (effective grinding length or EGL)
		Mill motor power	3.8 MW
	Ball mill	No of mills	1
		Size	5.5 m diameter (inside shell) x 9.0 m (EGL)
		Mill motor power	4.9 MW
	Pebble crusher	-	Installed at Year 4
		Model/type	Raptor 250 cone crusher (or equivalent)
Regrinding/classification	Regrind mill	Type	High Intensity Grind (HIG) mill
		No. of mills	1
		Mill motor power	1.6 MW
	Slimes circuit classification	Type/Arrangement	Cyclone cluster in series
		No. of stages	2
		Diameter	Primary stage cyclone – 800 mm Secondary stage cyclone – 250 mm
	Secondary grinding mill 1	Type	Ball mill
		No. of mills	1
		Mill motor power	3.1 MW
	Secondary grinding mill 2	Type	HIG mill
		No. of mills	1
		Mill motor power	3.0 MW
Flotation	Rougher	Type	DFR cells
		No. of cells	3
		Size (diameter by height or D x H)	2.0 m x 4.6 m
	Scavenger	Type	DFR cells
		No. of cells	4
		Size (D x H)	2.0 m x 4.6 m
	Cleaner 1	Type	DFR cells
		No. of cells	9
		Size (D x H)	1.4 m x 3.6 m
	Cleaner 2	Type	DFR cells
		No. of cells	5

Area	Type		Specifications
		Size (D x H)	1.1 m x 3.2 m
	Fines rougher	Type	DFR cells
		No. of cells	4
		Size (D x H)	1.4 m x 3.1 m
	Fines cleaner 1	Type	DFR cells
		No. of cells	6
		Size (D x H)	0.9 m x 2.5 m
	Fines cleaner 2	Type	DFR cells
		No. of cells	4
		Size (D x H)	0.7 m x 2.3 m
Concentrate dewatering	Concentrate thickener	Type	High-rate
		Size	12 m diameter
	Concentrate filter	Model/Type	Vertical plate, MCDTC-H2100 x 54
		Size	2,100 x 2,100 mm plates 30 mm chamber depth
		Filtration area	432 m^2

17.3 Process Description

17.3.1 Crushing and Stockpile

The crushing facility will be a single-stage crushing circuit that will process the run-of-mine (ROM) ore at a nominal processing rate of 473 t/h in Year 1 to Year 4 and 440 t/h from Year 5 onwards. The crushing facility will operate at 70% availability. The major equipment and facilities at the ROM receiving and crushing areas will include:

- Stationary ROM bin grizzly;
- ROM surge bin;
- Vibrating grizzly feeder;
- Primary jaw crusher;
- Coarse ore stockpile; and
- Stockpile reclaim apron feeders.

The ROM ore will be trucked from the open pit and dumped directly into the ROM surge bin or stockpiled on the ROM storage pad, which can be reclaimed by a front-end loader (FEL) for continuous feed. The ROM ore from the ROM bin will be withdrawn by the vibrating grizzly feeder where the coarse oversize will report directly into a single jaw crusher. The feed material will be crushed and will discharge from the crusher onto the primary crusher discharge conveyor delivering feed material to the coarse ore stockpile. The significant route length (725 m with 150 m lift) from the primary crusher to the coarse ore stockpile will require an overland conveyor.

The coarse ore stockpile feed overland conveyor will be fitted with a weightometer to monitor crushing plant throughput and assist with operational and metallurgical accounting. The coarse ore reporting to the coarse ore stockpile feed overland conveyor will be transferred to the coarse ore stockpile area. The coarse ore stockpile will provide 8 hours of live capacity.

Coarse ore from the stockpile will be reclaimed by two apron feeders at a combined nominal rate of 310 t/h discharging ore to the SAG mill feed conveyor to be fed into the SAG mill. The SAG mill feed conveyor will be equipped with a weightometer to provide data for feed-rate control to the grinding circuit.

17.3.2 Grinding and Classification

The primary grinding circuit for Year 1 to Year 4 will consist of only a SAG mill and ball mill in a closed circuit with classifying cyclones. A pebble crusher will be installed at the end of Year 4 and will start operating in Year 5. The primary grinding circuit for Year 5 onwards will consist of a SAG mill, pebble crusher and ball mill in a closed circuit with classifying cyclones.

The proposed ball mill circulating load is a nominal 250% of new feed.

The primary grinding circuit is designed for a product size 80% passing size (P_{80}) of 100 µm. The SAG mill will be driven by a single 3.8 MW wound rotor drive motor (WRIM) with a liquid resistance starter (LRS) and slip energy recovery (SER) unit to allow for variable speed operation. The single pinion ball mill will be driven by a single 4.9 MW fixed speed WRIM with an LRS.

As required, steel balls will be added into the SAG mill and ball mill using a ball bucket and kibble system to maintain grinding efficiency.

Process water will be added with the coarse ore to the SAG mill to achieve a slurry density of approximately 70% solids (by weight). The SAG mill discharge will pass through a trommel screen. During Year 1 to Year 4, screen oversize will be returned to the SAG mill feed conveyor. Once the pebble crusher is installed and operated from Year 5 onwards, the trommel screen oversize will be transferred to the pebble crusher via a pebble crusher feed conveyor and the crusher product returned to the SAG mill feed conveyor. Undersize from the trommel screen will discharge directly into the cyclone feed pump box, where it will be diluted with process water and pumped to the cyclone distribution manifold via a cyclone feed pump. Cyclones will classify the feed slurry to achieve overflow stream of 30% solids (by weight) comprising product sized particles, whilst the cyclone underflow fraction of 72% solids (by weight) will report to the ball mill.

Cyclone underflow will be ground in the ball mill. Ball mill discharge will flow through the ball mill discharge trommel screen and remove any trash or broken mill balls, which will then be discharged to a concrete ball mill scats bunker. Trommel screen undersize will discharge into the cyclone feed pump box and combined with SAG mill trommel undersize.

The cyclone overflow will report to a trash screen via gravity, which will remove trash to a trash bin. Trash screen undersize will then flow by gravity to the rougher flotation circuit.

Maintenance activities in the grinding and classification area will be serviced by the mill area crane, which will be used for ball mill charging duties and maintenance activities. Spillages in the grinding and classification area will be pumped by the mill area sump pump into the cyclone feed pump box.

In the feasibility stage, further assessment will be conducted to determine the inclusion of a gravity circuit in the flowsheet. Such a gravity circuit would recover free gold by means of centrifugal concentration. Cyclone underflow would report to a gravity circuit screen and its oversize would discharge back to the ground mill for further grinding. The gravity circuit screen undersize would discharge to a gravity concentrator(s).

17.3.3 Flotation and Regrinding

The flotation circuit will consist of roughers, scavengers, fines roughers, cleaners, and fines cleaners flotation, along with regrinding of rougher concentrate, slimes classification of rougher tailings and secondary grinding prior to scavenger flotation. The regrind mill will target a discharge size P_{80} of 15 µm, the slimes classification overflow will target a size P_{80} of 20 µm, and the secondary regrinding circuit will target a final discharge P_{80} of 30 µm.

Cyclone overflow from the primary cyclone will feed a bank of rougher flotation cells. Concentrate from the rougher cells will advance to the rougher concentrate regrind mill cyclone cluster, where its underflow will be pumped to a regrind mill and ground to a P_{80} of 15 µm. The reground product will then discharge to the two-stage cleaner flotation circuit. Overflow from the regrind cyclone will be fed directly to the two-stage cleaner circuit.

Rougher flotation tailings will be pumped to the slimes classification circuit, where two stages of cyclones will produce a P_{80} of 20 µm and will then report to the fines roughers flotation circuit. Cyclone underflow streams from both cyclones will be combined in the secondary mill feed pumpbox to be fed to the secondary regrinding circuit, which will consist of a ball mill in a closed circuit with classifying cyclones and subsequently a HIG mill. The secondary mills will produce a final discharge size P_{80} of 30 µm to feed directly into the scavenger flotation cells.

Concentrate from scavenger flotation will be sent back to the first cleaner circuit via the scavenger concentrate pumpbox and discharge pump. Scavenger flotation tailings will be combined with cleaner 1 tailings, fines rougher tailings and fines cleaner 1 tailings in a pumpbox before being pumped to the TSF.

Cleaner 1 concentrate will be subsequently cleaned in the cleaner 2 cells. Concentrate from the second cleaner will be combined with concentrate from second fines cleaner flotation cells in a pumpbox prior to being sent to the concentrate thickener. Tailings from second cleaner will be pumped back to the bank of cleaner 1 cell.

Overflow from the two-stage desliming circuit will undergo the fines rougher flotation circuit. Concentrate produced by the fines rougher cells will feed the first fines cleaner flotation cells and subsequently cleaned in second fines cleaner cells. The second fines cleaner tailings will be pumped back to the first fines cleaner.

Reagents used in the flotation circuit will include potassium amyl xanthate (PAX; collector), copper sulphate (promoter), methyl isobutyl carbinol (MIBC; frother), and flocculant for thickening.

17.3.4 Concentrate Dewatering

Concentrate from the second cleaner will be combined with concentrate from the second fines cleaner circuit in a pumpbox to be pumped to the concentrate thickener feed de-aeration tank. The final concentrate will be thickened in a high-rate thickener.

Thickener underflow will be pumped to a concentrate filter feed tank with a nominal residence time of 12 hours prior to dewatering in a pressure filter with filter availability of 85% or 7,446 hours per year. The dewatered concentrate will be discharged to a covered storage bunker with a stockpile nominal capacity of 7 days or 4,174 t. The final product will then be loaded to trucks by front-end loader.

17.3.5 Tailings Disposal

Scavenger tailings, cleaner 1 tailings, fines rougher tailings, fines cleaner 1 tailings will be combined in a pumpbox prior to being pumped to the TSF.

17.3.6 Reagents

The reagents will be prepared and stored in separate self-contained areas within the process plant and delivered by individual metering pumps to the required addition points for the reagents. Reagents will include:

- Collector: PAX is a sulphide mineral collector and will be supplied in 1,000 kg bulk bags as a dry reagent. PAX will be stored in the reagent's storage area of the process plant and delivered to the PAX mixing area. Water will be added to an agitated tank to produce a solution concentration of 15% w/w. The diluted mix will be transferred to the collector distribution tank. The collector will be distributed to required flotation dosing points by dedicated metering pumps.

- Promoter: copper sulphate ($CuSO_4$) is an activator to promote the interaction of collector molecules with the mineral surfaces and will be supplied in 1,200 kg bulk bags in the form of crystalline powder. $CuSO_4$ will be stored in a separate self-contained area within the process plant and delivered to the $CuSO_4$ mixing area. Water will be added to an agitated tank to produce a solution concentration of 15% w/w. The diluted mix will be transferred to the $CuSO_4$ distribution tank prior to distributing to required addition points by dedicated metering pumps.

- Frother: MIBC will be supplied in 810 kg IBC totes. MIBC will be delivered to required flotation dosing points directly from the IBC totes by dedicated metering pumps.

- Flocculant: MF336 (or similar). A flocculant mixing, storage and dosing system located in a separate self-contained area within the process plant will be provided to facilitate concentrate thickening. MF336 will be supplied in 25 kg bags and will be shipped as a dry reagent. The bags will be lifted and loaded into the flocculant hopper. Loose flocculant will be transported via a screw feeder to the flocculant mixing tank. Water will be added to the agitated mixing tank to produce a solution concentration of 0.25% w/w. The diluted flocculant mix will then be transferred to the flocculant storage tank via a transfer pump. The flocculant will be pumped by way of a metering pump to an inline mixer where the solution will be further diluted to 0.025% w/w and fed to the concentrate thickener.

17.3.7 Services

17.3.7.1 Air Services

Two rotary screw plant air compressors will supply low-pressure process air to the DFR cells. Flotation air will be stored in the flotation air receiver to account for variations in demand prior to being distributed to the flotation circuits. The total air demand for all DFR cells at site atmospheric pressure is 2,530 Am^3/h.

A third air compressor will supply high-pressure air for the concentrate filter press requirements. It will be equipped with its own dedicated air dryer and receiver. The same air compressor will provide intermediate pressure compressed air for instrument air requirements. Instrument air will be dried in air dryers prior to being distributed throughout the plant.

17.3.7.2 Water Services

17.3.7.2.1 Fresh Water

Fresh water will be pumped from borehole wells to feed the plant. Fresh water in the tank will be used to supply the following services:

- Fire water;
- Gland seal water;
- Potable water;
- Reagent mixing; and
- Make-up water for the process water system.

Fresh water will be supplied to the plant by two freshwater pumps in a duty/standby configuration.

17.3.7.2.2 Potable Water

Potable water will be sourced from the freshwater tank and treated in the potable water treatment plant. The treated water will be stored in a potable water storage tank for use by two potable water pumps in a duty/standby configuration.

17.3.7.2.3 Gland Water

Gland water will be supplied from the freshwater tank and distributed to the plant by two freshwater pumps in a duty/standby configuration. Gland water pumps will be used to boost the fresh water supply pressure to supply high-pressure gland water users such as filter feed pump.

17.3.7.2.4 Process Water

Process water will consist mainly of TSF reclaim water and concentrate thickener overflow. Process water will be stored in a process water storage tank and distributed by two process water pumps in a duty/standby configuration.

17.3.7.3 Assay/Metallurgical Laboratory and Quality Control

The process plant will be equipped with sampling points to collect shift and routine samples for AA and fire assays. Those samples will include feed, intermediate flotation products, tailings, and final products. The data obtained will be used for product quality control and routine process optimization.

The metallurgical laboratory will perform metallurgical tests for quality control and process flowsheet optimization. The metallurgical laboratory will include equipment such as laboratory crushers, ball mill, sieve screens, laboratory flotation cells, balances, and pH meters.

17.4 Plant Design

The process plant (including primary crushing) location selection was part of the overall site layout optimization study conducted at the commencement of this study phase and is discussed in more detail in Section 18. Process plant power requirements and reticulation are described in Section 18.

The primary crushing circuit will be in close proximity, and elevation, to the main pit exit. The location and design considers the association and interface with other major infrastructure in the area, specifically the main haul road, ore stockpile, ROM pad, WRSF, and water collection ponds.

The primary crushing circuit layout will consist of a modularized primary crusher station and steel supported ROM bin. The crushing station will be mounted on cut material at an elevation of 902 masl and separated from the ROM pad by an 11 m high mechanically stabilized earth wall.

The process plant (including stockpile and reclaim) will be located on the ridge line adjacent to the WRSF. The ridge will be excavated down to an elevation of 1014 masl to create a single plant pad. An overland conveyor will connect the primary crusher station to the process plant.

The process plant (including concentrate storage and loadout) will be housed within a single pre-engineered building that will be 30 m wide and 121 m long. The building will have multiple overhead travelling cranes to support construction and maintenance, and the roof line will have two step changes in height to reduce the overall building volume. The plant will be separated into five main areas within three areas that will be based on building height:

- High section:
  - primary grinding and classification,
  - regrinding, slimes classification and secondary grinding,
- Middle section:
  - flotation and reagents,
  - concentrate thickening and filtration,
- Low section (no fixed crane allowance):
  - concentrate storage and loadout.

The main water tanks (fresh and process) and concentrate thickener will be located outside of the main building and covered by a roof extension from the main building.

17.5 Process Control Strategy

The process control strategy to be implemented for the Eskay Creek project is typical of those used in modern mineral processing operations.

Field instruments will provide inputs to a set of programmable logic controllers (PLCs). The PLCs will be used to control and monitor operation of the plant and will be broken into different process areas. Each process area will be controlled by a single PLC system. The PLCs will be tied together to form a plant wide control system by the use of an ethernet communication system.

The PLCs will perform the control functions by:

- collecting status information of drives, instruments, and packaged equipment;
- providing drive control and process interlocking;
- providing proportional-integral-derivative (PID) control for process control loops.

Process control and monitoring for the facility will be performed in two centralized control rooms housed in the main process plant and in the primary crusher area. Human machine interface (HMI) operator stations will be located in the control rooms. HMIs will contain graphical representation of process equipment. The PLC in conjunction with the HMI will perform all equipment and process interlocks, level control, alarms, trends, and report generation.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

This section describes the permanent physical infrastructure and logistics support facilities for the project. The infrastructure and facilities will include:

- Main access road: a 59 km all-season gravel road from Highway 37 (Stewart Cassiar Highway) to the site.
- Internal roads: all roads within the site required to connect the facilities, maintaining separation of light and heavy vehicles.
- Site logistics: adequate infrastructure to support the storage, management and transport of goods and materials into and from the plant during construction and operations.
- Site buildings: all buildings required to support the facilities and operations. These will include the gate house, administration building, laboratory, plant workshop, and plant offices.
- Accommodations camp: a permanent 260 person camp for operations and an additional 40 person dormitory for construction. These camps will use common facilities.
- Pioneering facilities enhancement: the installation of demountable office facilities and other temporary infrastructure to provide adequate facilities to support Early Works construction activities.
- Water supply: the water supply will be from a local well field in close proximity to the processing plant. Water will be pumped directly from the wells via a short pipeline to the water storage tank within the plant.
- Water distribution: the storage and distribution of water services from the storage tanks to the facilities. The services will include fresh water, fire water, gland water, potable water, and sewage.
- Power supply: the high-voltage (HV) power supply system to provide for a demand load on the site of 21 MW at 13.8 kV, including a 20 km long, 69 kV overhead transmission line from the Volcano Creek 287 kV substation to the site main HV transformer.
- Power distribution: the distribution of medium voltage power at 13.8 kV to the facility substations across the site, including local facility transformers, low voltage distribution and other ancillaries to support the operation.
- Mine infrastructure area: all the facilities that form the mine infrastructure area; truck bays, ancillary equipment bays, wash bays, tire change area, welding area, lubrication storage area, diesel storage and distribution, offices, and warehouse.
- Communications: the network communication architecture, energy management system, and associated hardware to support the operation.
- Mobile equipment: the mobile equipment required to support the operation.

- Security facilities: infrastructure to ensure the safety of personnel and assets on and off site including the site gate house, security fencing and closed-circuit television (CCTV) systems.

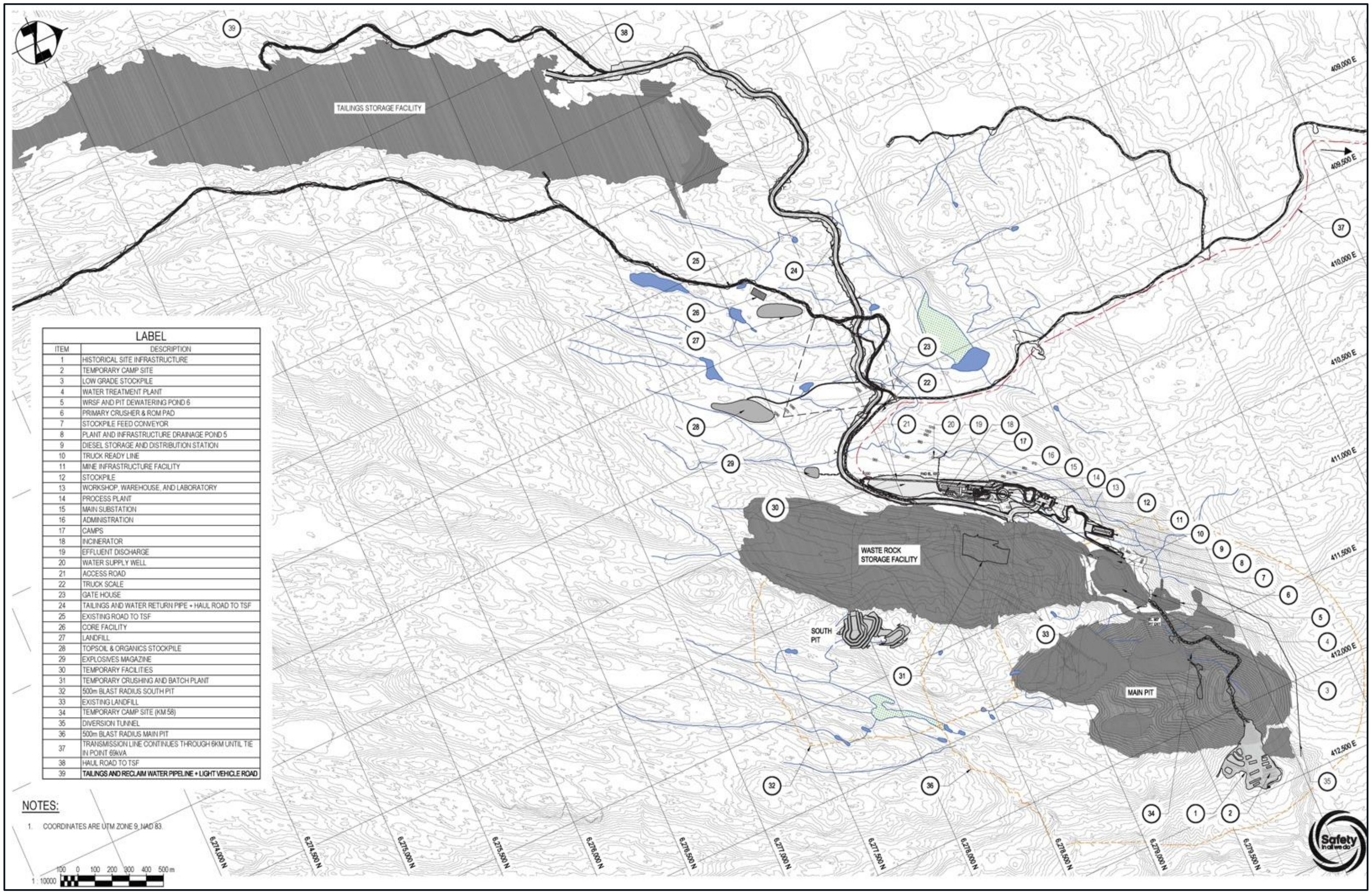
- Information Technology: the information technology (IT) and communications requirements for the project including data, media, and voice transmission services.

At the commencement of this study phase, an overall site layout optimization study was conducted, with the objective of finalizing the location of major infrastructure and agreeing on the area layout concepts to ensure a low cost and effective design and operating solution.

The optimization study considered five plant and infrastructure locations coupled with three ROM/crusher locations. The options were established and then discussed during a strengths, weaknesses, opportunities, threats (SWOT) analysis workshop. A quantitative analysis and cost comparison was completed based on the SWOT analysis, followed by a fatal flaws discussion and final option ranking.

The result of the optimization study and all Project infrastructure are indicated in Figure 18-1.

Figure 18-1: Project Proposed Layout Plan



LABEL

ITEM	DESCRIPTION
1	HISTORICAL SITE INFRASTRUCTURE
2	TEMPORARY CAMP SITE
3	LOW GRADE STOCKPILE
4	WATER TREATMENT PLANT
5	WRSF AND PIT DEWATERING POND 6
6	PRIMARY CRUSHER & ROM PAD
7	STOCKPILE FEED CONVEYOR
8	PLANT AND INFRASTRUCTURE DRAINAGE POND 5
9	DIESEL STORAGE AND DISTRIBUTION STATION
10	TRUCK READY LINE
11	MINE INFRASTRUCTURE FACILITY
12	STOCKPILE
13	WORKSHOP, WAREHOUSE, AND LABORATORY
14	PROCESS PLANT
15	MAIN SUBSTATION
16	ADMINISTRATION
17	CAMPS
18	INCINERATOR
19	EFFLUENT DISCHARGE
20	WATER SUPPLY WELL
21	ACCESS ROAD
22	TRUCK SCALE
23	GATE HOUSE
24	TAILINGS AND WATER RETURN PIPE + HAUL ROAD TO TSF
25	EXISTING ROAD TO TSF
26	CORE FACILITY
27	LANDFILL
28	TOPSOIL & ORGANICS STOCKPILE
29	EXPLOSIVES MAGAZINE
30	TEMPORARY FACILITIES
31	TEMPORARY CRUSHING AND BATCH PLANT
32	500m BLAST RADIUS SOUTH PIT
33	EXISTING LANDFILL
34	TEMPORARY CAMP SITE (KM 58)
35	DIVERSION TUNNEL
36	500m BLAST RADIUS MAIN PIT
37	TRANSMISSION LINE CONTINUES THROUGH 6KM UNTIL TIE IN POINT 69kVA
38	HAUL ROAD TO TSF
39	**TAILINGS AND RECLAIM WATER PIPELINE + LIGHT VEHICLE ROAD**

NOTES:

1. COORDINATES ARE UTM ZONE 9, NAD 83.

Note: Figure prepared by Ausenco, 2021.

18.2 Off-Site Road and Logistics

Access to the Eskay Creek Project is via the existing Eskay Mine Road, which is a 59 km all-season gravel road that connects to Highway 37 (Stewart Cassiar Highway) approximately 135 km north of Meziadin Junction. The Eskay Mine Road has a combination of single and double lane sections with a total of 8 single lane bridge structures, and has a design speed of 30 km/h to 60 km/h. The road and bridges were independently assessed, and an opportunity exists to upgrade two of the older bridges to accommodate 72,300 kg gross vehicle weight (GVW) trucks. These would be special 9-axle "B-train" truck and trailer units that would operate under a "bulk haul" permit from the Province of BC Ministry of Highways to move concentrate from the mine approximately 250 km to existing export terminal facilities in Stewart, BC. The nominal round trip cycle time for this route including loading and unloading times is about 15 hours.

Multiple options for the export of concentrate were studied, with two options through Stewart identified as preferred. The first uses bulk haul of concentrate with heavy-haul side-dump truck and trailer units to the SBT facility, which will store the product in a separate bulk storage building before loading bulk carrier vessels with a conventional bulk shiploader. The second option uses specialized bulk container units that would be trucked to SWP where loaded containers would be swapped with empty ones and used as temporary storage until a bulk vessel arrives. The containers would be lifted by the ship's cranes using a specialty spreader unit that would discharge the concentrate directly into the vessel's hold.

Both transportation options have similar overall logistics costs for the movement of concentrate from the mine into a ship. The bulk haul trucks can carry up to 48 t of concentrate, whereas the tare weight of the containers limits the load in each container to 23 t and each truck in this option to 46 t. The bulk carrier vessel would be the same in each case and would transport the concentrate to a terminal facility nearest the preferred smelter location in southern China.

Construction materials and mine consumables would be moved through the SBT site, which has a general cargo dock. This facility already serves as an import hub for grinding media used by other mines in the region, and there may be synergies available for back-haul using the bulk containers through this operation.

The movement of personnel for the operation of Eskay Creek will be done with buses moving employees from various communities from the region along the same road.

18.3 Stockpiles

Stockpiles are discussed in Section 16 of this Report.

18.4 Waste Rock Storage Facilities

The planned waste storage is discussed in Section 16.9.

18.5 Tom MacKay Storage Facilities

Tom MacKay Storage Facility (TMSF) and associated surface water management design features were undertaken by Ausenco.

18.5.1 Historical Tailings Deposition

The Tom MacKay Storage Facility was used by the previous operator for subaqueous tailings disposal due to the PAG nature of the tailings produced from the Eskay Creek Mine. In 2002, the BC Government, in accordance with Schedule 2 of Section 36 of the Fisheries Act, classified the lake as a tailings impoundment area. Over 585,000 dry tonnes of tailings were deposited in the facility during the period 2001–2008.

18.5.2 Waste Material Storage Disposal and Site Selection

A series of meetings were held between Skeena, Ausenco, AGP and SRK to review the waste storage options during the development of the PFS. Various disposal options were developed for both PAG and NAG tailings and waste rock. The desk top study looked at environmental and technically acceptable options. From the available short and long term mitigation measures for PAG materials subaqueous deposition was chosen. Therefore, the storage of both PAG and NAG tailings and PAG waste rock into TMSF and the NAG waste rock deposited in the WRSF next to the open pits was chosen as the preferred option.

The general area includes a number of small catchment basins, with the TMSF being the largest along with this facility being the only one to contain both the tailings and PAG waste rock. Figure 18-2 shows the general physiographic and hydrogeological setting of the TMSF.

The general design criteria for the siting study considered tailings and PAG waste rock storage requirement of 76.7 Mt deposited subaqueously while maintaining a minimum of 3 m of water cover during operation and 6 m of water cover at closure over the tailings and PAG waste rock to prevent acidification, together with ensuring the consolidated bed of tailings and PAG waste rock is not remobilized due to ice and wind/wave action.

The TMSF was selected as the preferred tailings and PAG waste rock storage option since it is permitted as a waste storage facility and has sufficient capacity to contain 76.7 Mt of tailings and PAG waste rock. The TMSF requires three embankments to contain the required volume of tailings and PAG waste rock and water cover.

18.5.3 Dam Break Analysis

The embankments for the tailings and PAG waste rock embankments at Eskay are designed in accordance with Canadian Dam Association (CDA) “Dam Safety Guidelines” (CDA 2007; 2013), which also provides guidelines in evaluating the classification of dams in terms of the consequence of failure. The stability of the TMSF embankments were evaluated as part of the PFS design for a range of conditions and a failure of these dams is not likely to occur. The dam breach and inundation study for the TMSF was completed for hypothetical failures under extreme and highly unlikely events. The results of the analysis do not reflect upon the structural integrity or safety of the dams.

The dam breach and inundation study for the TMSF was completed following CDA guidelines (CDA 2007; 2013). The study was undertaken to provide a preliminary understanding of the potential consequences of a TMSF embankment failure and was structured to estimate the potential inundation limits that would result from a dam breach during the last year of operations, i.e., the ultimate configuration. It also considers that very little waste materials will be released from the facility since the PAG waste rock is located near the embankments acting as embankments and the tailings at the back of the facility contained by the waste rock.

The tailings and PAG waste rock Facility is located at the north end of TMSF and the headwater of Tom MacKay creek. Tom MacKay Creek flows into Ketchum Creek and then into Unuk River. Based on a dam break occurred on any of the three (3) north embankments, water would flow into Tom MacKay Creek.

Tom MacKay Creek is steep with a bed morphology of cobble substrate and vegetated banks with exposed bedrock with multiple fish barriers. Based on information provided by RTEC, this is a non-fish bearing stream. Ketchum Creek is a steep medium size creek with a bed morphology of cobbles substrate and vegetated banks with exposed bedrock with multiple fish barriers. The Unuk River is a medium to large low gradient braided river that flow into the Pacific Ocean through the United States. The river is fish bearing with a number of species.

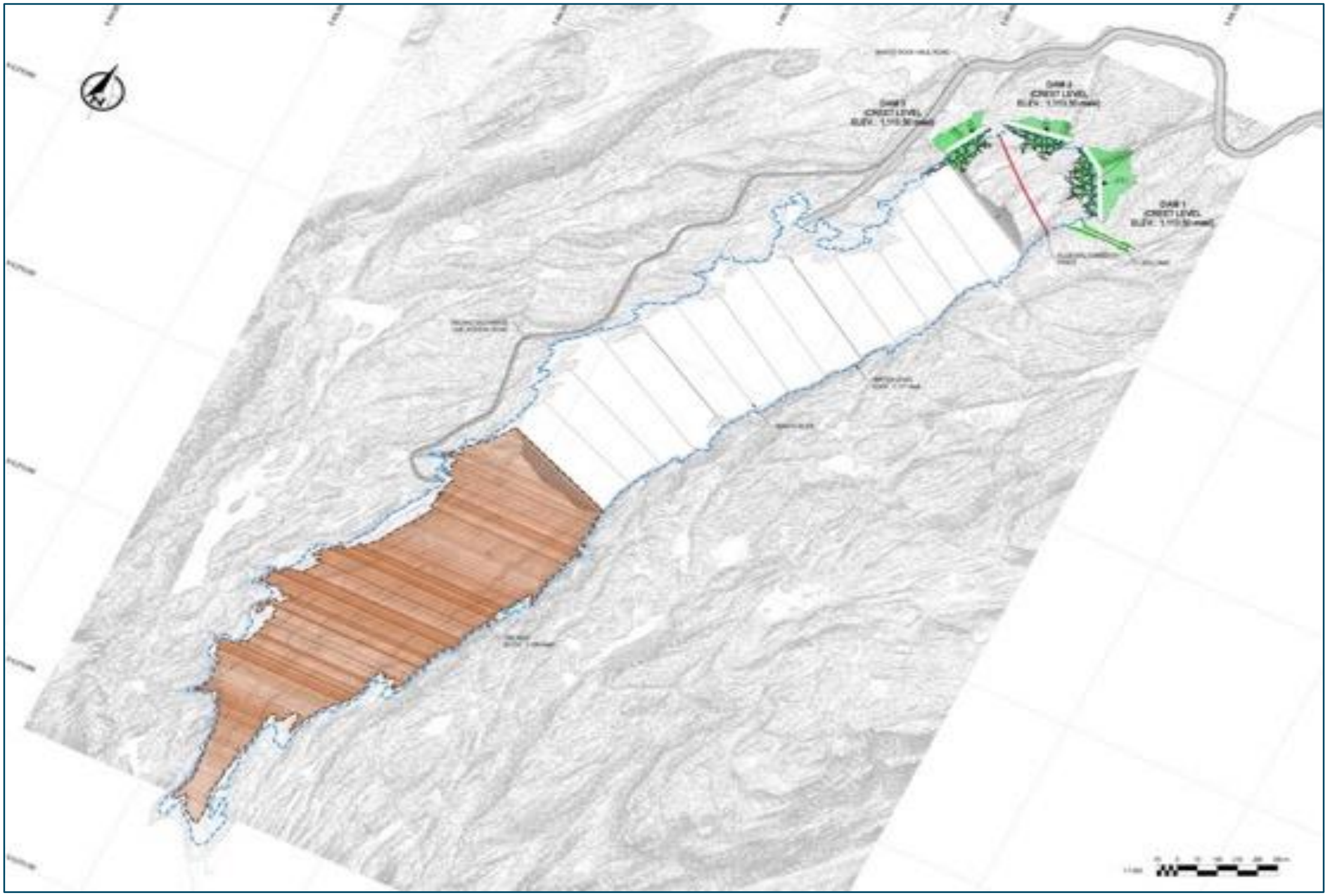
The outflow hydrographs generated due to hypothetical dam breach, including the PMF, was routed downstream using HEC-RAS model. The model was used to predict the extent of flooding due to the dam breach. Modeling of flood was undertaken to estimate the incremental impacts of failure should the TMSF breach during an extreme flood event. The effects of the dam breach are combined with occurrence of a PMF in the project area and downstream to the Unuk River. HEC-RAS was used to prepare inundation map for the impacted areas.

Based on the dam breach analysis and expected area of inundation downstream of the tailings and PAG waste rock storage facility, the consequence of a dam failure based on HSRC Guidance Document, Section 3.4 (BC Ministry of Energy and Mine 2016) and CDA (2013) Dam Safety Guidelines is "very high" for the TMSF. Therefore, the facility was design in accordance with those guideline parameters.

18.5.4 Tom MacKay Storage Facility Design Assumptions/Criteria

The proposed process plant site location in relation to the TMSF was shown in Figure 18-1. The flotation process will produce a combined tailing stream that contains both NAG and PAG materials. Sub-aqueous disposal requirement was conservatively assumed as a design requirement.

Figure 18-2: TMSF General Layout



Note: Figure prepared by Ausenco, 2021.

The TMSF was designed based on the following criteria:

- Required storage of 26.4 Mt of NAG and PAG tailings;
- Tailings particle size P_{80} = 45 µm;
- Tailings discharge solids content = 22% (by mass);
- Dry tailings density of 1.40 t/m^3;
- Required storage of 50.3 Mt of PAG Waste Rock;
- Dry PAG waste Rock density of 1.95 t/m^3
- Subaqueous deposition;
- Minimum stability factors of safety of 1.5 under static conditions and 1.0 under seismic loading, in accordance with CDA Dam Safety Guidelines;
- A penstock with a maximum design discharge of 3,000 L/s; and
- A spillway constructed in Year 5 during the Embankment raises to safely pass the inflow design flood (IDF), resulting from the probable maximum flood (PMF) event.

The proposed TMSF design assumptions include:

- Minimizing disturbance footprint through use of existing mine infrastructure;
- Limiting watershed disturbance to a single catchment basin;
- Limiting impacts to wildlife and fisheries resources;
- Designing for sub-aqueous deposition closure;
- Meeting or exceeding applicable regulatory requirements and industry guidelines for stability and design flood events.

18.5.5 Tom MacKay Storage Facility Design and Construction

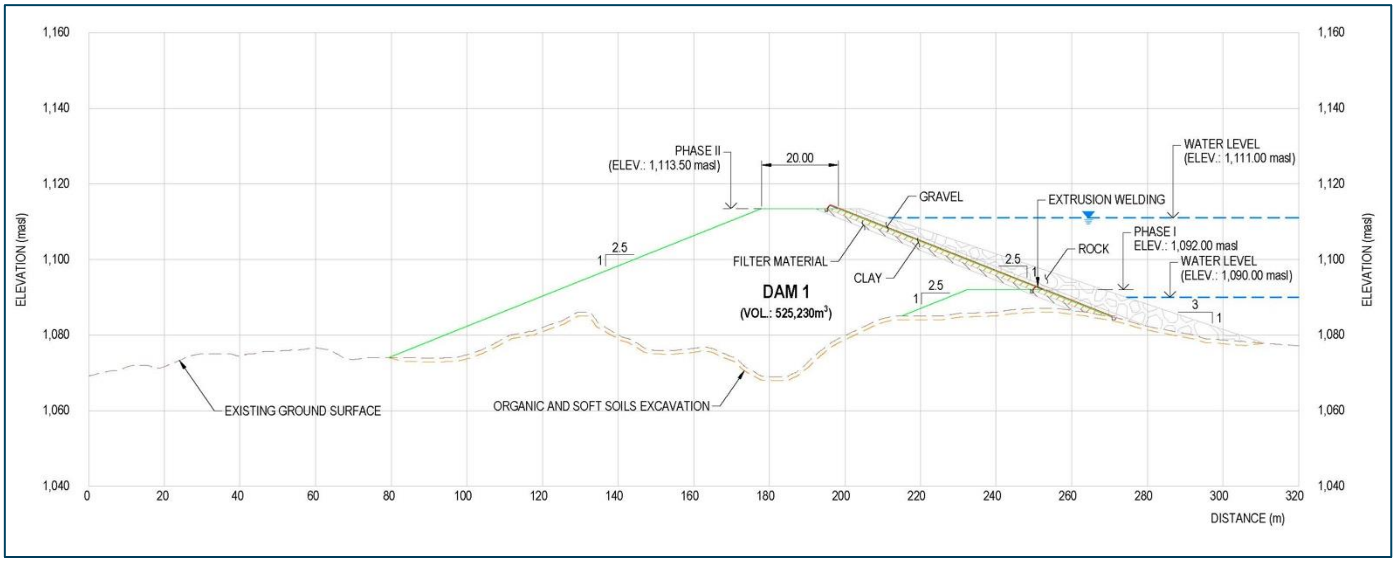
The TMSF is approximately 3.8 km long and 0.5 km wide, and its long axis orientation is southwest–northwest. The current facility ranges in depth from 10 m at the south end to 42 m in the north–central section of the lake. The existing volume of water in the TMSF is around 12.9 Mm3 at elevation 1,079 masl, which is the current outlet elevation of the basin.

The TMSF is designed to be founded on bedrock with low permeability characteristics to limit seepage below the embankment. The overall design objective of the TMSF is to protect the regional groundwater and source waters resources during both operations and over the long term (post-closure). TMSF development will be phased with downstream embankment construction methodology. NAG mine waste from the pit will be utilized as the primary construction material.

The upstream side of the embankment will be lined with a geomembrane to minimize potential seepage through the dams. The geomembrane will be anchored to the bedrock using a concrete/plinth to create a watertight seal. Between the geomembrane liner and the waste rock shell is a filter zone and low permeability zone to aid in minimizing seepage through the embankments. The filter and low permeability zones will be processed material sourced from local borrow areas.

TMSF will be constructed in two phases over the life of mine based on storage and operating criteria. The TMSF embankment design concept is in shown in Figure 18-3, based on the geotechnical investigation.

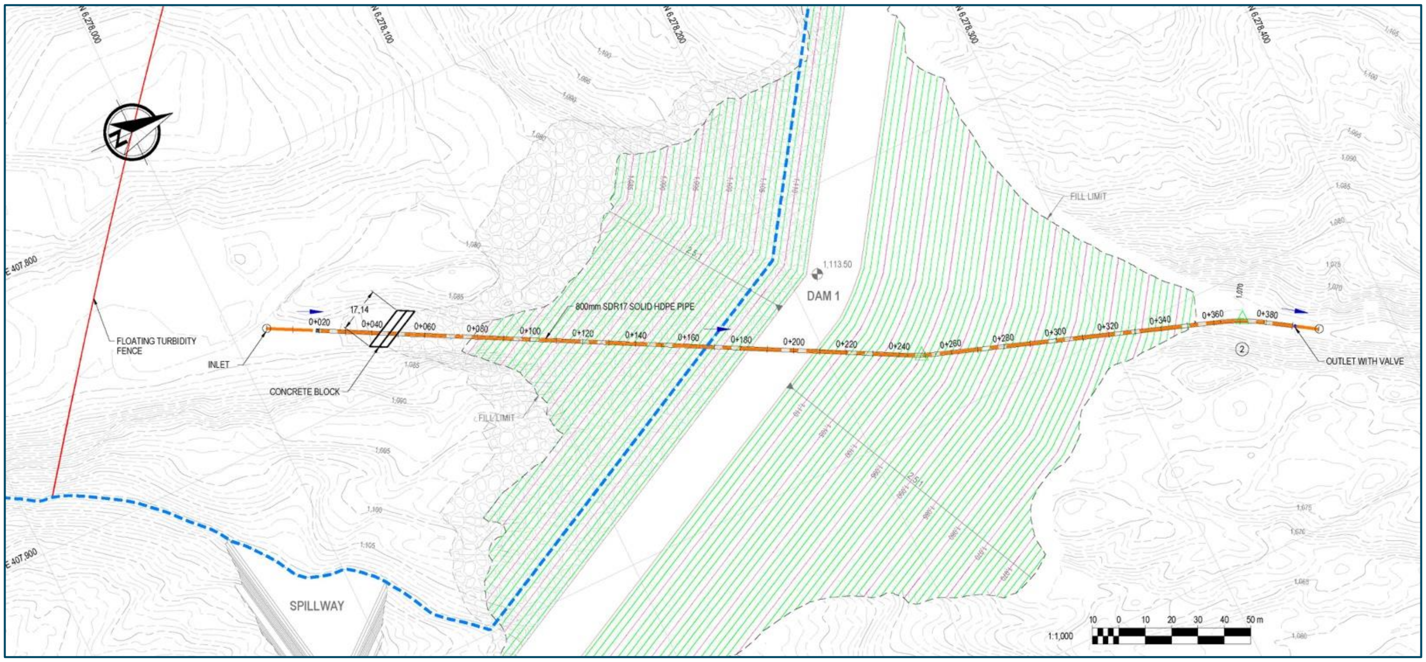
Figure 18-3: Typical Embankment Section



Note: Figure prepared by Ausenco, 2021.

For Phase 1 all three embankments will be constructed to a height of 1,092 masl and a crest width of 20 m to allow for vehicles and equipment access during construction. The embankment has 2.5:1 (H:V) upstream slope and 2.5:1 (H:V) downstream slope. A penstock will be installed through embankment 1 along the thalweg of Tom Mackay Creek. The penstock will be constructed with HDPE Solid wall pipe with a seepage collar on the upstream toe and a gate valve at the downstream toe to regulate flow leaving the facility. The penstock has maximum discharge capacity of 3,000 l/s (Figure 18-4). The upstream face of the embankments will be lined with gravel and rip rap to protect the liner system. The work will be completed by a civil contractor. For Phase 2 all three embankments to their ultimate height in year 4 of operations to a height of 1113.5 masl. The three embankments will range in height from 23m to 43m. The Phase 2 construction will utilize downstream stream construction method. The design is the same as Phase 1 with both upstream and downstream slopes of 2.5:1 (H:V) and a crest width of 20m. A spillway will be constructed in Phase 2 and will be constructed on Ri rap and grout to convey the PMF. Closure will utilize a water cover of 6m to prevent the PAG tailings and waste rock from becoming acidic.

Figure 18-4: TMST Penstock



Note: Figure prepared by Ausenco, 2021.

A haul road and tailings road will be constructed from the open pit/plant to TMSF. The roads vary in width and are described in Section 15.1.5. Between the embankment and the waste rock storage area a floating turbidity fence will be installed. The turbidity fence reduces and/or eliminates the passage of fine grained suspend solids that would otherwise be discharge downstream. The water reclaim for the project is located between the floating turbidity fence and the embankments to provide process water for the plant.

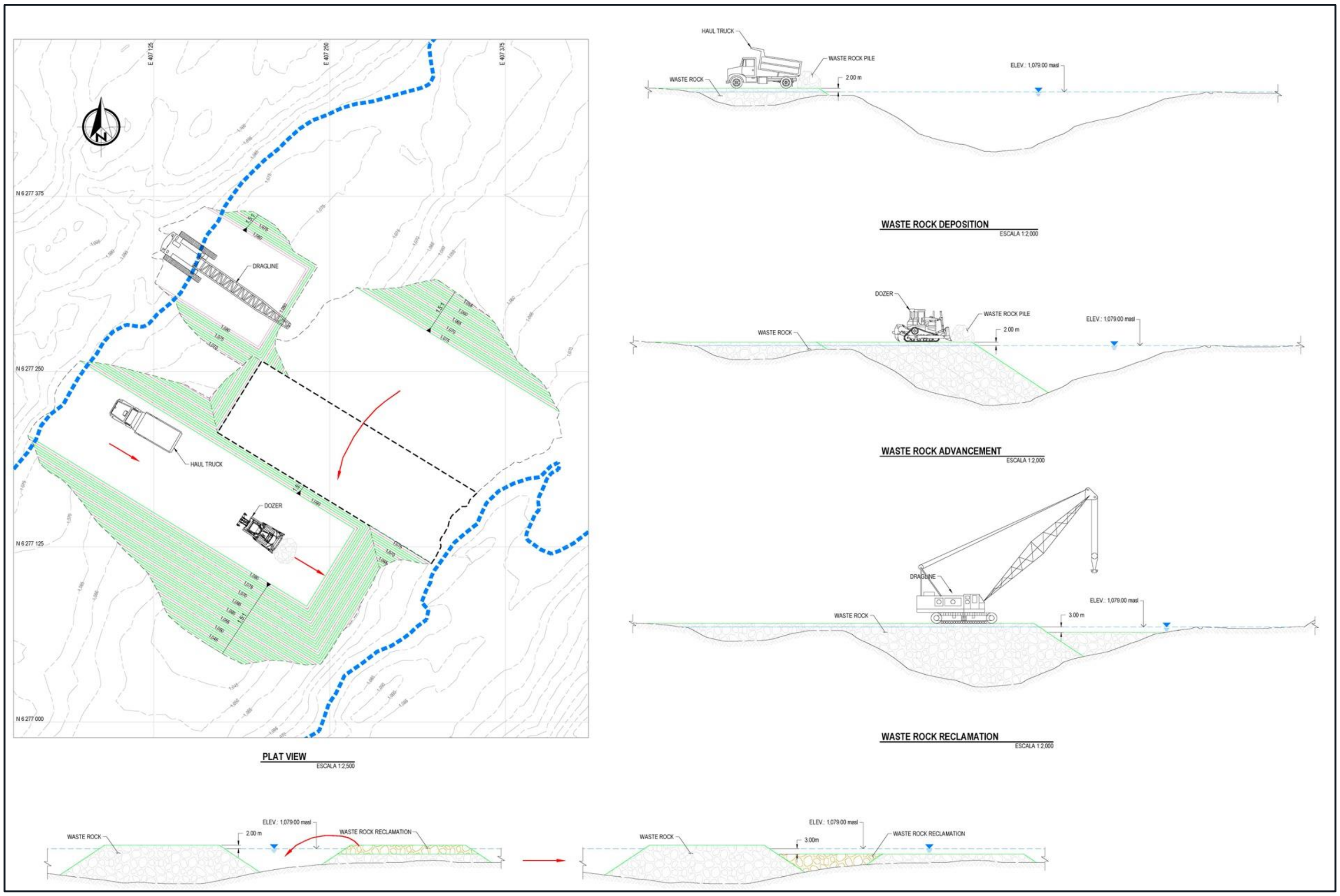
18.5.6 Operations of the Tom MacKay Storage Facility

The operational plan of the TMSF is to deposit slurry tailings at the south end of the facility due to the fine grain nature of the material to allow for additional time for suspend solids to settle out and PAG waste rock at the north end of the facility since the majority of the material is large diameter and suspended solids will settle out faster.

The tailings will be deposited via a barge and single submerged spigot at the bottom of the facility to promote faster settlement of the tailings. The discharge line will be moved around the south end of the facility to promote layering deposition of the tailing.

The PAG waste rock deposition is more completed due to methodologies to place the larger diameters materials sub-aqueously into TMSF. The waste rock will be transport by haul truck to TMSF. The deposition of the waste rock will be by creating berms across the facility west to east. The berms will be constructed 2 metres above the water surface with a crest width of 65 m to provide sufficient operating area for haul trucks, dozers, and dragline excavator. Once completed the next berm will be constructed next to the completed berm. During the construction of the next berm, a dozer and dragline excavator will remove the upper 5 m and place the material to the south of the berm to minimize sediment migration toward the north due to excavation operations (Figure 18-5). The final height of the berm will be 3 m below the water surface.

Figure 18-5: PAG Waste Rock Deposition Plan



Note: Figure prepared by Ausenco, 2021.

Each berm will be constructed 0.5 m higher than the previous berm to accommodate for the displacement of water by the tailings, PAG waste rock and the retention of inflow water (rainfall, run-on from the surrounding watershed, and snow melt). The design calls for the release of a base flow and only allows for retention of water during peak runoff months. Based on one the PAG waste rock deposition plan and the TMSF water balance considering both extreme wet and dry conditions there is sufficient water to maintain a minimum 3 m water cover of the waste materials during operations and 6 m of water cover post closure. It will take 1 to 1.5 years, depending on the end of operations, to raise achieve the 6 m of water cover and continual flow through the spillway.

Tailings will be slurried from the process plant to the TMSF by way of a pipeline, which would extend onto the TMSF to a floating barge. Due to the fine ore grind, P80 = 45 µm, the end of the pipeline will be positioned close to the bottom of facility (deposited tailings) to maximize settling and minimize entrainment of fine particles to the surface of the TMSF. The minimum water depth over the tailings would be 3 m during operations and 6 m at closure to prevent both wind and ice remobilization of the tailings. In addition, the tailings will be stored at the south end of the facility to allow additional time for fine grain particles to settle. The barge would move around the TMSF to develop an even tailings distribution across the TMSF floor.

The tailings and PAG waste rock deposition rates are provided in Table 18-1 and the projected TMSF storage capacities are outlined in Table 18-2. Tailings are planned to be discharged at 22% solids and will have an overall dry bulk density of 1.4 t/m^3 and the waste rock will have an overall bulk density of 1.95 t/m^3. The TMSF has sufficient capacity to store tailings and PAG waste with small embankments to an elevation of 1092.00 masl during the initial years of operations while maintaining 3 m (3–4 Mm^3) of water cover over the tailings and PAG waste rock beds. In year 4 of operations, a single embankment raise will be required to be constructed to an elevation of 1,113.50 masl so as to store the balance of the LOM tailings and PAG waste rock while maintaining 6 m of water cover post closure.

Table 18-1: Planned Tailings and PAG Waste Rock Deposition Schedule

Year	Annual Tailings Production (t)	Annual PAG Waste Rock Production (t)	Total Waste Deposition (t)
-3	-	370,474	370,474
-2	-	434,926	434,926
-1	-	1,514,734	1,514,734
1	2,005,833	3,256,851	5,262,684
2	2,900,000	4,273,193	7,173,193
3	2,900,000	3,596,569	6,496,569
4	2,700,000	4,482,986	7,182,986
5	2,700,000	6,158,237	8,858,237
6	2,700,000	5,319,427	8,019,427
7	2,700,000	6,576,124	9,276,124
8	2,300,000	5,412,437	7,712,437
9	2,300,000	5,370,487	7,670,487
10	2,300,000	3,587,303	5,887,303
11	913,215	-	913,215
Total	26,419,048	50,353,748	76,772,796

Note: Table prepared by Ausenco, 2021.

Table 18-2: **TMSF Projected Tailings and Pag Waste Rock Storage Capacity**

Elevation (masl)	Accumulated Capacity (Mm³)	Comments
1,050	0	TMSF Lowest Portion of Bed
1,079	12.6	Existing TMSF Water Surface
1,087	21.1	Top of Waste Materials Phase 1
1,090	25.2	Water Surface Phase 1
1,092	27.4	Top of Embankments Phase 1
1,108	44.6	Top of Waste Materials Phase 2
1,111	57.3	Water Surface Phase 2
1,113.5	63.4	Top of Embankments Phase 2

Note: Table prepared by Ausenco, 2021.

18.5.7 Tailings Storage Facility Stability

A section through the highest portions of the embankments were selected as the critical section as shown in Figure 18-5. Stability of the embankments were assessed using the limit-equilibrium modelling software Slope/W, (Geostudio, 2018). Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with CDA guidelines of 1.5 FOS for static and 1.0 FOS for pseudostatic. The tailings embankment is designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event. The embankment stability analyses exceeded both static and pseudo-static CDA guidelines.

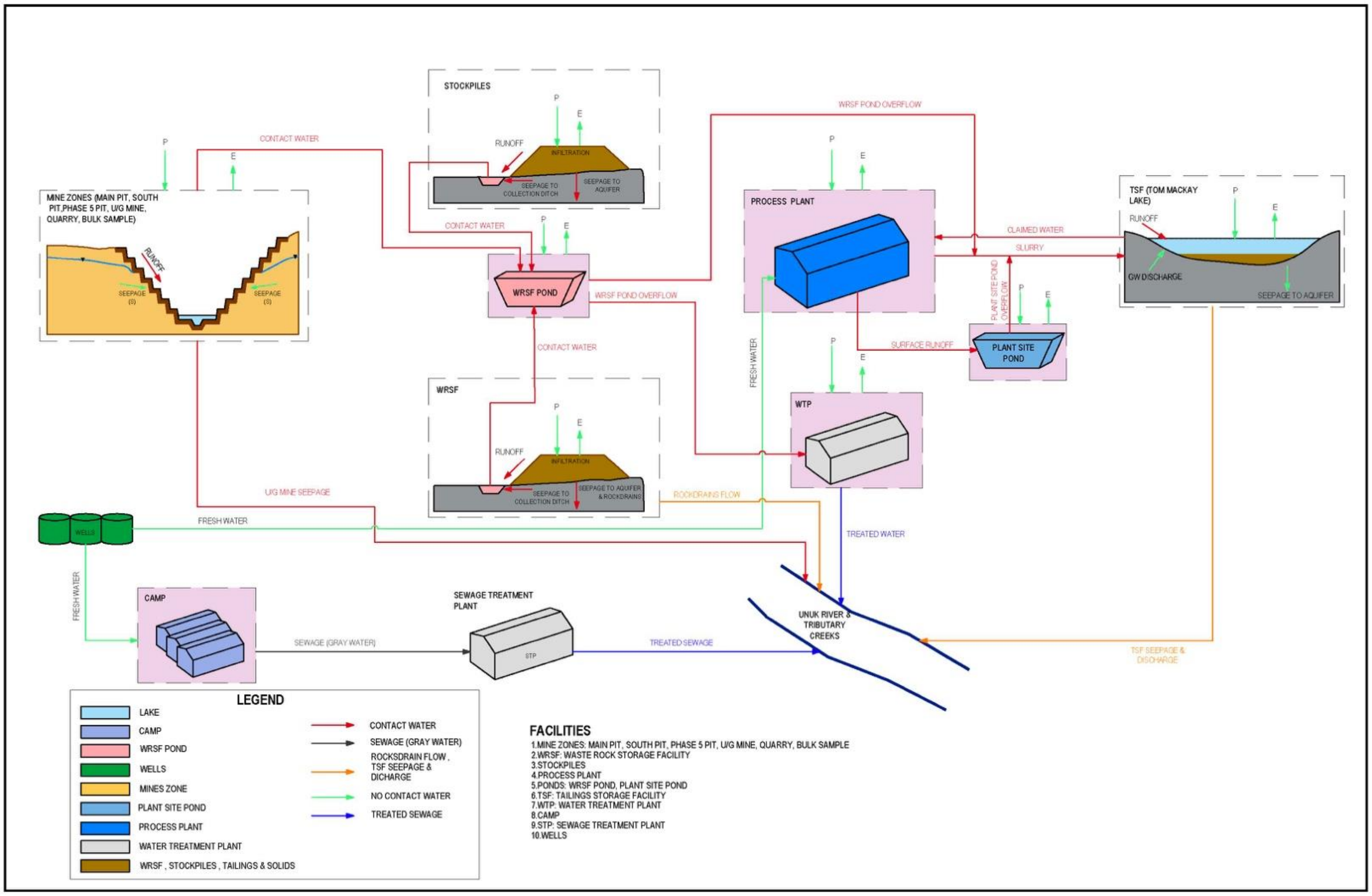
18.5.8 Tom MacKay Storage Facility Closure

TMSF closure will consist of removing the tailings discharge line and barge, process water pipeline, the pit dewatering pipeline and the reclaim of any road not required for post closure monitoring. Since the tailings and PAG waste rock will remain subaqueous, there is no cover system planned. Surface runoff from the TMSF watershed will flow through the permanent spillway to provide a minimum 6 m water cover over the waste materials. Ausenco performed a water balance to look at the effects of extreme climate events, especially droughts. The results showed that the facility would maintain 6 m of water cover even during extreme drought conditions.

18.6 Water Balance

A site-wide water balance (GoldSim) was developed based on the conceptual model shown in Figure 18-6. The GoldSim model was used to inform water management and predict the potential contact water volumes through the life of mine.

Figure 18-6: Water Balance Schematic



Note: Figure prepared by Hemmera, 2021.

The industrial water requirements will come from the TMSF, which are estimated to be 113 L/s to be used in mineral processing. Fresh/fire water will be pumped from a local fresh water supply well into a fresh/fire water tank.

The planned camp will be supplied for all its water needs from a local well. It is estimated that the average consumption of water, based on the size of the camp, is 1 L/s. Any effluent coming from the camp will be treated and discharged into the TSF.

No diversion works are anticipated. There will be inflow of water into the TMSF from direct rainfall and snow runoff from the surrounding catchment into the TMSF.

Average wet and storm rainfall conditions were modelled conveying the pit water to the TMSF. The key findings include:

- Maximum contact water flows occur in June during freshet whereas minimum contact water flows occur in the winter months (November through March);

- Maximum TSF discharge rate (3 m^3/s via the penstock) is required to maintain water levels in the TSF in every year from 2024 to 2035. No spillage from the TSF is predicted while the penstock is being employed;

- Pit water will be sent directly to a water treatment plant (WTP), then to D7 polishing ponds, and finally to Ketchum Creek during pre-production. The water treatment plant's maximum capacity has been designed to accommodate the pit water with additional treatment capacity. The WTP has a capacity of approximately 150 L/s, which supports pre-production operations;

- Once the tailings pipeline is installed and operations begin, pit water will report to the tailings mixing tank at the plant and sent with the tailings in the tailings transportation pipeline to the TMSF. As the open pit becomes larger, pit dewatering flow rates will increase. The pit water sent to the TMSF with the tailings stream ranges: Year 1 - 22.7 to 77 l/s in an average year to 24.5 to 139.3 l/s in a wet year and Year 11 – 65.6 to 210.1 l/s in an average year to 73.3 to 376.3 l/s in a wet year; and

- The WDW water management includes both contact and non-contact water management structures. The facility is located in a relatively small watershed. The non-contact water will pass underneath the facility in a rock drain that converts to 2 solid wall HDPE pipes that discharge water directly into Tom MacKay Creek. The surface contact water from the WRSF will be conveyed in both temporary and permanent diversion channel to contact water 5 Pond to remove sediment 10 microns and above prior to releasing water into Tom MacKay Creek. The contact water management system was designed for 1:200 year event and the non-contact water management system for 1:475 year event.

18.7 Site Infrastructure

18.7.1 Mine Facilities

The key facilities required in support of the mining operation include:

- Administration offices for the G&A staff and the Owner's mining staff:
  - Mine office/dry
  - Main administration building
  - Planning & exploration
- Truckshop, warehouse and truck wash sized for 144 t haul trucks and lighter vehicles;
- Diesel storage and distribution;
- Propane storage and distribution;
- Miscellaneous facilities: gatehouse, ready line, incinerator, tire change, truck scale, Class III landfill, and explosives storage facility.

18.7.2 Process Facilities

The key facilities required in support of the process operation include:

- Process plant and crushing facility;
- Assay laboratory;
- Process plant warehouse/workshop.

18.8 Camps and Accommodations

The permanent camp will be housed in portable modular units comprising 260 individual dormitories. In addition to the dormitories area, the camp will have a kitchen/dining area, recreation room, a boot/jacket room for personnel to enter and leave accommodations, and security/medical facilities.

There will be a temporary camp for accommodating construction crew comprising 40 individual-type dormitories. The temporary camp will be located on the wings of the permanent camp.

Each of the permanent and temporary camps will need to be multiple-level type to minimize footprints. Both will be heated with propane and will be connected with portable generator(s) for emergency power supply.

18.9 Power and Electrical

Project power will be provided through a 20 km long 69 kV overhead transmission line. The source of power will be from the Volcano Creek 287 kV substation. The estimated power demand for the Project is 21 MW.

At the mine site location, a 69 kV/13.8 kV transformer will step down the transmission voltage for utilization. A main substation building will distribute power from a 13.8kV switchgear to various locations of the mine site including process and infrastructure areas.

An electrical building will be installed in the process area to feed all process related equipment. Step down transformers at this building will provide power supply to 4160 V and 600 V loads.

Power feeders to infrastructure areas such as the mine infrastructure area, office buildings, accommodation camp, gate house, water supply wells, and the tailings water return barge will be provided through 13.8 kV overhead distribution lines with step down transformers at each location.

18.10 Water Supply

The fresh water supply for the site will be from a local well field located South-West of the process plant facility. The nominal supply flowrate will be 113 m3/h. Vertical pumps within the wells will pump the fresh water, via a 700 m long 100 mm nominal bore HDPE pipeline, directly to the freshwater storage tank within the process plant.

Freshwater will be distributed from the freshwater storage tank for the following requirements:

- Freshwater requirements within the process plant (including reagent mixing);
- Fire water requirements for all infrastructure, the fire water storage capacity will be within the fresh water tank;
- Gland water requirement for pump glands within the process plant;
- Potable water requirements for the camp and other infrastructure (including plant safety showers), this will be via the potable water treatment plant and potable water tank;
- Process water make-up requirements as necessary, this is a direct feed to the process water tank.

18.11 QP Comments on "Item 18: Project Infrastructure"

The project as envisaged in the PFS will use conventional infrastructure and construction methods. Electrical requirements will be sourced from existing hydroelectric facilities.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The proposed Eskay Creek operation will produce a gold concentrate on site, which will then be shipped to an out-of-province processing facility. There is currently no contract in place with any smelter or buyer for the concentrate.

19.2 Markets

The proposed concentrate is a complex gold concentrate with a relatively low gold content and elevated levels of arsenic, mercury, and antimony. Deleterious element assays are notably elevated in the first few years of the planned mine life (arsenic in Years 1 and 2, and mercury in Years 1 to 3) before dropping to values that fall within typical industry expectations. Given the complexity of the Eskay Creek concentrate, combined with the historical production of relatively difficult-to-market concentrates from the mine during its previous operational period, two independent, preliminary market studies were completed to support the NSRs used in the 2021 PFS.

Concentrate quality parameters are based on the results of ICP analysis of gold–silver concentrates produced during the variability flotation testwork at BaseMet (described in Section 13). Based on the available concentrate analyses, the 2021 PFS considers the concentrates will likely be sent to an Asian port for smelting and refining.

Ausenco provided the expected concentrate composition and tonnage at Eskay Creek to four concentrate marketing specialists (WoodMac, Open Minerals, Hartree, and Trafigura). Final concentrate grades in the variability testwork ranged from 12–90 g/t Au, averaging 38.1 g/t Au. To assess variations in NSR, two final concentrate gold grades (25 and 45 g/t Au) were selected and quoted by each researcher. The pros and cons of selecting the two concentrate grades are summarized in Table 19-1. Gold smelter contract estimates were received from all four marketing specialists and one copper smelter contract estimate was provided by WoodMac.

Table 19-1: 25 Gold g/t vs 45 Gold g/t Concentrate Grade Comparison

25 g/t Gold Concentrate Grade		45 g/t Gold Concentrate Grade	
Pros	Cons	Pros	Cons
• Higher metal recovery • Lower penalty grades	• Higher transport costs • Lower NSR	• Lower tonnage leading to lower overall, transport and treatment costs • Higher NSR	• Lower metal recovery • High penalty grades

The results of the marketing studies include:

- Producing higher-grade concentrates, where feasible, produces better NPVs for most of the variability tests;
- High mercury and arsenic concentrates will need to be sold as blending material for cleaner concentrates. Selling the concentrate in small volumes will help reduce deductions from penalty minerals;

  - Sales in the first three years will be the most challenging when the penalty grades are highest;

- The most likely market for the concentrate is China, where the material will be imported as a gold concentrate (exceeding the minimum gold content criterion) and will therefore not be subject to arsenic import limits that would be imposed on base metal concentrate imports. The Chinese market offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate;

- Chinese gold smelters can typically monetize antimony at the levels found in the Eskay Creek concentrates;

- Chinese copper smelters are a possible secondary outlet, but only in the later years when the deleterious elements have decreased;

  - European and Japanese copper smelters are unlikely able to take the concentrate due to the high mercury and antimony grades;

- Better NSRs are expected from copper smelters. However, there is limited confidence in whether significant sales to copper smelters will be possible;

- An additional minimum Chinese 13% value-added tax (VAT) is expected if gold grades are lower than 20 g/t and would significantly decrease its marketability.

Based on the predicted analysis, the Eskay Creek concentrates will be readily saleable. The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across several buyers since individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable effluent limits. An alternative option is to sell the concentrate to traders who may be able to buy all concentrate and spread distribution across a range of end customers, potentially including a mix of gold and copper smelters. Expectations of NSR may be achieved and penalties for deleterious elements may be minimized.

19.3 Contracts

No contracts have been concluded at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

Most concentrate is traded on the basis of term contracts. These contracts frequently run for terms of 1–10 years, although many long-term contracts are treated as evergreen arrangements, which continue indefinitely with periodic renegotiation of key terms and conditions. In other words, a term contract is a frame agreement under which a specified tonnage of material is shipped from mine to smelter, with charges renegotiated at regular intervals (typically annually).

Spot contracts are normally a one-off sale of a specific quantity of concentrate with a merchant or smelter. The material is paid for in much the same way as a concentrate shipped under a term contract. Merchant business is a mixture of one-off contracts with smelters and long-term contracts with both miners and smelters.

Often terms of sale for a term contract between miners and smelters are at "benchmark terms", which is the consensus of annual terms for the sale of concentrate and negotiated annually. Spot sales are made at spot terms and negotiated on a contract-by-contract basis.

19.4 Smelter Terms Assumptions

Concentrate grades for gold, silver, mercury, antimony, and arsenic are expected to vary throughout the life of mine which will impact the marketability and net revenue. The contract terms for the study, the terms from the equity researchers for a variable range of Au concentrate g/t cases are compared in Table 19-2.

Table 19-2: **Payabilities Contract**

Item	Info	Units	Payabilities Contract
Gold Payable(%)Gold	Concentrate Grade Ranges	%	
	35–40 g/t	%	79.5
	40–45 g/t	%	83
	45–50 g/t	%	85.5
	50–55 g/t	%	87.5
	55–60 g/t	%	89
	60–65 g/t	%	90
	65–70 g/t	%	91
	70+ g/t	%	92.5
Silver Payable (%) t	Concentrate Grade Ranges	%	
	200–500 g/t	%	75
	500–1,000 g/t	%	80
	1,000–1,500 g/t	%	82.5
	1,500–2,000 g/t	%	85
	2,000–3,000 g/t	%	87.5
	3,000+ g/t	%	90
Deductions Gold	–	g/t	0
Deductions Ag	–	g/t	0
Treatment Charges	–	$/dmt	0
Recovery Charges Gold	–	($/oz payable)	0
Recovery Charges Ag	–	($/oz payable)	0
Hg	Free limit	g/t	400
	Per 1 g/t over	USD/t	0.09
Sb	Free limit	g/t	20,000
	Per 100 g/t over	USD/t	0.02
As	Free limit	%	0.5
	Per 0.1% over	USD/t	3.75

19.5 Transportation and Logistics

Concentrate volumes are expected to decrease over the mine life as the feed grade decreases, as presented in Table 19-3. This results in an easier blending of the deleterious elements out of the concentrate over time.

Table 19-3: Concentrate Volumes by Year

Mine Year	Yr- 1	Yr -2	Yr -3	Yr -4	Yr- 5	Yr- 6	Yr- 7	Yr -8	Yr- 9
Total Feed (Mt)	2.3	2.5	2.5	2.5	2.5	2.5	2.5	2.5	1.9
Au (g/t)	4.29	3.87	3.30	3.09	3.63	2.96	2.66	2.22	2.38
Conc t (kt/a)	372	356	304	278	334	266	226	178	190

Ausenco conducted a concentrate logistics trade-off study to assess the transportation costs for the 2021 PFS. The base case for logistics is moving the concentrate by bulk bags to Stewart, where they will be loaded into containers for export via container vessels to China. The estimated transport costs for 72.3 t GVW trucks are shown in Table 19-4.

Table 19-4: Eskay Creek Transport Costs

Description	72.3 t GVW			
	Unit Rate	Units	Quantity	Container
Bulk Container Annual Lease	$1,800	ea	960	$1,730,000
Container Revolver Annual Lease	$90,000	ea	2	$180,000
Trucking to Stewart	$ 61.4	t	371,657	$22,830,000
Containerized Bulk Loading	$15.7	t	371,657	$5,820,000
Ocean Freight Bulk	$54.8	t	371,657	$20,370,000
Total Annual Logistics Cost		$t/a		$50,930,000
Cost per Tonne		$		$137

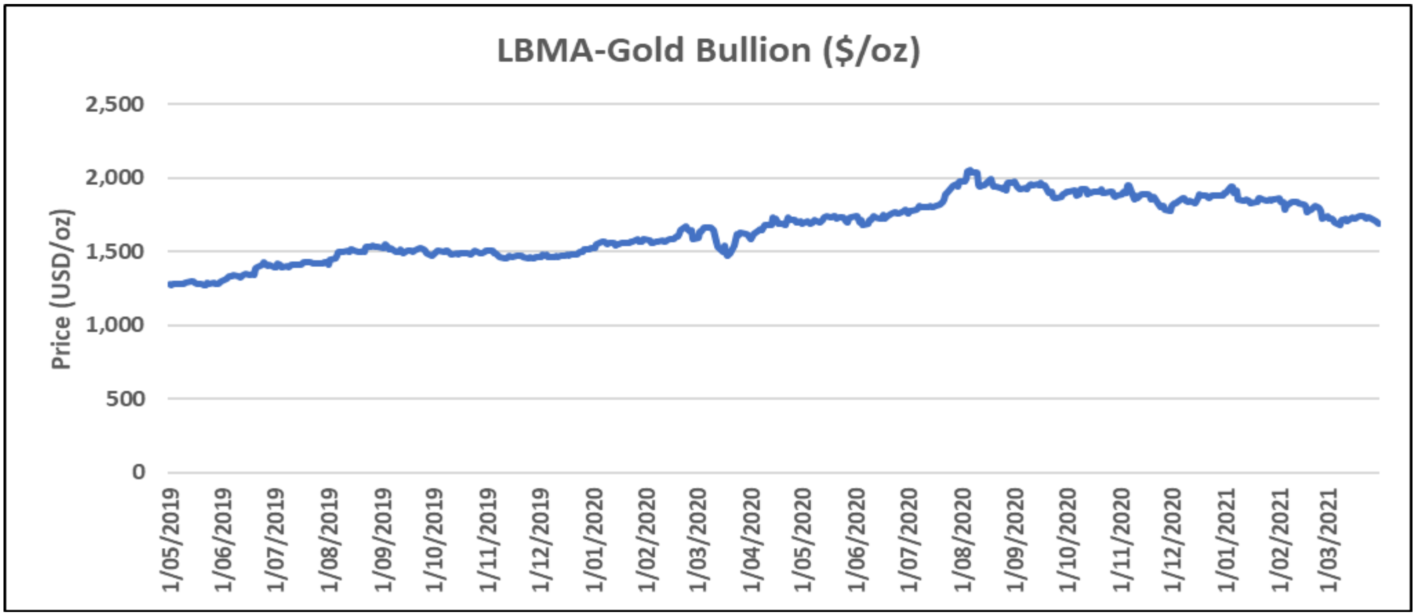
19.6 Insurance, Representation and Marketing

No allowance has been made for insurance, marketing or representation.

19.7 Metal Prices

The economic analysis included in the 2021 PFS is based on a two-year average of gold and silver prices as of March 31, 2021. The estimated prices are based on the daily closing price of gold and silver prices from the LBMA (Figure 19-1 and Figure 19-2).

Figure 19-1: LBMA Gold Price (2-year span, US$/oz)



Note: Figure from S&P Market Intelligence, March, 2021.

Figure 19-2: LBMA Silver Price (2-year span, US$/oz)

Note: LBMA Silver Price (2-year span, US$/oz)

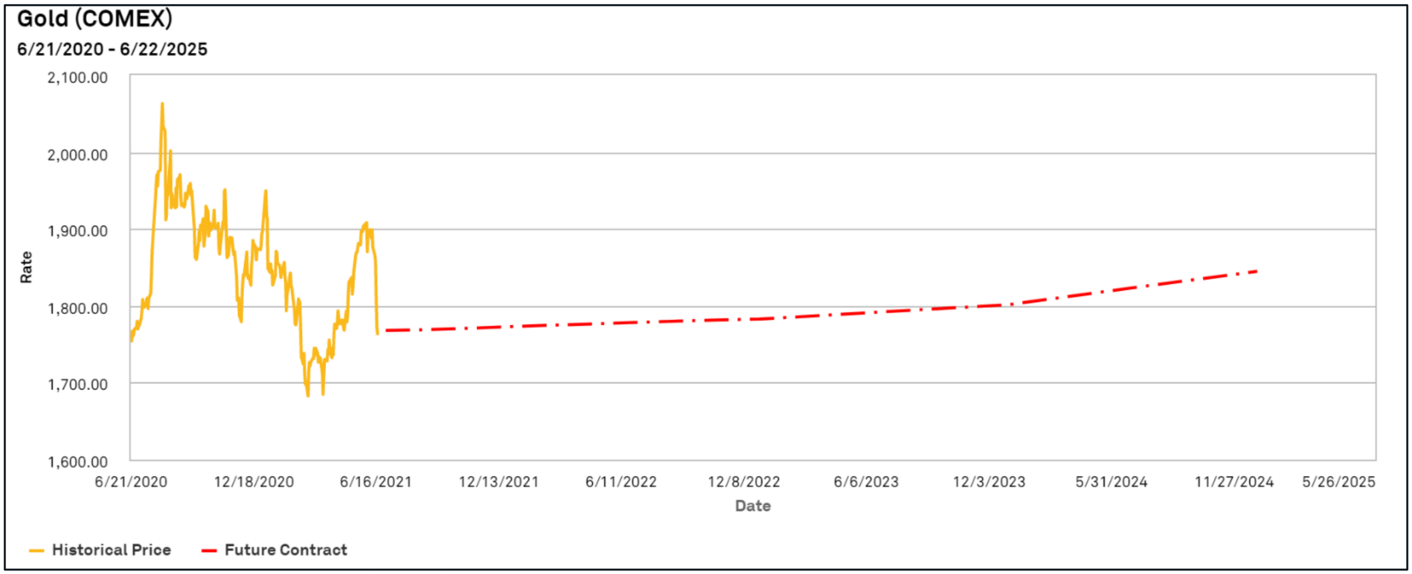
Although recent feasibility or pre-feasibility studies that are publicly available use consensus estimates for gold and silver prices, which are higher, the two-year average reflects a more conservative approach. The gold and silver prices used in the economic analysis are as follows:

- Gold price: US$1,658.48/oz;
- Silver price: US$19.67/oz.

19.7.1 Metal Price Forecasting

With production at Eskay Creek expected to commence within four years, a review of copper and gold future contracts traded on the COMEX market over a five-year period (see Figure 19-3), subsequently reflects significantly higher gold prices ranging between $1,750 and $1,850/oz Au when compared to the pricing used in the base case scenario.

Figure 19-3: Historical Gold Price and Futures Gold Contracts on the COMEX Market



Note: Data from Commodity Exchange (COMEX). June 2021.

Ausenco and Skeena established metal price projections for use in the 2021 PFS, which this Report is based on. The projections incorporate consideration of the recent metal market information combined with two-year trailing metal prices, and the COMEX forecasts.

- Gold: US$1,550/oz;
- Silver: US$22.00/oz.

An exchange rate of 0.794:1 US$:C$ was used.

19.8 Comments on "Item 19: Market Studies and Contracts"

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses to support the 2021 PFS.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Settings

Information on the project climate and physiographic setting is included in Section 5.

20.1.1 Vegetation

Based on the FLNRORD 2018 biogeoclimatic zone maps, the Project area is represented by three biogeoclimatic zones:

- The Engelmann spruce–subalpine fir zone occurs in the planned mine site area and Tom MacKay Creek, lower Argillite Creek, and upper Eskay Creek watersheds. It includes continuous forest cover at its lower and middle elevations and subalpine parkland near its upper limits. Engelmann spruce dominates the canopy of mature stands, while subalpine fir is most abundant in the understorey (Meidinger and Pojar, 1991);
- The Mountain hemlock zone occurs in subalpine areas west and southwest of the mine site area. The major tree species include mountain hemlock, subalpine fir with Sitka spruce, and western hemlock occurring at lower elevations (Hallam Knight Piésold Ltd, 1993);
- The Interior cedar hemlock zone occurs in valley bottoms and low-elevation uplands along Iskut River and Forest Kerr Creek. Vegetation is dominated by black cottonwood with Sitka spruce and birch present in lesser numbers (Hallam Knight Piésold Ltd, 1993).

20.1.2 Wildlife

Large wildlife species recorded within the Project area include grizzly bear, black bear, and mountain goat. Small mammals present in the project vicinity include American marten, wolverine, voles, and the hoary marmot. Furbearing mammals with suitable habitat in the project area include grizzly bear, wolf, lynx, ermine, mink, fisher, least weasel, and snowshoe hare (Hallam Knight Piésold Ltd, 1993).

Biophysical inventory mapping identified the project area as potentially suitable to support woodland caribou and moose (MOE, 1982). However, the Project area is not overlapped by any caribou herd ranges shown on provincial range mapping (BC, 2019).

Mid and lower elevations provide habitat for porcupine, northern flying squirrel and red squirrel. Plovers, Canada goose, harlequin duck, and numerous passerine species have been recorded in the area. Raptors recorded in the area include bald eagle, sharp-shinned hawk, and owls.

Four species of amphibian and one reptile species are known to inhabit the project area. They include the common garter snake, long-toed salamander, western toad, wood frog and rough-skinned newt. Wood frogs are the only amphibian recorded near the project area (Hallam Knight Piésold Ltd, 1993).

Wildlife studies are ongoing for the Project. Approximately 43 species of conservation concern may occur in the region but have not necessarily been confirmed to occur within the Project study area (RTEC, 2021).

Wildlife studies are ongoing for the Project. Approximately 43 species of conservation concern may occur in the region but have not necessarily been confirmed to occur within the Project study area (RTEC, 2021).

20.1.3 Fisheries and Aquatic Resources

Fisheries resources of rivers, lakes and their tributaries potentially affected by the Eskay Creek mine, were assessed from 1982 to 1993 (Hemmera, 2000) and again in 2020 (RTEC, 2021). No fish have been observed or captured in the upper tributaries of the Unuk River in the vicinity of the Project, in headwater lakes including Albino Lake, Little Tom Mackay Lake, and the TMSF, nor in the Eskay Creek and Tom Mackay Creek downstream of the former mine. The high-alpine, natural lakes and streams in the Tom MacKay watershed, including Albino Lake and TMSF, are naturally low in plant nutrients and barren of fish due to impassible waterfalls as well as gradient/velocity barriers to approximately 10 km downstream of the former Eskay Creek mine site (McGurk et al., 2006; Hallam Knight Piésold Ltd, 1993).

A series of obstacles to fish passage are also located immediately upstream of the confluence of Tom MacKay Creek with Ketchum Creek. Pink, chum, chinook, and sockeye salmon, Dolly Varden, and cutthroat trout, were observed 7–8 km downstream of the planned mine site in the Unuk River (Hemmera, 1997).

20.1.4 Environmental Studies

Several environmental studies were completed at the Eskay Creek mine under various owners throughout the mine life. Key reports reviewed are discussed in this sub-section. The environmental baseline data were mostly collected between 1990 and 1993 by Hallam-Knight Piésold for Prime Resources Ltd to support their application for a Mine Development Certificate. Updates to the baseline studies were made in 1997 to support the proposed mill installation at the mine site (Hemmera, 1997), and again in 2000 to apply for a separate Environmental Assessment (EA) certificate to deposit tailings and waste rock in the Tom MacKay Storage Facility (TMSF) (Hemmera, 2000). Environmental monitoring was also completed during and after operations.

In 2020, Skeena began additional environmental, social, economic, heritage and health baseline studies to reflect current environmental and social conditions. These studies will help refine the project design and support applications for provincial and federal submissions.

20.2 Environmental Management

20.2.1 Historical Waste Disposal Activities

Waste rock was stored underwater at the permitted Albino SF from 1994 onwards. No surface waste rock storage facilities were developed. In late 1997, the processing plant was permitted, constructed, and began operations. The filtered tailings generated from the mill were initially trucked to the Albino SF along with the waste rock until 2001 (Barrick, 2014a).

From September 2001 to the end of operations in 2008, slurry tailings were discharged into TMSF via a dedicated pipeline while waste rock continued to be stored in the Albino SF (Barrick, 2014b). A small percentage of slurry tailings were trucked to the Albino SF during maintenance or other events that restricted normal pipeline discharge to the TMSF.

Throughout the mining operation, water exiting the underground workings underwent water treatment using chemical additives and a series of settling ponds prior to discharge at the pond outlet (permitted D7 discharge point) which flowed 600 m downhill into Ketchum Creek. The sludge from the water treatment ponds was also disposed into Albino Lake. Additionally, a landfill was utilized for non-hazardous industrial waste (URS, 2005).

Significant reclamation activities started in 2007; activities included removal of surface buildings including the mill, concrete pads and decommissioning of the tailings pipeline. Details of the reclamation activities undertaken to date are included in annual reclamation reporting (Barrick, 2019). The Eskay Creek Mine has been in care and maintenance since mining operations ceased in 2008, with ongoing site management. Under the federal Metal Mine Effluent Regulations (MMER; currently Metal and Diamond Mining Effluent Regulations, MDMER), the mine status is a "Recognized Closed Mine" since 2011 (Barrick, 2019).

20.2.2 Waste Management – Waste Rock and Tailings Disposal

The main waste management issue for the project is the prevention and control of metal leaching/acid rock drainage (ML/ARD) from the tailings and waste rock.

The Project will create waste rock from mine development and tailings as a by-product of mineral processing. The waste streams will be managed on site as follows:

- NAG waste rock will be deposited in two locations: approximately 90% will be stored in the WDW facility that will be located to the west of the north pit. The remaining 10% of the total waste rock will be backfilled in the north pit;
- PAG waste rock will be deposited in the TMSF with a water cover (refer to discussion in Section 18.5);
- Tailings will be deposited sub-aqueously in the TMSF with a water cover (refer to discussion in Section 18.5). The TMSF is already permitted for tailings disposal from the former underground mine.

During the operations, a kinetic testing program for underground mine waste rock was initiated in August 2006 to supplement the existing static testing data to quantify the ML/ARD potential. Additional kinetic testing was undertaken in January 2008 for waste rock samples from the Albino SF which accepted both waste rock and tailings. The objectives of the 2008 program were to determine the subaerial and subaqueous weathering characteristics of rocks present at the site. Specific characterizations included rates of sulphide oxidation, neutralization potential consumption, and metal release to help estimate the time to sulphide and neutralization potential exhaustion, future geochemical conditions of the rock, and prediction of mine water quality. Selection of mine rock samples for testing was based upon a review of acid base accounting and metal analyses for more than 1,000 samples from Eskay Creek mine that were collected during operations.

Mine rock samples collected from the 2006 and 2008 programs were subjected to kinetic testing that consisted of:

- Humidity cell testing;
- Subaqueous column testing;
- Column testing for two backfill samples.
- Receiving environment monitoring of the drainage from the Albino SF and the TMSF has not shown any significant ML/ARD-related changes in water quality. Waste disposal at the end of operations required additional water quality

monitoring and management of the accumulated waste rock to achieve full subaqueous disposal by 2010. The drainage from Albino SF and TMSF had water quality within the applicable permit limits since 2010.

- In 2020, a geochemical study was initiated on new waste rock, ore, tailings and overburden sources for the Project together with the existing tailings in TMSF. The purpose of this study was to update and inform waste management decisions for the Project design. The following analyses were undertaken and are ongoing:

- Acid base accounting on new waste rock, historic tailings from TMSF, new tailings, ore streams and borrow material;

- Humidity cell testing on new waste rock, historic tailings from TMSF, new tailings and ore streams;

- Subaqueous column testing on new waste rock, historic tailings from TMSF and new tailings;

- Field leachate barrels on new waste rock.

To manage the potential for ML/ARD, Skeena has incorporated design features and mitigation measures that are consistent with best practices for waste and water management, including:

- WRSF seepage collection systems;

- Water treatment plant;

- Subaqueous disposal of PAG tailings.

20.2.2.1 Non-Hazardous Waste

Non-hazardous waste management will involve the segregation of industrial and domestic waste into streams. Waste collection and disposal facilities will include one or more incinerators for domestic/putrescible waste, separate waste collection areas for recyclable and industrial waste streams for off-site disposal, and sewage effluent and sludge disposal for onsite disposal. Waste collection areas will be managed following regulatory requirements and best management practices for the safety of workers and environment, including standard operating procedures for spill management, fire safety and wildlife attractant.

20.2.2.2 Hazardous Waste

Hazardous waste materials such as spoiled reagents, waste petroleum products and used batteries will be generated throughout the life of the project, from construction to decommissioning. Storage facilities will facilitate the segregation and inventory of the various hazardous waste streams generated during the project. A separate secure storage area will be established with controls and best management practices to maintain the safety of workers and the environment. Hazardous materials will be labelled and stored in appropriate containers for shipment to approved off-site disposal facilities. Waste streams will be tracked in accordance with federal and provincial regulations, such as the federal *Transportation of Dangerous Goods Act, 1992* (SC 1992, c 34).

20.2.3 Water Management

Mine water can be divided into two categories depending on the potential for contamination:

- Non-contact water from upstream catchments that has not been in contact with mine workings and surface infrastructure will be kept from water which will come into contact with mine workings and surface infrastructure. Non-contact water will be diverted around the mine site as much as possible;

- Contact water will interact with potential sources of contamination including seepage from the WRSF, temporary stockpiles, process water, infrastructure surface runoff, and pit dewatering. Contact water will be collected and if required, treated to meet permit discharge limits prior to discharge. Process water will be discharged to the TMSF.

A conceptual site-wide water balance is included in Section 18.6. Water management will include collecting surface water from disturbed areas (mine-contact) to:

- Minimize surface erosion interactions with water;

- Recycle mine-contact water whenever possible;

- Treat mine-contact water as required;

- Monitor water quality to meet discharge standards prior to discharge.

20.3 Site Management and Monitoring

The Project will be designed, constructed, operated, and decommissioned to meet all applicable provincial and federal environmental and safety standards, regulations, and permit conditions. Skeena will implement an Environmental Management System (EMS) in advance of construction that defines the processes, resources, responsibilities, and specific management plans to ensure compliance. The draft EMS will be developed during the permitting process and include ongoing monitoring, management steps, and reporting to relevant parties.

Site water management will be a critical component of Project design, execution, operation, and closure. To mitigate the potential contamination of water from a variety of sources (air, land, and process), Skeena will develop a Water Management Plan and Dust Control Management Plan that applies to all activities.

20.4 Closure Plan

In summary, the mine closure strategy for the mine will be to have a stable, revegetated site with best mitigation of potential ML/ARD and water quality risks that is consistent with the Tahltan and Skeena's agreed Social and Environmental Design Principles and post-mining end land uses. A Closure and Reclamation Plan will be developed as during the permitting process to achieve end land use objectives (e.g. wildlife habitat, in consideration of Indigenous interests. Closure planning will include Indigenous groups and stakeholders to determine post-mining land use objectives and supporting strategies, including addressing regulatory requirements. Achieving the desired outcomes will be an iterative process during the design and permitting process and incorporate social, environmental, engineering, technical, and Tahltan criteria.

Closure activities may include:

- Decommissioning of all surface workings, with the exception of those required for long term monitoring, such as site access road, water management structures, transmission line, environmental monitoring installations, and TMSF embankments;
- Establish stable water conveyance structures to mitigate long-term erosion and stability concerns, and develop post-closure tunnel reclamation;
- Maintaining water cover of PAG waste rock and tailings in TMSF to meet water quality objectives without ongoing treatment for ARD;
- Development of a pit lake to mitigate ML/ARD risk from pit walls;
- Potential for water treatment of pit and waste rock storage seepage and runoff to meet discharge requirements;
- Backfilling, resloping, scarifying, and revegetation of decommissioned areas to perpetuate a long-term revegetated state;
- Implementing and maintaining a long-term monitoring plan.

Closure activities will be completed progressively throughout mine operations as guided by the reclamation plan.

In accordance with the Mines Act permit, mine closure, reclamation and post-closure costs are updated every 5 years to reflect the current liability, and to inform the establishment of a reclamation security bond. The estimated closure and reclamation costs are included in the economic analysis in Section 22.

20.5 Permitting

20.5.1 Environmental Approvals

Proposed mining projects that exceed thresholds are governed by environmental legislation and must undergo an EA process provincially to amend or issue an EA Certificate (EAC), and often a concurrent federal Impact Assessment (IA) and federal decision. Once approved, a proposed mining project will be issued an EAC, which enables subsequent construction and operational permits to be issued and executed. The EA process is a means of addressing the potential for adverse environmental, social, economic, health, and heritage effects or the potential adverse effects on Indigenous interests or rights prior to project approval.

At a provincial level, proposed mining developments that exceed any of the thresholds specified in the Reviewable Projects Regulation (BC Reg. 370/2002), are required under the BCEAA to obtain an EA Certificate (EAC) before the issuance of any permits to construct or operate. The project will require a provincial EAC.

At a federal level, proposed mining developments that exceed any of the thresholds specified in the Regulations Designating Physical Activities (SOR/2012-147), are required under the Impact Assessment Agency of Canada's (IAAC) Impact Assessment Act (IAA) to obtain a federal decision statement before the issuance of any permits to construct or operate. The project will require a federal decision statement.

The proposed Project is anticipated to undergo a concurrent EA/IA, called a substituted process, under federal and provincial regulations. Since the Eskay Creek Mine has two existing Certificates, one or both will be amended through a substituted EA/IA process.

The Eskay Creek Mine went through two EA processes in its history. An application for a Mine Development Certificate (MDC) was approved in 1994 and MDC was issued under previous environmental review legislation and is considered equivalent to an EAC under present legislation. In 2000, an application for an EAC was reviewed and a Project Approval Certificate was approved for disposal of mine tailings into TMSF and is also equivalent to a present day EAC.

The 1993 MDC enabled the proponent to obtain construction/operation permits to build the Eskay Creek Mine, including underground mining, surface workings, and use of Albino Lake as a waste rock storage facility and offsite shipping of ore. In 1997, permits were amended to build a mill onsite and dispose of tailings with waste rock to Albino Lake. Once the Project Approval Certificate was issued in 2000 for the use of Tom MacKay Lake as a tailings disposal facility, construction and operation permits were obtained.

For the proposed Project, Skeena will undertake a substituted process to amend an existing EAC or obtain a new EAC. The process to follow for the EA/IA is being developed with the provincial and federal regulators, the Tahltan Nation and Skeena, based upon the legislative steps, criteria and procedures.

No technical or policy issues are anticipated for obtaining the required Project permits and approvals, given its long mining history.

20.5.2 Anticipated Provincial Permits and Authorizations

In addition to obtaining the EAC, the Project will require permits and authorizations in accordance with provincial and federal legislation and regulations prior to construction. No permits for project commercial development will be issued before an EAC is obtained. Consequently, Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction.

Table 20-1 presents a preliminary list of the key provincial authorizations, licences, and permits that will be required to develop the Project.

20.5.3 Anticipated Federal Approvals and Authorizations

Table 20-2 presents a preliminary list of the key federal authorizations, licences, and permits required for project development.

Table 20-1: Summary of Provincial Permits, Licences and Approvals Possibly Required for the Project

Authorization	Responsible Agency	Legislation	Purpose
Amendment to Permit M-197	BC Ministry of Energy, Mines and Petroleum Resources (EMPR) (now BC Ministry of Energy, Mines and Low Carbon Innovation (EMLI)	*Mines Act*, Health, Safety and Reclamation Code for Mines in BC	Approve the new mine plan and reclamation program.
Water System Construction Permit Water System Operating Permit	Ministry of Health	*Drinking Water Protection Act*, Drinking Water Protection Regulation	Authorize construction and operation of potable water supply system for camp and process plant.
Food Facility - Health Approval Application	Ministry of Health	*Drinking Water Protection Act*	Approve opening and operation of food service facility
Sewage Registration *Environmental Management Act*	Ministry of Health	Sewage Registration	Authorize sewage treatment plant
Amendment to *Environmental Management Act* (Effluent) Permit 10818	BC Ministry of Environment and Climate Change Strategy (ENV)	*Environmental Management Act*,	Authorize discharges from sedimentation ponds, tailings storage facility, seepage
Environmental Management Act (Air) Permit 12977	ENV	*Environmental Management Act*	Authorize solid, air emissions and effluent discharges from incinerator and process plant
Hazardous Waste Registration	ENV	*Environmental Management Act* Hazardous Waste Regulation	Register hazardous waste transfer facility, plant truck shop
Fuel Storage Registration	ENV	*Environmental Management Act*	Authorize bulk fuel storage
Water Licence	ENV	*Water Sustainability Act*	Authorize storage, use or diversion of surface water or groundwater for one or more purposes.
Approval for Works in and about a Stream (Section 11)	ENV	*Water Sustainability Act*	Approve changes in or about a stream
Investigation or Inspection Permit	FLNRORD	*Heritage Conservation Act*, RSBC 1996, c. 187	Undertake archaeological impact assessment (AIA)
Site Alteration Permit	FLNRORD	*Heritage Conservation Act*	Required to alter an archaeological site (should any be identified and impacted by the Project)

Authorization	Responsible Agency	Legislation	Purpose
Occupant Licence to Cut	FLNRORD	*Forest Act*	Authorizes cutting and removal of timber on Crown land
Road Use Permit	FLNRORD	*Forest Act*	Authorizes use of existing Road

Table 20-2: Summary of Federal Permits, Licences and Approvals Possibly Required for the Project

Authorization	Responsible Agency	Legislation	Purpose
Explosives Permit	Natural Resources Canada	*Explosives Act*	Required to manufacture, store and use explosives
Fisheries Authorization	Fisheries and Oceans Canada	*Fisheries Act*	Required if the Project will result in the harmful alteration, disruption or destruction of fish habitat or death of fish
Metal and Diamond Mining Effluent Regulations (MDMER) Schedule 2 amendment	Environment & Climate Change Canada (ECCC)	*Fisheries Act*	Schedule 2 amendment may be required to amend the existing tailings impoundment sizes
Migratory Bird Permit	ECCC	*Migratory Birds Convention Act,*	Required if nesting habitats used by migratory birds might be impacted or if activities occur during the nesting season (e.g., clearing of vegetation)
Species at Risk Permit	ECCC	*Species at Risk Act*	Authorizes an activity affecting listed wildlife species, any part of its critical habitat or the residences of its individuals
Environmental Emergency Registration	ECCC	Environmental Emergency Regulations	Registers substances over specified volumes site must have suitable emergency response plan for the substances
Nuclear Safety Authorization	Canadian Nuclear Safety Commission	*Nuclear Safety and Control Act*	Required for possession of instruments containing radioactive material, such as nuclear density gauges (portable and fixed)
Radio Licence	Industry Canada	*Radio Communication Act*	Authorizes use of radio equipment on site.
Navigable Waters Approval	Transport Canada	*Canadian Navigable Waters Act*	Required for works that take place within navigable waters that do not meet works established under the Minor Works Order and which may interfere with navigation
Transportation of Dangerous Goods Permits	Transport Canada	*Transportation of Dangerous Goods Act*	Authorizes transportation and handling of dangerous goods

20.6 Considerations of Social and Community Impacts

20.6.1 Social Setting

The Project is located in the Regional District of Kitimat-Stikine (RDKS) which spans over 100,000 km in Northwestern BC. Approximately one-third of the 40,000 to 45,000 people in the region are Indigenous, which is higher than the provincial average (MSBED, 2005). The Project is within the territory of the Tahltan Nation and the asserted traditional territory of the Tsetsaut Skii km Lax Ha. The closest Indigenous community is the Tahltan community of Iskut (125 km north and 175 km via road). Other Tahltan communities are located north/northeast of the Project, and include Dease Lake (190 km northeast, 253 km via road) and Telegraph Creek (142 km north, 362 km via road). Stewart is the closest non-Indigenous community to the Project (83 km to the south; 261 km via road). Many of the smaller communities in RDKS have predominantly Indigenous populations that are isolated from one another as well as from the main regional centre of Terrace.

Land and resource use within the region include trapping, guided hunting, commercial recreation, and outdoor recreation including fishing, hunting, camping, hiking, snowmobiling, all-terrain vehicle (ATV) riding and skiing. In the vicinity of the Project, there are mineral, water and range tenures, guide outfitter and traplines. There are seasonal Tahltan cabins along the Eskay Mine Road.

Currently primary resources industries, mainly mining and forestry, comprise a key proportion of the larger regional (northwest and central BC) employment market at 4.6% and 2.6%, respectively. The forestry industry has been in decline in recent decades. Since the mid-1990s, the regional population has dropped by almost 15% although in recent years, the rate of decline has begun to slow (MSBED, 2005).

Community and socio-economic impacts of the Project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers, as well as local/northern businesses and contractors. Such opportunities could reduce and possibly reverse the out-migration to larger centres.

Estimates made in 1993 expected the historic Eskay Creek mine to generate approximate 2,250 person years of direct and indirect employment for BC residents; approximately 50% of these would be for residents of northern BC (Hallam Knight Piésold Ltd, 1993). These estimates are an indicator of the potential employment benefits to local communities in the Project area.

20.6.2 Engagement and Consultation

20.6.2.1 Consultation Policy Requirements

Provisions for consultation with Indigenous Nations and the public are a component of the provincial and federal legislation for both the EA processes and permitting activities. Skeena is developing an Engagement Plan for the Project as required by the provincial and federal EA processes. This plan provides a summary of Skeena engagement activities as well as serve as a guide for Skeena's engagement activities with identified Indigenous Nations and stakeholders throughout the EA process. The Engagement Plan will be submitted with the Initial Project Description to begin the EA process.

Ongoing and future engagement and consultation measures by Skeena are driven by best practices as well as Skeena's internal company policies. These measures will at a minimum comply with federal and provincial regulations.

20.6.2.2 Indigenous Nations

Skeena recognizes engagement and support of the Project from Indigenous Nations from initial project design until post-closure is critical for the success of the Project. Skeena is and will consult with local Indigenous Nations to gain that support, yet also recognizes this is part of the EA process at both the provincial and federal level. Engagement with local Indigenous Nations will continue throughout the Project design, construction, operations, closure, and post-closure.

The Project is located within the traditional territory of the Tahltan Nation and the asserted territory of the Tsetsaut Skii Km Lax Ha. The historical environmental process and subsequent expansions included consultation with the Iskut Band, Tahltan Band, and the Tahltan Central Government.

Project traffic will use Highways 37 and 37A which pass through the Nass Area and Nass Wildlife Area (as defined by the Nisga'a Final Agreement) and the traditional territory of the Gitanyow Nation.

20.6.2.3 Government

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will form a project specific working group at the early stages of the EA process, which will include representatives from many government groups. Skeena will consult with the working group on project-related developments during the EA process.

20.6.2.4 Public and Stakeholders

Skeena will consult with the public and relevant stakeholder groups, including tenure holders, businesses, economic development organizations, businesses, and contractors (e.g., suppliers and service providers), and special interest groups (e.g., environmental, labour, social, health, and recreation groups), as appropriate.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

LOM Project capital costs total $627.7M which can be broken down as follows:

- Initial capital cost: includes the costs required to construct all of the surface facilities, and open pit development to commence a 2.9Mtpa operation. The initial capital cost is estimated to be $487.9M
- Sustaining capital costs: include all the costs required to sustain operations, with the most significant component being open pit mine development. Sustaining capital costs total $47.4 M over the LOM; and,
- Closure costs: include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $92.4 M.

The following basic information pertains to the estimate:

- Base date is Q2, 2021;
- Expressed in Canadian dollars (CAD);
- Currency exchange rate USD 0.78 : CAD 1.00;
- This is a Class 4 estimate prepared in accordance with AACE International’s Cost Estimate Classification System. The accuracy range is -20% to +30%.

21.2 Capital Cost Estimates

21.2.1 Summary

Table 21-1 summarizes the capital cost estimate.

Table 21-1: Capital Cost Estimate Summary (CAD M)

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Mine			
Pre-stripping	88.2	0.0	88.2
Mining equipment	14.1	17.2	31.3
Mine infrastructure	4.0	18.1	22.1
Mine infrastructure (WRSF, waste management pond & channels, initial dewatering, water treatment plant, Truck Shop)	13.6	4.7	18.3

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Sub-total mine	*119.9*	*40.0*	*159.9*
	Processing		
Ore handling	17.4		17.4
Processing plant	97.4	1.3	98.7
Tailings and reclaim water	8.1	6.1	14.2
Onsite infrastructure	68.1		68.1
Sub-total processing	*191.0*	*7.4*	*198.4*
	Offsite Infrastructure		
Access road	4.3		4.3
Power supply	24.9		24.9
Sub-total offsite Infrastructure	*29.2*		*29.2*
Sub-total direct costs	*340.1*	*47.4*	*387.5*
Indirect Costs	68.0		68.0
Sub-total directs + indirect costs	*408.1*	*47.4*	*455.5*
Owner's costs	27.2		27.2
Total excluding contingency	*435.3*	*47.4*	*482.7*
Project contingency	52.6		52.6
Sub-total	*487.9*	*47.4*	*535.3*
Closure costs		92.4	92.4
Total	*487.9*	*139.8*	*627.7*

21.2.2 Mine Capital Costs

The mining capital cost estimate is grouped into three main categories:

- Pre-production stripping costs;
- Mining equipment capital;
- Miscellaneous mine capital;

The cost breakdown has been shown in Table 21-2.

Table 21-2: Mining Capital Cost Estimate (CAD M)

Mining Capital Category	Initial Cost ($ M)	Sustaining Cost ($ M)	Total Capital Cost ($ M)
Pre-production stripping	88.2	-	88.2
Mine equipment capital	14.1	17.2	31.3
Miscellaneous mine capital	4.0	18.1	22.1
Mine Infrastructure	13.6	4.7	18.3
Total	*119.9*	*40.0*	*159.9*

21.2.3 Pre-Production Stripping

Mining activity commences in advance of the process plant achieving commercial production. This includes the movement of 11.8 Mt of waste and placement of 0.6 Mt of mill feed in a stockpile adjacent to the primary crusher. The mine operating costs associated with this time period are included in the capital cost estimate and expected to cost $88.2 M. This cost covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs, and financing costs.

21.2.4 Mining Equipment Capital

The mining equipment capital costs reflect the use of financing of the major equipment and some support equipment. Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed. The remaining cost was included in operating costs (refer to Section 21.2.11).

The base costs provided by the vendors are included in a calculation for each unit cost calculation and options added to that. The capital cost, the cost of financing, and down payment, are shown in Table 21-3.

The cost of spare truck boxes, loader buckets and is included in the capital cost for the major equipment cost estimate, due to the remote nature of the mine.

The distribution of capital costs is completed using the number of units required within a period. If new or replacement units are needed, that number of units, by the unit cost (20% of that for major equipment) is applied to the capital cost in that period. There is no allowance for escalation in any of these costs

The balancing of equipment units based on operating hours is completed for each major piece of mine equipment. The smaller equipment was based on number of units required, based on operational experience. This includes such things as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc. For Eskay Creek, additional support equipment for snow removal and site water control was included to accommodate the expected climatic conditions which includes on average 13 m of snow.

The most significant piece of major mine equipment is the haulage trucks. At the peak of mining, twelve units are necessary to maintain mine production. This happens from Year 3 onwards. It should be noted that two different truck fleets are included in the estimate. The early works/pre-stripping period utilizes four smaller 91 t units with the appropriately sized loading fleet (11.5 m3 loaders) and drills (140 mm). When the mine starts production (Year 1) the transition to the larger 144 t trucks will occur and the larger drills (229 mm). The smaller trucks will be relegated to snow removal and tailings dam maintenance duties. The earlier loading fleet will be used at the primary crusher and stockpile management duties. The smaller drills will be used for pre-shear drilling, horizontal drain holes and backup drilling duties.

The maximum hours per truck/per year are set at 6,000. There are periods where the maximum hours per unit are below what the maximum possible can be. In those situations, increasing the maximum on the number of trucks still leaves residual hours required to complete the material movement, therefore, the number of total trucks is unchanged. In these cases, the hours required are distributed evenly across the number of trucks on site and available.

The other major mine equipment is determined in the same manner. Therefore, in some instances the loaders have a longer period of life (same number of hours between replacements) due to the sharing of hours with the other units in the fleet.

Table 21-3: Major Mine Equipment – Capital Cost, Full Finance Cost and Down Payment (C$)

Equipment	Unit	Capacity	Capital Cost ($)	Full Finance Cost ($)	Down Payment ($)
Production drill	mm	140	1,562,000	1,646,000	312,000
Production drill	mm	229	3,405,000	3,588,000,	681,000
Production/crusher loader	m^3	11.5	2,797,000	2,947,000	559,000
Production/crusher loader	m^3	23	7,489,000	7,890,000	1,498,000
Hydraulic shovel	m^3	22	9,322,000	9,822,000	1,864,000
Haulage truck	t	91	1,977,000	2,083,000	395,000
Haulage truck	t	144	3,473,000	3,659,000	695,000
Dragline (44 m boom)	m^3	3.8	3,685,000	3,883,000	737,000
Track dozer	kW	474	1,855,000	1,954,000	371,000
Grader	kW	163	357,000	376,000	71,000
Support excavator	m^3	6.7	2,100,000	2,213,000	420,000

The support equipment is usually replaced on a number of years of usage basis. For example, pickup trucks are replaced every three years, with the older units possibly being passed down to other departments on the mine site. However, for the purpose of the capital cost estimate, new units are considered for mine operations, engineering, and geology.

The number of pieces of major equipment required by year are shown in Table 21-4.

Table 21-4: Mine Equipment on Site

Equipment	Yr-3	Yr-2	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
Production drill(140mm)	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Drill (229mm)	-	-	2	3	3	3	3	3	3	3	3	3	3
Production loader(11.5m^3)	1	1	2	2	2	2	2	2	2	2	2	2	2
Production loader(23.5m^3)	-	-	-	1	1	1	1	1	1	1	1	1	1
Hydraulic shovel	-	1	2	2	2	2	2	2	2	2	2	2	2
Haulage truck (91t)	3	3	4	4	4	4	4	4	4	4	4	4	4
Haulage truck (144 t)	-	-	-	5	10	12	12	12	12	12	12	12	12
Track dozer	3	3	4	5	5	5	5	5	5	5	5	5	5
Grader	2	2	3	3	3	3	3	3	3	3	3	3	3

Equipment	Yr-3	Yr-2	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
Support excavator	1	1	1	1	1	1	1	1	1	1	1	1	1
Snow plow	2	3	3	3	3	3	3	3	3	3	3	3	3
Dragline	-	1	1	1	1	1	1	1	1	1	1	1	1

In the case of the production loader, there is one full time at the primary crusher when the plant commences operation. Its role is to tram material from stockpile and manage the blending of various mill feed types.

The support excavator is a larger unit meant to clean mill feed from previously-mined stopes and windrow the material for loading by either the hydraulic shovel or production loader. It is capable of loading the haul trucks, but is not expected to, as a result of the significant loading time that would result.

The expected equipment life is:

- Production drill: 25,000 hrs. (140 mm), 45,000 hrs (229 mm);
- Production loader: 35,000 hrs. (11.5 m3), 60,000 hrs (23.5 m3);
- Electric Hydraulic shovel: 72,000 hrs.;
- Haulage truck: 35,000 hrs. (91 t), 50,000 hrs. (144 t);
- Dragline: 10 years;
- Track dozer: 35,000 hrs.;
- Grader: 25,000 hrs.;
- Support excavator: 7 years;.

Other support equipment is normally determined in number of years and varies by its duty in the mine. Light plants for example are replaced each four years. The integrated tool carrier for site support is purchased once at the project start and is not replaced over the mine life.

21.2.5 Miscellaneous Mine Capital

The miscellaneous mine capital includes various separate line items in the costing. These are shown in Table 21-5.

The engineering office equipment includes such items as desktop computers, plotter, copies of the mining and geology software, and survey equipment with associated peripherals. This cost is estimated at $1.2 M, with the majority being the mining/geology software.

The dispatch system will use an iPad-based system with a Wi-Fi system in the pit area. This provides checklists and truck routing in addition to data collection.

The communication system is the establishment of radio/cell coverage in the pit area for use by mine engineering and operations complete with lightning protection.

The dewatering system includes pumps and piping required to draw the existing underground water level down below the active pit level and handle expected annual rainfall. The pumps will be electric and will lift the water to the pit rim then pump horizontally to the settling ponds on the west side of the pit for treatment (if required), and discharge to the environment. The initial cost is carried under Infrastructure capital.

Pit area preparation will include the removal of merchantable timber, grubbing, and any topsoil removed and stockpiled.

A diversion tunnel is needed to divert Tom McKay Creek around the north end of the pit as mining advances. To access the entrance and exit, this will require two access roads in the valley. The west access is in rugged terrain and is expected to cost more than the east access road. The west access road is estimated at 1.4 km in length and the east access road at 0.9 km.

The tunnel would be 2 kilometres long to have sufficient distance to divert the creek around the proposed active pit. This will accommodate the flow of Tom McKay creek.

Electrified hydraulic shovels will require a power line around the pit. The enclosed line is expected to be 6 kilometres in length from the main substation.

Table 21-5: Miscellaneous Mine Capital (C$)

Miscellaneous Mining Capital	Initial Cost ($)	Sustaining Cost ($)	Total Capital Cost ($)
Engineering office equipment	1,200,000	–	1,200,000
Dispatch system	800,000	–	800,000
Communications	400,000	–	400,000
Dewatering system – pumps/pipe	–	3,440,000	3,440,000
Pit area (clear/grub)	200,000	–	200,000
Tunnel access road – West	–	420,000	420,000
Tunnel access road – East	–	270,000	1,275,000
Tunnel – River diversion	–	14,000,000	14,000,000
Pit Powerline	1,380,000	–	1,380,000
Total	***3,980,000***	***18,130,000***	***22,110,000***

21.2.6 Mine Infrastructure

Various aspects of the site infrastructure were examined by Ausenco and other sub-consultants. These costs covered areas of the mine not directly associated with mining but in a support or preparation role. These include preparation of the waste storage facility foundations, water contact ponds and ditches, truck shop and water treatment. The cost has been detailed in Table 21-6.

The waste dump preparation is the cost to put in underdrains prior to waste placement. This is for control of water to the settling ponds and if required, treatment.

The water management ponds are the collecting areas of mine surface water runoff for testing and if required, treatment. The water contact channels direct the water to these ponds.

The initial dewatering system is built and installed as part of the mine infrastructure. This is the initial cost with the ongoing cost included in the miscellaneous mine capital.

The water treatment system has been sized for the expected flow rates in the mining area. It is required from the start of construction.

The truck shop is to be built adjacent to the plant facilities. This shop will accommodate the mine fleet for required preventative maintenance and rebuild functions.

The cost shown for the diversion tunnel includes detailed design as part of the overall infrastructure scope.

Table 21-6: Mine Infrastructure Capital (C$)

Mine Infrastructure Capital	Initial Cost ($)	Sustaining Cost ($)	Total Capital Cost ($)
Waste Dump Preparation	2,809,000	4,089,000	6,898,000
Water Management Pond	1,726,000	-	1,726,000
Water Contact Channels	566,000	375,000	942,000
Dewatering system – pumps/pipe	2,244,000	-	2,244,000
Open Pit Water Treatment System	1,836,000	-	1,836,000
Truck Shop	4,128,000	-	4,128,000
Diversion Tunnel	-	185,000	185,000
Mine Facilities	282,000	-	282,000
Total	***13,590,000***	***4,649,000***	***18,239,000***

21.2.7 Process Plant Capital Cost Estimate

21.2.7.1 Estimate Sources

Civil quantities were derived from a 3D model of the project site and priced using unit rates provided from experienced local contractors.

The MIA buildings (Truck Shop and Mine Dry Office), electrical materials, instrumentation materials and piping materials costs have been based on historical information from similar projects. Installation costs of these items have been included by applying discipline rates of placement and labour rates from experienced local contractors.

The Water Treatment Plant cost has been provided by McCue Engineering Contractors.

The project indirect costs associated with these mine infrastructure costs are included with the overall project indirect costs.

21.2.7.2 Estimate Summary

The capital cost estimate has been developed to AACE Class 4. A summary of the project capital cost estimate is presented in Table 21-77.

Table 21-7: Capital Cost Estimate Summary

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Processing			
Ore handling	17.4		17.4
Grinding, milling & classification	31.7	1.3	33.0
Separation & concentration	60.7		
Reagents & process utilities	5.0		
Tailings & reclaim water	8.2	6.1	14.3
Site preparation	14.1		
Onsite roads	15.7		
Onsite power transmission	13.2		
Other onsite infrastructure	25.0		
Sub-total Processing	***191.0***	***7.4***	***198.4***

21.2.8 Offsite Infrastructure Costs

Off-site infrastructure includes:

- Substation at Volcano Creek (287–69 kV);
- High voltage (HV) overhead power line 20 km x 69 kV: benchmarked cost per distance;
- Widening of the access road: semi-detailed quantities.

21.2.9 Indirect Costs

Indirect costs are those that are required during the project delivery period to enable and support the construction activities. Indirect costs include:

- Temporary facilities, common equipment and services, and vendor representation to support construction;
- Permanent Camp operations and maintenance during construction period;
- Note: a permanent camp (included in Direct Costs) will provide accommodation for the construction workers, with an allowance (included in Indirect Costs) for extra bed rentals (at peak times) at the existing camp site.
- First fills and spares;
- Commissioning;
- EPCM: home office engineering; site and home office expenses.

The indirect cost estimate was developed using a blend of first principles methods and percentages of direct costs. EPCM (Ausenco and Third Parties) was estimated at 14.4% of total direct costs (excluding mining costs), field indirect costs were by first principles method and equates to 13% of total direct costs (excluding mining costs) and Spares and First Fills was estimated at an average of 2% of total direct costs (excluding mining costs).

The indirect cost estimate is presented in Table 21-78.

Table 21-8: Indirect Costs Summary

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Indirect Costs			
Engineering, Procurement and Construction Management	33.5		33.5
Temporary Facilities	1.5		1.5
Temporary Services	13.7		13.7
Accommodation Camp Catering	13.3		13.3
Commissioning	1.9		1.9
Spares, First Fills and Vendor Reps	4.1		4.1
Sub-total Indirect Costs	*68.0*		*68.0*

21.2.10 Owners Costs

Owner's costs were estimated at 8% of total direct and indirect costs. These costs include an allowance for:

- General and administrative costs for the Owner's project team on and off-site;
- Security and First Aid
- COVID Testing
- Pre-production operations.

21.2.11 Closure Costs

Closure Costs based on the details provided in Section 20.4 are estimated to an approximate number of 92.4 M.

21.3 Operating Cost Estimate

21.3.1 Summary

The operating cost estimate provided in Table 21-9 is based on a combination of first-principal calculations, experience, reference projects and factors as appropriate for a PFS.

Table 21-9: Operating Cost Estimate Summary (C$)

Operating Cost	Annual Cost ($M)	Annual Cost ($/t Processed)
Processing	37.46	14.18
Maintenance	7.97	3.02
G&A	16.46	6.23
Road and bridge maintenance	2.70	1.02
Mining	80.74	30.56
Total	*145.34*	*55.01*

21.3.2 Mining

The Eskay Creek mine operating costs have been estimated from base principals with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The price provided for the project was $1.18/L delivered to the site. The mine fleet will be primarily diesel powered except for the loading shovels which will be electric powered. The dewatering pumps will be electric powered also and a price of $0.06 per kilowatt hour was used for any equipment using electricity.

21.3.2.1 Labour

Labour costs for the various job classifications were obtained from salary surveys in British Columbia and other operations. A burden rate between 39% and 44% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis using a rotation of either two weeks on/two weeks off or 4 days x 3 days. Mine positions and salaries are shown in Table 21-10.

The mine staff labour remains constant from Year 3 until Year 8, when positions are removed as the mine winds down. During the pre-production period there is one trainer and Year 1 and 2 there will be two trainer positions in mine operations.

Hourly employee labour force levels in mine operations and maintenance fluctuate with production requirements. The Year 5 hourly labour requirements are shown in Table 21-11. Labour costs are based on an Owner-operated scenario, with Skeena responsible for the maintenance of the equipment with its own employees.

Overseeing all the mine operations, maintenance, engineering, and geology functions will be a Technical Superintendent. This person would have the Mine General Foreman and Maintenance Superintendent reporting to them, as well as the Chief Engineer and Chief Geologist.

The Mine General Foreman would have the Shift Foremen report directly to them.

The mine will have four mine operations crews, each with a Senior Shift Foremen who will have one Junior Shift Foreman reporting to them. Over the mine life, there will also be a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person would also be a backup Senior Mine Shift Foreman. The Training Foreman roles are only required on site until the end of Year 2, at which time the positions are eliminated. The Mine Operations department will have its own Clerk/Secretary.

Table 21-10: Mine Staffing Requirements and Annual Employee Salaries (Year 5)

Position	Employees	Annual Salary (C$/a)
Mine Maintenance		
Maintenance Superintendent	1	207,000
Maintenance General Foreman	1	178,100
Maintenance Shift Foremen	4	144,900
Maintenance Planner/Contract Administration	2	132,100
Clerk	1	85,800
Subtotal	*9*	
Mine Operations		
Mine Operations/Technical Superintendent	1	220,800
Mine General Foreman	1	191,800
Senior Shift Foreman	4	144,900
Junior Shift Foreman	4	132,100
Road Crew/Services Foreman	1	144,900
Clerk	1	85,800
Subtotal	*12*	
Mine Engineering		
Chief Engineer	1	194,600
Senior Engineer	1	164,400
Open Pit Planning Engineer	2	144,900
Geotechnical Engineer	1	144,900
Blasting Engineer	1	144,900
Blasting/Geotechnical Technician	2	98,700
Dispatch Technician	1	98,700
Surveyor/Mining Technician	2	98,700
Surveyor/Mining Technician Helper	2	92,300
Clerk	1	85,800
Subtotal	*14*	
Geology		
Chief Geologist	1	180,700
Senior Geologist	1	151,800
Grade Control Geologist/Modeller	4	125,100
Sampling/Geology Technician	6	98,700
Clerk	1	85,800
Subtotal	*13*	

Total	*48*	

Table 21-11: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary (C$/a)
Mine General		
General Equipment Operator	16	103,100
Road/Pump Crew	8	100,000
General Mine Labourer	8	80,100
Light Duty Mechanic	4	133,000
Tire Technician	4	107,900
Lube Truck Driver	4	92,600
Subtotal	*44*	
Mine Operations		
Driller	16	107,900
Blaster	2	107,900
Blast Helper	4	80,100
Loader Operator	4	119,300
Hydraulic Shovel Operator	8	119,300
Haul Truck Driver	48	103,000
Dozer Operator	12	107,900
Grader Operator	6	107,900
Crusher Loader Operator	4	119,300
Snow plow/Water Truck	8	101,000
Subtotal	*144*	
Mine Maintenance		
Heavy/Light Duty Mechanics	30	133,000
Welder	17	133,000
Electrician	2	133,000
Apprentice	7	93,300
Subtotal	*56*	
Total Hourly	*244*	

The Chief Engineer will have one Senior Engineer and two Open Pit Engineers reporting to them. The Blasting Engineer would be included in the Short-Range Planning Group and would double as Drill-And-Blast Foreman as required. The Geotechnical Engineer would cover all aspects of the wall slopes and WRSFs, together with shared technicians in blasting.

The Short-Range Planning Group in Engineering will have two Surveyor/Mine Technicians and two Surveyors/Mine Helpers. These employees will assist in the field with staking, surveying, and sample collection with the geology group; they will have a Clerk/Secretary to assist the team.

In the Geology Department, there will be one Senior Geologist reporting to the Chief Geologist. There will also be four Grade Control Geologists/Modellers; two will be in short range and grade control drilling, and the others will be in long range/reserves. There will also be six Grade Control/Sampling Technicians and one Clerk/Secretary.

Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman who in turn will report to the Maintenance Superintendent. There will be two Maintenance Planners/Contract Administrators and a Clerk.

The hourly labour force includes positions for the Light Duty Mechanic, Tire Men, and Lube Truck Drivers. These positions will all report to Maintenance. There will generally be one of each position per crew. Other general labour includes General Mine Labourers (two per crew) and Trainees (one per crew until Year 5) plus two Road/Pump Crew personnel per crew for water management/snow removal.

The drilling labour force is based on one operator per drill, per crew while operating. This peaks at 20 Drillers in Year 2 and maintains that level until Year 7 and then drops down over time as the drilling hours are diminished.

Shovel and Loader Operators peak at 12 in Year 2 and hold at that level until Year 8. Haulage Truck Drivers peak at 56 in Year 6 and 7 and then taper off to the end of the mine life.

Maintenance factors are used to determine the number of Heavy-Duty Mechanics, Welders and Electricians are required and are based on the number of equipment operators. Heavy Duty Mechanic requirements work out to 0.25 mechanics required for each Drill Operator for example. Welders are 0.25 per operator and Electricians are 0.05 per operator.

The number of Loader, Truck and Support Equipment Operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit, to match the mine crews.

21.1.1.1 Equipment Operating Costs

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Eskay Creek PFS. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the R&M are expressed in $/h form.

Tire costs were also collected from various vendors for the sizes expected to be used. Estimates of tire life are based on AGP's experience. The operating cost of the tires is expressed in a $/hr form also. The life of the haulage truck tires is estimated at 5,000 hours per tire for the 144 t trucks and 5,500 hours for the 91 t trucks with proper rotation from front to back. Each truck tire for the 144 t truck costs $21,000 so the cost per hour for tires is $25.20 /hr for the truck using six tires in the calculation.

Ground engaging tools (GET) costing is estimated from other projects and is an area that would be fine-tuned once the project was operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 24.1 m/hr for mill feed and waste. The equipment costs used in the estimate are shown in Table 21-12.

Table 21-12: Major Equipment Operating Costs – No Labour ($/hr)

Equipment	Fuel/ Power	Lube/Oil	Tires/ Undercarriage	Repair & Maintenance	GET/ Consumables	Total
Production drill – 140 mm	76.70	7.67	3.00	125.00	102.34	314.71
Production drill – 229 mm	121.54	12.15	6.00	96.27	174.48	410.45
Production/crusher loader - 11.5 m3	100.30	10.03	20.80	72.94	10.00	214.07
Production loader - 23 m3	135.70	20.36	46.86	149.46	32.17	384.54
Hydraulic shovel – 22 m3	66.00	-	-	185.42	30.00	281.42
Haulage truck – 91 t	88.50	8.85	14.73	66.59	3.00	181.67
Haulage truck – 144 t	103.84	10.38	25.20	91.95	4.00	235.37
Track dozer	94.4	9.44	10.00	64.76	5.00	183.60
Grader	25.96	2.60	4.00	15.60	5.00	53.16
Dragline	106.20	10.62	10.00	103.69	5.00	235.51
Support excavator – 6.7 m3	70.80	14.16	-	58.24	8.00	151.20

21.3.2.2 Drilling

Drilling in the open pit will use down the hole hammers drill rigs. The preproduction drilling will be with the smaller drills and 140 mm bits but convert to 229 mm bits with the main production drill. The pattern size varies between mill feed and waste and is blasted in recognition of the equipment being used. The material will be smaller and finer to improve productivity and reduce maintenance costs as well as improve plant performance. The drilling pattern parameters are shown in Table 21-13.

Table 21-13: Drill Pattern Specifications

		Drill 140 mm		Drill 229 mm	
Specification	Unit	Mill Feed	Waste	Mill Feed	Waste
Bench height	m	8	8	8	8
Sub-drill	m	0.8	0.8	1.2	1.2
Blasthole diameter	mm	140	140	229	229
Pattern spacing - staggered	m	4.8	4.6	6.9	6.9
Pattern burden – staggered	m	4.2	4.0	6.0	6.0
Hole depth	m	8.8	8.8	9.2	9.2

The sub-drill is included to allow for caving of the holes in weaker zones, reducing re-drill requirements or short holes that would affect bench floor conditions. The extra sub-drill is above what is normally required.

The parameters used to estimate drill productivity are shown in Table 21-14.

Table 21-14: Drill Productivity Criteria

		Drill 140 mm		Drill 229 mm	
Drill Activity	Unit	Mill Feed	Waste	Mill Feed	Waste
Pure penetration rate	m/min	0.55	0.55	0.50	0.50
Hole depth	m	8.8	8.8	9.2	9.2
Drill time	min	16.00	16.00	18.40	18.40
Move, spot and collar hole	min	3.00	3.00	3.00	3.00
Level drill	min	0.50	0.50	0.50	0.50
Add steel	min	0.50	0.50	0.00	0.00
Pull drill rods	min	1.50	1.50	1.00	1.00
Total setup/breakdown time	min	5.50	5.50	4.50	4.50
Total drill time per hole	min	21.5	21.5	22.9	22.9
Drill productivity	m/hr	24.6	24.6	24.1	24.1

21.1.1.2 Blasting

An emulsion product will be used for blasting to provide water protection. With the high rainfall known to occur in the area and large snow melt, it is expected that a water-resistant explosive will be required. The powder factors used in the explosives calculation are shown in Table 21-15.

The blasting cost is estimated using quotations from a local explosives vendor. The emulsion price is $94.28/100 kg. The operations will be responsible for guiding the loading process, including placement of boosters/Nonels, and stemming and firing the shot.

The explosives vendor will lease the explosives and accessories for a monthly cost. A service charge for the vendors pickup trucks, pumps, labour, and cost of the explosives plant are included. The total monthly cost was $141,000 per month.

Table 21-15: Design Powder Factors

	Unit	Drill 140 mm		Drill 229 mm	
		Mill Feed	Waste	Mill Feed	Waste
Powder Factor	kg/m^3	0.70	0.78	0.78	0.78
Powder Factor	kg/t	0.26	0.29	0.29	0.29

21.3.2.3 Loading

Loading costs for both mill feed and waste are based on the use of hydraulic shovels and front-end loaders. The shovels will be the primary diggers with the front-end loader as backup/support units. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-16, as at Year 5. This highlights the focus of the shovels over the loaders.

The trucks present at the loading unit refers to the percentage of time a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the loading unit. One of the largest operating cost items is haulage and minimizing this cost by maximizing the truck productivity is crucial to lower operating costs. The value of 80% comes from the standby time shovels typically encounter due to a lack of trucks.

Table 21-16: Loading Parameters – Year 5

	Unit	Hydraulic Shovel	Front End Loader
Bucket capacity	m^3	22	23
Truck capacity loaded	t	144	144
Waste tonnage loaded	%	85	15
Mill feed tonnage loaded	%	75	25
Bucket fill factor	%	95	95
Cycle time	sec	38	40
Trucks present at loading unit	%	80	80
Loading time	min	2.60	2.70

21.3.2.4 Hauling

Haulage profiles were determined for each pit phase for the primary crusher, waste rock facility or PAG storage at the tailings facility. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on the trucks is limited to 50 km/hr for tire life and safety reasons although few locations in the mine plan appeared to offer the truck the opportunity to accelerate to that velocity. Calculation speeds for various segments are shown in Table 21-17.

Table 21-17: Haulage Cycle Speeds

	Flat (0%) On Surface	Flat (0%) In-pit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)
Loaded (km/hr)	50	40	16	12.1	30	30
Empty (km/hr)	50	40	35	25	35	35

21.3.2.5 Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PEA, some of the factors used are shown in Table 21-18.

These factors resulted in the need for five track dozers, three graders, one dragline and one support backhoe. Their tasks will include clean-up of the loader faces, roads, WRSFs, and blast patterns. The graders will maintain the crusher and waste haul routes. In addition, snow plow/water trucks will have the responsibility for patrolling the haul roads for snow removal and controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for cleaning out sedimentation ponds and water ditch repairs.

The dragline will be responsible for pulling the PAG material stored at the tailings facility beneath the water level. The dragline is used for safety reasons with dozing material in the tailings facility. The extended operating range of the dragline allows the material to be moved while positioned on stable ground.

The hours generated in this manner were applied to the individual operating costs for each piece of equipment. Many of these units will be support equipment, so no direct labour is allocated to them due to their variable function. The operators will come from the General Equipment operator pool.

Table 21-18: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track dozer	30%	Of haulage hours to maximum of 4 dozers
Grader	15%	Of haulage hours to maximum of 3 graders
Crusher loader	50%	Of loading hours to maximum of 1 loader
Snow plow/water truck	12%	Of haulage hours to maximum of 3 trucks
Pit support backhoe	35%	Of loading hours to maximum of 1 backhoe
Dragline	12	Hours/day/unit
Road crew backhoe	8	hours/day/unit
Road crew dump truck	8	hours/day/unit
Road crew loader	8	hours/day/unit
Lube/fuel truck	8	hours/day/unit
Mechanics truck	14	hours/day/unit
Blasting loader	8	hours/day/unit
Blaster’s truck	8	hours/day/unit
Integrated tool carrier	4	hours/day/unit
Light plants	12	hours/day/unit
Pickup trucks	8	hours/day/unit

21.3.2.6 Grade Control

Grade control will be completed with a separate fleet of RC drill rigs. These rigs will drill the deposit off on a 10 x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60º.

In areas of low-grade mineralization or waste, the pattern spacing will be 20 x 10 m, with sampling over 6 m. These drill holes will be used to find undiscovered veinlets or pockets of mineralization. Over the life of the mine, a total of 210,000 m of drilling are expected to be completed for grade control work. A total of 231,000 samples is anticipated to be assayed from that drilling.

The grade control holes will serve two purposes:

- Definition of the mill feed grade and contacts;
- Location of previous underground infrastructure prior to blasthole rigs drilling.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Additional costing for blasthole sampling has not been included. This may add additional level of grade control but may not be necessary once a gold deportment study is completed to best determine the sampling protocol.

Costs associated with this separate drill program will be tracked as a distinct line item for the mining cost. The drill crew will be one driller and two helpers with oversight by the Mine Geology Department. The cost of this drilling is expected to be over $2 M/a.

21.3.2.7 Dewatering

Pit and underground workings dewatering will be an important part of mining at Eskay Creek. Significant volumes will need to be pumped initially to allow the open pit to advance, in addition to the normally elevated rain/snow amounts.

For the purposes of the PFS, historical dewatering data was reviewed and compared this to the proposed mining area to estimate the water volume that will be required to be pumped. Initial pumping in Year -3 is expected to be just over 1 million cubic metres. That climbs rapidly to 3.5 Mm3 in Years -2 to Year 1 then levels at around 4.8 Mm3 for the remainder of the mine life. This volume also includes the WRSF areas as water from these areas is expected to be controlled, sampled and treated if required.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface. These pumps will be electric to reduce the cost of this operation.

Additional dewatering in the form of horizontal drill holes is included as part of the dewatering costs. These holes will be campaigned and will be part of the sustaining mine capital.

Dewatering is expected to cost $4.6 M over the proposed mine life.

21.3.2.8 Leasing

Leasing of the mine fleet is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment. Both Caterpillar and Komatsu have the ability, and desire, to allow leasing of their product lines.

Indicative terms for leasing provided by the vendors are:

- Down payment = 20% of equipment cost;
- Term Length = 3-5 years (depending on equipment);
- Interest Rate = LIBOR plus a percentage;
- Residual = $0.

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.067 to equate to the rate. It does not consider a declining balance on the interest, but rather the full amount of interest paid over the term, equally distributed over those years. The calculation is as follows:

- Annual Lease Cost = {[(Initial Capital Cost) x 80%] x 1.067} / term in years

The initial capital, down payments, and annual leasing costs were included in Section 21.2.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All the major mine equipment, and the majority of the support equipment, where it was considered reasonable, was assumed to be leased. If the equipment had a life greater than the lease term length, then the following years onward of the lease did not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

Using the leasing option adds $0.38/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it was $3.24/t mill feed.

21.3.2.9 Total Mine Costs

The total life of mine operating costs per tonne of material moved and per tonne of mill feed processed are shown in Table 21-19 and Table 21-20.

The General Mine Engineering includes the cost associated with a contract crushing plant to make stemming material and road crush. That cost is approximately $2.7 M/a.

Table 21-19: Open Pit Mine Operating Costs – with Leasing ($/t Total Mined)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mined	0.63	0.39	0.42	0.47
Drilling	$/t mined	0.37	0.33	0.35	0.36
Blasting	$/t mined	0.61	0.51	0.55	0.56
Loading	$/t mined	0.28	0.21	0.22	0.23
Hauling	$/t mined	0.61	0.72	0.81	0.82
Support	$/t mined	0.70	0.55	0.61	0.64
Grade control	$/t mined	0.12	0.06	0.09	0.09
Leasing costs	$/t mined	0.92	0.49	0.42	0.38

Dewatering	$/t mined	0.03	0.02	0.02	0.03
Total	*$/t mined*	*4.27*	*3.29*	*3.49*	*3.58*

Table 21-20: Open Pit Mine Operating Costs – with Leasing ($/t Mill Feed)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mill feed	6.05	4.09	4.28	4.03
Drilling	$/t mill feed	3.51	3.50	3.60	3.05
Blasting	$/t mill feed	5.85	5.42	5.66	4.79
Loading	$/t mill feed	2.69	2.24	2.21	1.96
Hauling	$/t mill feed	5.80	7.55	8.33	6.99
Support	$/t mill feed	6.70	5.84	6.28	5.48
Grade control	$/t mill feed	1.14	0.67	0.88	0.80
Leasing costs	$/t mill feed	8.84	5.21	4.35	3.24
Dewatering	$/t mill feed	0.25	0.22	0.23	0.22
Total	*$/t mill feed*	*40.81*	*34.72*	*35.81*	*30.56*

21.3.3 Processing

Processing costs for power, consumables, maintenance consumables and labour are summarized in Table 21-21.

21.3.4 Power

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh.

Power consumption was derived from calculated power draw of the ball and SAG mills, plus an allowance for the remainder of the plant, based on typical flotation plants. The average on-line power draw is estimated at 19 MW.

Annual energy consumption is estimated at 147,416 MWh, or about $8.85 M.

21.3.5 Consumables

Processing reagent and consumable costs were estimated based on the throughput. Costs are summarized in Table 21-22.

Costs for liners were estimated based on vendor information and benchmarking similar plants. Costs for mill balls were estimated for expected consumption based on an assumed abrasion index (Ai) of 0.22. These costs are summarized in Table 21-23.

Reagent costs were based on:

- Consumption rates determined in test work;
- Data base unit costs for the reagents;

Reagent costs are summarized in Table 21-24.

21.3.6 Maintenance Consumables

Annual maintenance spares and consumable costs were estimated at 4% of total installed costs for mechanical equipment, plate work, support steel and electrics ($47.5 M).

This results in an annual maintenance consumables cost estimate of $1.86 M.

21.3.7 Labour

Labour costs include all processing and maintenance costs (Table 21-25).

Costs were estimated from a breakdown of staffing positions, estimated at 98 in total, excluding G&A manpower.

Labour costs are based on annual salaries inclusive of all burdens applicable to the site.

Table 21-21: Processing Costs (C$)

Processing Cost item	Annual Cost ($M)	Annual Cost ($/t Processed)
Processing Production Labour	9.04	3.42
Processing Maintenance Labour	5.04	1.91
Power	8.45	3.20
Operating Consumables - Process	19.97	7.56
Maintenance Consumables	1.86	0.70
Light Vehicles & Mobile Equipment	1.08	0.41
Sub-Total	*45.44*	*17.20*
G&A	16.46	6.23
Road and Bridge Maintenance	2.70	1.02
Total	*64.60*	*24.25*

Table 21-22: Processing Reagent and Consumable Costs (C$)

Consumable Item	Annual Costs ($M)
Crushing & Conveying	0.18
Grinding/Milling/Classification	4.73
Flotation/Regrind	14.99
Product Dewatering & Storage	0.08
Total	*19.97*

Table 21-23: Costs for Liners and Media

Consumable Item	Annual Consumption	Annual Cost (C$000)
Crusher liners	3 sets	179
SAG mill liners	1 set	513
Pebble crusher liners	2 sets	10
Ball mill liners	1 set	260.3
SAG mill balls	0.3 kg/t	1,364

Consumable Item	Annual Consumption	Annual Cost (C$000)
Ball mill balls	0.709 kg/t	2,573
Reline contractors	N/A	10
Regrind mill spares	1 set	274
Secondary mill spares	1 set	408
Regrind mill media	0.040 kg/t	312
Secondary mill media	0.146 kg/t	1,141
Total		*7,045*

Table 21-24: Reagent Costs

Reagent	Addition Rate (kg/t)	Annual Cost (C$000)
PAX	0.73	5,771
MIBC	0.25	2,202
Copper sulphate	0.60	4,877
Flocculant	0.002	22.7
Total		*12,872*

Table 21-25: Labour Costs

Cost Centre	Number	Annual Cost (C$M)
Plant management	2	0.47
Foremen and working staff	15	2.86
Mill operators and sample preparation	51	6.71
Plant maintenance	30	4.04
Total	*98*	*14.07*

21.3.8 General and Administration

The G&A operating costs were estimated based on benchmarked data from similar projects in B.C. Canada. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

The annual G&A cost is estimated at $16.46 M/a.

21.4 QP Comments on "Item 21: Capital and Operating Costs"

Capital costs are estimated at $487.9 M of initial capital, $139.8 M of sustaining capital, for an overall capital cost estimate of $627.7 M;

Process operating costs of $145.34 M/a, or $55.01/t processed.

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

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

- Mineral Resource and reserve estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and ability to mine in areas previously exploited using underground mining methods as envisaged;
- Sustaining costs and proposed operating costs;
- Interpretations and assumptions as to joint venture and agreement terms;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what are estimated;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade or recovery rates;
- Geotechnical or hydrogeological considerations during mining being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;

- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;

- Ability to maintain the social licence to operate;

- Accidents, labour disputes and other risks of the mining industry;

- Changes to interest rates;

- Changes to tax rates.

This PFS assumes that permits have to be obtained in support of operations, and approval for development to be provided by Skeena's Board.

22.2 Methodology Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess impact of variations in metal prices, head grades, operating costs, and capital costs. The capital and operating cost estimates were developed specifically for this Project and are summarized in Section 21 of this Report (presented in 2021 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- Construction period of three years;

- Mine life of 9.8 years;

- Base case gold price of US$1,550/oz and silver price of US$22/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;

- United States to Canadian dollar exchange rate assumption of 0.78 (US$/C$)

- Cost estimates in constant Q2 2021 C$ with no inflation or escalation factors considered;

- Results are based on 100% ownership with 2% NSR;

- Capital costs funded with 100% equity (i.e. no financing costs assumed);

- All cash flows discounted to start of construction;

- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold concentrate into the international marketplace;
- No contractual arrangements for smelting or refining currently exist.

22.3.1 Taxes

The Project has been evaluated on an after-tax basis to provide approximate value of the potential economics. The tax model was prepared by an independent tax consultant. The calculations are based on the tax regime as of the date of the PFS.

At the effective date of the cashflow, the Project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%);
- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

Total tax payments are estimated to be C$1,145 M over the LOM.

22.3.2 Working Capital

A high-level estimation of working capital has been incorporated into the cash flow based on Accounts Receivable (0 days), Inventories (30 days) and Accounts Payable (30 days).

22.3.3 Closure Costs

Total closure cost is estimated to be C$92 M.

22.4 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$2,174 M, the internal rate of return IRR is 68.3%, and payback is 1.3 years. On an after-tax basis, the NPV 5% is C$1,399 M, the IRR is 55.5%, and the payback period is 1.4 years.

A summary of the Project economics is included in Table 22-1 and shown graphically in Figure 22-1. The cashflow on an annualized basis is provided in Table 22-2.

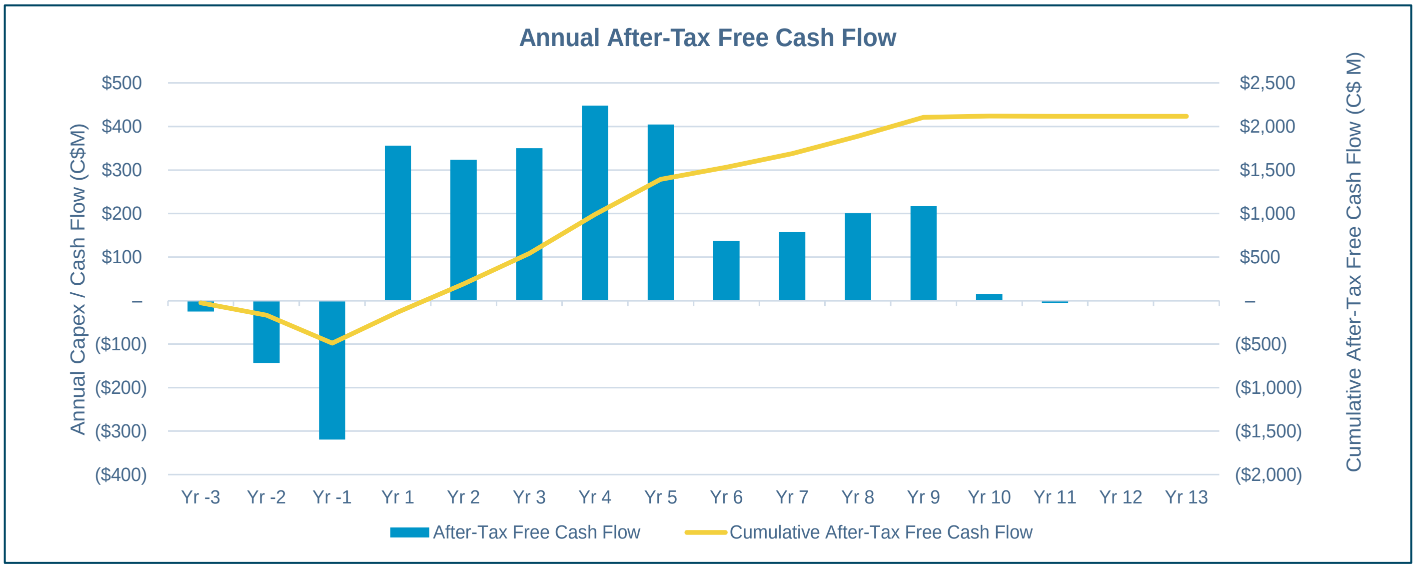
Table 22-1: Summary, Projected LOM Cashflow Assumptions and Results

	Units	Values
General Assumptions		
Gold price	(US$)	1,550
Silver price	(US$)	22
Exchange rate	(US$/C$)	0.78
Fuel cost	(C$/litre)	1.18
Power cost	(C$/kwh)	0.06
Discount rate	(%)	5%
Net smelter royalty	(%)	2%
Contained Metals		
Contained gold ounces	(koz)	2,866
Contained silver ounces	(koz)	80,197
Production		
Gold recovery	(%)	84.2%
Silver recovery	(%)	87.3%
LOM gold production	(koz)	2,448
LOM silver production	(koz)	70,902
LOM gold equiv. production	(koz)	3,455
LOM avg. annual gold production	(koz per annum)	249
LOM avg. annual silver production	(koz per annum)	7,222
LOM avg. annual gold equiv. production	(koz per annum)	352
Operating Costs Per Tonne		
Mining cost	(C$/t mined)	$3.6
Mining cost	(C$/t milled)	$30.6
Processing cost	(C$/t milled)	$18.2
G&A cost	(C$/t milled)	$6.2
Total operating costs	(C$/t milled)	$55.0
NSR Parameters		
Gold payability	(%)	83.9%
Silver payability	(%)	83.2%
Transport to smelter	(C$/wmt)	$146
Cash Costs and All-in Sustaining Costs		
LOM cash cost net of silver by-product	(US$/oz Au)	$84
LOM cash cost co-product	(US$/oz AuEq)	$509

	Units	Values
LOM AISC net of silver by-product	(US$/oz Au)	$138
LOM AISC co-product	(US$/oz AuEq)	$548
Capital Expenditures		
Initial capex	(C$M)	$488
Sustaining capex	(C$M)	$47
Closure capex	(C$M)	$92
Economics		
Pre-tax NPV (5%)	(C$M)	$2,174
Pre-tax IRR	(%)	68.3%
Pre-tax payback period	(years)	1.3
Pre-Tax NPV / Initial Capex	(x)	4.5 x
After-tax NPV (5%)	(C$M)	$1,399
After-tax IRR	(%)	56%
After-tax payback period	(years)	1.4
After-Tax NPV / Initial Capex	(x)	2.9 x
Average annual after-tax free cash flow (Year 1–9)	(C$M)	$265
LOM after-tax free cash flow	(C$M)	$2,118

Notes: Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. AISC includes cash costs plus corporate G&A, sustaining capital and closure costs. Gold equivalent (AuEq) calculated using the formula: Au (g/t) + [Ag (g/t) / 70].

Figure 22-1: Projected LOM Cashflow



Note: Figure prepared by Ausenco, 2021.

Table 22-2: Projected Cashflow on an Annualized Basis

Dollar figures in real C$mm unless otherwise noted	Units	Total / Avg.	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Free Cash Flow Valuation																		
Gross Revenue		$5,795.8	–	–	–	$594.9	$670.3	$762.4	$946.6	$848.5	$416.4	$442.0	$487.9	$505.4	$121.3	–	–	–
Penalties	C$mm	($74.0)	–	–	–	($39.9)	($11.3)	($6.9)	($13.0)	($0.7)	–	–	($2.1)	($0.1)	($0.0)	–	–	–
Transport	C$mm	($268.4)	–	–	–	($30.1)	($31.6)	($31.8)	($30.7)	($30.3)	($29.8)	($26.8)	($20.1)	($22.9)	($14.3)	–	–	–
Net Smelter Return		$5,453.4	–	–	–	$524.9	$627.4	$723.7	$902.9	$817.5	$386.6	$415.2	$465.7	$482.5	$107.0	–	–	–
Operating Expenses	C$mm	($1,453.4)	–	–	–	($130.9)	($171.5)	($171.6)	($172.2)	($162.7)	($159.4)	($154.3)	($133.7)	($122.9)	($74.0)	–	–	–
Royalties	C$mm	($109.1)	–	–	–	($10.5)	($12.5)	($14.5)	($18.1)	($16.4)	($7.7)	($8.3)	($9.3)	($9.6)	($2.1)	–	–	–
EBITDA	C$mm	$3,891.0	–	–	–	$383.5	$443.3	$537.6	$712.7	$638.5	$219.4	$252.6	$322.6	$349.9	$30.8	–	–	–
Initial Capex	C$mm	($487.9)	($25.2)	($143.6)	($319.2)	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining Capex	C$mm	($47.4)	–	–	–	($15.7)	($13.8)	($6.9)	($7.8)	($1.0)	($1.4)	($0.6)	($0.1)	($0.1)	–	–	–	–
Closure Capex	C$mm	($92.4)	–	–	–	($4.2)	($4.3)	($8.5)	($8.7)	($8.9)	($9.1)	($9.3)	($9.5)	($9.7)	($9.9)	($10.1)	–	–
Change in Working Capital	C$mm	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Pre-Tax Unlevered Free Cash Flow		$3,263.3	($25.2)	($143.6)	($319.2)	$363.7	$425.3	$522.2	$696.1	$628.6	$208.9	$242.6	$312.9	$340.1	$20.9	($10.1)	–	–
Pre-Tax Cumulative Unlevered Free Cash Flow			($25.2)	($168.7)	($487.9)	($124.2)	$301.1	$823.3	$1,519.4	$2,148.0	$2,356.9	$2,599.6	$2,912.5	$3,252.5	$3,273.4	$3,263.3	$3,263.3	$3,263.3
Unlevered Cash Taxes	C$mm	($1,145.4)	–	–	–	($7.8)	($101.9)	($171.9)	($248.0)	($223.8)	($71.7)	($85.4)	($112.2)	($122.9)	($6.2)	$4.9	$0.9	$0.6
Post-Tax Unlevered Free Cash Flow		$2,117.9	($25.2)	($143.6)	($319.2)	$355.9	$323.4	$350.4	$448.1	$404.8	$137.2	$157.2	$200.7	$217.2	$14.6	($5.3)	$0.9	$0.6
Post-Tax Cumulative Unlevered Free Cash Flow			($25.2)	($168.7)	($487.9)	($132.0)	$191.4	$541.7	$989.9	$1,394.7	$1,531.9	$1,689.1	$1,889.8	$2,107.0	$2,121.6	$2,116.4	$2,117.2	$2,117.9
Production																		
Open Pit Production																		
Ore Mined	'000t	26,419	28	11	522	2,908	2,416	1,924	3,083	3,164	2,955	2,585	2,853	3,092	879	–	–	–
Stockpile Rehandle	'000t	4,657	28	11	522	1,711	616	68	183	464	262	248	153	392	–	–	–	–
Waste Mined	'000t	211,611	1,899	1,389	8,517	16,264	28,848	28,675	26,215	24,507	25,443	22,462	14,147	10,908	2,337	–	–	–
Total Material Mined (Includes Rehandle)	'000t	242,687	1,954	1,411	9,560	20,883	31,881	30,666	29,480	28,135	28,659	25,296	17,153	14,392	3,216	–	–	–
Total Material Mined (Excl. Rehandle)	'000t	238,030	1,927	1,400	9,039	19,172	31,265	30,598	29,298	27,671	28,397	25,048	17,000	14,000	3,216	–	–	–
Strip Ratio		8.01	–	–	–	8.11	9.95	9.89	9.04	9.08	9.42	8.32	5.24	4.04	1.06	–	–	–
Total Mill Feed	'000t	26,419	–	–	–	2,006	2,900	2,900	2,900	2,700	2,700	2,700	2,700	2,700	2,213	–	–	–
Beginning Stockpile Inventory	'000t		–	28	39	560	1,463	979	2	185	649	903	789	942	1,334	0	0	0
Add: Mine to Stockpile	'000t	4,657	28	11	522	1,711	616	68	183	464	262	248	153	392	–	–	–	–
Less: Stockpile to Mill	'000t	(4,657)	–	–	–	(809)	(1,100)	(1,044)	–	–	(7)	(363)	–	–	(1,334)	–	–	–
Ending Stockpile Inventory	'000t		28	39	560	1,463	979	2	185	649	903	789	942	1,334	0	0	0	0
Au Head Grade	g/t	3.4	–	–	–	5.0	3.5	3.8	4.2	4.5	2.8	2.6	3.2	2.8	1.1	–	–	–
Ag Head Grade	g/t	94.4	–	–	–	97.4	94.2	119.1	165.1	131.7	57.6	64.8	53.3	113.6	29.3	–	–	–
Contained Gold	kozs	2,866.5	–	–	–	322.4	322.9	356.9	394.8	391.8	245.3	229.6	275.4	246.0	81.3	–	–	–
Contained Silver	kozs	80,196.9	–	–	–	6,284.2	8,784.0	11,108.1	15,392.4	11,428.2	5,000.8	5,620.9	4,629.0	9,863.4	2,086.1	–	–	–
Au Recovery	%	84.2%	0%	0%	0%	85%	88%	86%	88%	89%	82%	88%	84%	81%	68%	0%	0%	0%
Ag Recovery	%	87.3%	0%	0%	0%	90%	93%	91%	93%	93%	87%	93%	88%	70%	72%	0%	0%	0%
		Total LOM																
Recovered Gold in Concentrate	kozs	2,448.4	–	–	–	275.0	283.2	306.6	347.4	346.8	201.6	202.7	230.0	199.8	55.3	–	–	–
Recovered Silver in Concentrate	kozs	70,902.3	–	–	–	5,655.8	8,125.2	10,075.0	14,299.5	10,674.0	4,335.7	5,233.0	4,078.1	6,924.1	1,502.0	–	–	–
Recovered Gold Equivalent in Concentrate	kozs	3,454.8	–	–	–	355.3	398.5	449.6	550.4	498.3	263.2	277.0	287.9	298.1	76.7	–	–	–
Au Payability		83.9%	79.5%	79.5%	79.5%	85.5%	85.5%	85.5%	86.0%	86.0%	79.5%	79.5%	86.0%	85.5%	79.5%	79.5%	79.5%	79.5%
Ag Payability		83.2%	75.0%	75.0%	75.0%	80.0%	82.5%	85.0%	87.5%	85.0%	80.0%	82.5%	82.5%	85.0%	80.0%	75.0%	75.0%	75.0%
Total Payable Gold (Open Pit)	kozs	2,070.5	–	–	–	235.2	242.1	262.1	298.8	298.2	160.3	161.1	197.8	170.8	44.0	–	–	–
Total Payable Silver (Open Pit)	kozs	59,613.9	–	–	–	4,524.6	6,703.3	8,563.8	12,512.1	9,072.9	3,468.5	4,317.3	3,364.4	5,885.5	1,201.6	–	–	–
Total Payable Gold Equivalent (Open Pit)	kozs Au Eq	2,916.6	–	–	–	299.4	337.3	383.7	476.4	427.0	209.5	222.4	245.5	254.4	61.1	–	–	–
Macro Assumptions																		
Gold Price	US$/oz	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550	$1,550
Silver Price	US$/oz	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22	$22
FX	C$:US$	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
Revenue																		
Gold Revenue	C$mm	$4,114.4	–	–	–	$467.3	$481.2	$520.8	$593.7	$592.6	$318.6	$320.2	$393.0	$339.4	$87.4	–	–	–
Silver Revenue	C$mm	$1,681.4	–	–	–	$127.6	$189.1	$241.5	$352.9	$255.9	$97.8	$121.8	$94.9	$166.0	$33.9	–	–	–
Total Revenue	C$mm	$5,795.8	–	–	–	$594.9	$670.3	$762.4	$946.6	$848.5	$416.4	$442.0	$487.9	$505.4	$121.3	–	–	–
Total Mill Feed	'000t	26,419	–	–	–	2,006	2,900	2,900	2,900	2,700	2,700	2,700	2,700	2,700	2,213	–	–	–
Mass Pull		6.12%	–	–	–	9.04%	6.56%	6.60%	6.38%	6.77%	6.66%	5.99%	4.49%	5.11%	3.89%	–		
Concentrate Produced	000t (dmt)	1,618	–	–	–	181	190	191	185	183	180	162	121	138	86	–	–	–
Concentrate Au Grade	g/t	46	–	–	–	47	46	50	58	59	35	39	59	45	20	–	–	–
Concentrate Ag Grade	g/t	1,375	–	–	–	971	1,329	1,635	2,405	1,816	751	1,008	1,046	1,904	542	–	–	–
Penalties	C$mm	$74.0		–	–	$39.9	$11.3	$6.9	$13.0	$0.7	–	–	$2.1	$0.1	$0.0	–	–	–
Transport to Smelter	C$mm	$268.4	–	–	–	$30.07	$31.57	$31.77	$30.70	$30.33	$29.81	$26.82	$20.11	$22.91	$14.28	–	–	–
Net Smelter Return		$5,453.4	–	–	–	$524.9	$627.4	$723.7	$902.9	$817.5	$386.6	$415.2	$465.7	$482.5	$107.0	–	–	–
Royalty	%	2.00%	–	–	–	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	–	–	–
Total Royalties	C$mm	$109.1	–	–	–	$10.5	$12.5	$14.5	$18.1	$16.4	$7.7	$8.3	$9.3	$9.6	$2.1	–	–	–
Penalties																		
Total Antimony (Sb) Penalty	C$mm	$5.2	–	–	–	–	$3.0	–	$2.2	–	–	–	–	–	–	–	–	–
Total Arsenic (As) Penalty	C$mm	$18.2	–	–	–	$9.9	$4.6	–	$0.8	$0.7	–	–	$2.1	$0.1	$0.0	–	–	–
Total Mercury (Hg) Penalty	C$mm	$50.7	–	–	–	$30.0	$3.7	$6.9	$10.0	–	–	–	–	–	–	–	–	–
Total Penalties	C$mm	$74.0	–	–	–	$39.9	$11.3	$6.9	$13.0	$0.7	–	–	$2.1	$0.1	$0.0	–	–	–

Operating Costs																	
Per Tonne Basis																	
Mining Cost - OP	C$/t mined OP	$3.58	–	–	–	$4.27	$3.22	$3.29	$3.46	$3.49	$3.29	$3.52	$3.98	$4.06	$6.20	–	–	–
Processing Cost	C$/t milled	$18.22	–	–	–	$18.22	$18.22	$18.22	$18.22	$18.22	$18.22	$18.22	$18.22	$18.22	$18.22	–	–	–
G&A Cost	C$/t milled	$6.23	–	–	–	$6.23	$6.23	$6.23	$6.23	$6.23	$6.23	$6.23	$6.23	$6.23	$6.23	–	–	–
Annual C$M Basis																	
Mining Cost - OP	C$mm	$807.4	–	–	–	$81.9	$100.6	$100.7	$101.3	$96.7	$93.4	$88.3	$67.7	$56.9	$19.9	–	–	–
Processing Cost	C$mm	$481.3	–	–	–	$36.5	$52.8	$52.8	$52.8	$49.2	$49.2	$49.2	$49.2	$49.2	$40.3	–	–	–
G&A Cost	C$mm	$164.6	–	–	–	$12.5	$18.1	$18.1	$18.1	$16.8	$16.8	$16.8	$16.8	$16.8	$13.8	–	–	–
Total Operating Costs	C$mm	$1,453.4	–	–	–	$130.9	$171.5	$171.6	$172.2	$162.7	$159.4	$154.3	$133.7	$122.9	$74.0	–	–	–
Operating Costs per Tonne Milled - excl. smelter costs & royalties	*C$/t milled*	*$55.0*	–	–	–	*$65.3*	*$59.2*	*$59.2*	*$59.4*	*$60.3*	*$59.0*	*$57.1*	*$49.5*	*$45.5*	*$33.5*	–	–	–
Cash Costs																	
By-Product Basis																	
Cash Cost *	US$/oz Au	$84	–	–	–	$278	$122	($50)	($311)	($120)	$482	$327	$278	($48)	$1,003	–	–	–
All-in Sustaining Cost (AISC) **	US$/oz Au	$138	–	–	–	$0	$190	$5	($260)	($86)	$548	$390	$328	$11	$1,233	–	–	–
Co-Product Basis																	
Cash Cost *	US$/oz AuEq	$509	–	–	–	$551	$525	$457	$383	$384	$733	$664	$525	$477	$1,156	–	–	–
All-in Sustaining Cost (AISC) **	US$/oz AuEq	$547	–	–	–	$610	$573	$494	$415	$407	$783	$710	$565	$516	$1,321	–	–	–
* Cash costs consist of mining cost, processing cost, site G&A, treatment and refining charges & royalties		** AISC includes cash costs plus corporate G&A, sustaining capital and closure costs															
Capital Expenditures																	
Initial Capital																	
Mining Equipment	C$mm	$14.1	$6.9	$2.4	$4.8													
Mining Other	C$mm	$17.6	$1.4	$6.2	$9.9													
Pre-Production Stripping	C$mm	$88.2	$16.8	$24.6	$46.9													
Processing - Secondary Grinding	C$mm	$21.6	–	$6.5	$15.1													
Processing - Fines Flotation	C$mm	$6.6	–	$2.0	$4.6													
Processing - Earth Works	C$mm	$14.1	–	$4.2	$9.9													
Processing	C$mm	$86.6	–	$26.0	$60.6													
Onsite Infrastructure	C$mm	$53.9	–	$16.2	$37.8													
Offsite Infrastructure (Access Road, Water, Power)	C$mm	$37.4	–	$11.2	$26.2													
Processing Indirects (Incl. EPCM)	C$mm	$68.0	–	$20.4	$47.6													
Owners Cost	C$mm	$27.2	–	$8.2	$19.1													
Contingency	C$mm	$52.6	–	$15.8	$36.8													
Sub-Total Initial Capital		$487.9	$25.2	$143.6	$319.2													
Sustaining Capital																	
Mining	C$mm	$40.0				$15.7	$12.5	$6.9	$1.7	$1.0	$1.4	$0.6	$0.1	$0.1	–	–	–	–
Processing	C$mm	$1.29				–	$1.3	–	–	–	–	–	–	–	–	–	–	–
Onsite Infrastructure (Tailings + Water)	C$mm	$6.12	–	–	–	–	–	–	$6.12	–	–	–	–	–	–	–	–	–
Sub-Total Sustaining Capital	C$mm	$47.4				$15.7	$13.8	$6.9	$7.8	$1.0	$1.4	$0.6	$0.1	$0.1	–	–	–	–
Closure Cost	C$mm	$92.4				$4.2	$4.3	$8.5	$8.7	$8.9	$9.1	$9.3	$9.5	$9.7	$9.9	$10.1		
Total Capital Expenditures	C$mm	$627.7	$25.2	$143.6	$319.2	$19.8	$18.0	$15.4	$16.5	$9.9	$10.5	$10.0	$9.7	$9.9	$9.9	$10.1	–	–

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, foreign exchange, capital costs, and operating costs. Table 22-3 summarizes the sensitivity analysis results. Figure 22-2 shows the pre-tax sensitivity analysis findings, and Table 22-4 shows the results post-tax. Analysis revealed that the Project is most sensitive to changes in metal prices and head grade, then, to a lesser extent, to operating costs and capital costs. The project economics are less sensitive to head grades due to the impact of variable mineralogy, lower concentrate grades and penalty elements on concentrate net smelter returns.

Table 22-3: Sensitivity Analysis Summary

Sensitivity Summary	Lower Case	Base Case	Higher Case
Gold Price (US$/oz)	$1,400	$1,550	$1,700
Silver Price (US$/oz)	$20	$22	$24
After-Tax NPV(5%) (C$M)	$1,162	$1,399	$1,635
After-Tax IRR (%)	48.9%	55.5%	61.5%
After-Tax Payback (years)	1.6	1.4	1.2
After-Tax NPV / Initial Capex	2.4 x	2.9 x	3.4 x
Average Annual After-tax Free Cash Flow (year 1-10) (C$M)	$231	$265	$300

Figure 22-2: NPV & IRR Sensitivity Results

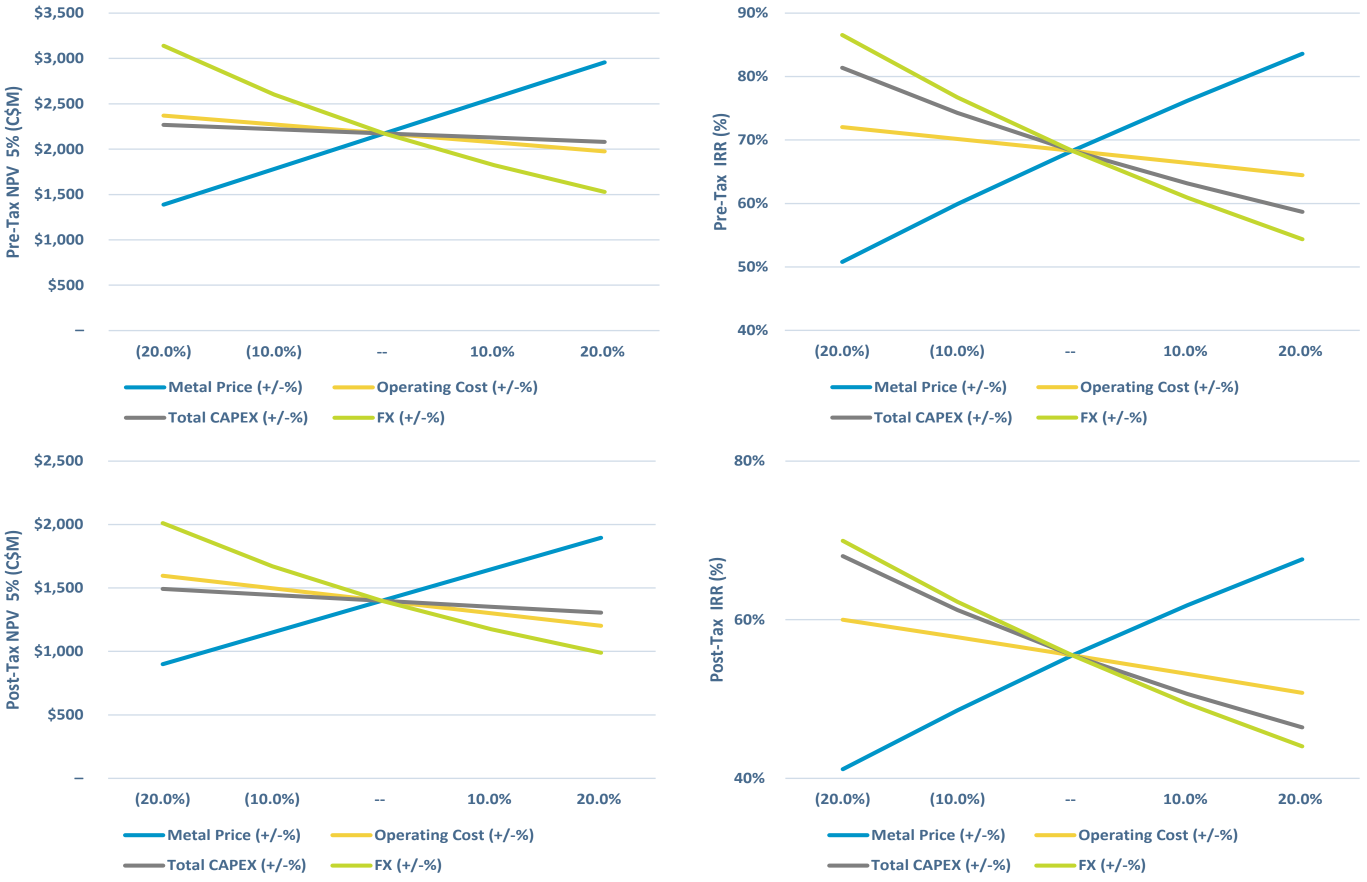


Table 22-4: Pre & Post-Tax Sensitivity

Pre-Tax NPV Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
$18.00	$1,428	$1,698	$1,967	$2,236	$2,685
$20.00	$1,532	$1,801	$2,070	$2,340	$2,788
$22.00	$1,635	$1,904	$2,174	$2,443	$2,892
$24.00	$1,739	$2,008	$2,277	$2,546	$2,995
$26.00	$1,842	$2,111	$2,381	$2,650	$3,099

Pre-Tax IRR Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
$18.00	51.7%	58.1%	64.2%	70.0%	79.1%
$20.00	54.1%	60.3%	66.3%	71.9%	81.0%
$22.00	56.4%	62.5%	68.3%	73.9%	82.8%
$24.00	58.6%	64.6%	70.3%	75.8%	84.5%
$26.00	60.8%	66.6%	72.3%	77.7%	86.3%

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
0.0%	$2,483	$2,873	$3,263	$3,653	$4,304
3.0%	$1,928	$2,239	$2,550	$2,861	$3,380
5.0%	$1,635	$1,904	$2,174	$2,443	$2,892
8.0%	$1,284	$1,503	$1,722	$1,940	$2,305
10.0%	$1,096	$1,287	$1,479	$1,671	$1,991

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
0.0%	56.4%	62.5%	68.3%	73.9%	82.8%
3.0%	56.4%	62.5%	68.3%	73.9%	82.8%
5.0%	56.4%	62.5%	68.3%	73.9%	82.8%
8.0%	56.4%	62.5%	68.3%	73.9%	82.8%
10.0%	56.4%	62.5%	68.3%	73.9%	82.8%

Pre-Tax NPV Sensitivity To FX (CAD:USD)

FX (CAD:USD)	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
0.68	$2,124	$2,433	$2,742	$3,051	$3,566
0.73	$1,863	$2,151	$2,438	$2,726	$3,206
0.78	$1,635	$1,904	$2,174	$2,443	$2,892
0.83	$1,435	$1,688	$1,941	$2,194	$2,616
0.88	$1,257	$1,496	$1,735	$1,973	$2,371

Pre-Tax IRR Sensitivity To FX (CAD:USD)

FX (CAD:USD)	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
0.68	66.8%	73.2%	79.3%	85.3%	94.7%
0.73	61.3%	67.6%	73.6%	79.3%	88.4%
0.78	56.4%	62.5%	68.3%	73.9%	82.8%
0.83	51.8%	57.8%	63.5%	68.9%	77.6%
0.88	47.5%	53.4%	59.0%	64.3%	72.8%

Pre-Tax NPV Sensitivity To Opex

Opex	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
(20.0%)	$1,832	$2,102	$2,371	$2,640	$3,089
(10.0%)	$1,734	$2,003	$2,272	$2,542	$2,990
--	$1,635	$1,904	$2,174	$2,443	$2,892
10.0%	$1,537	$1,806	$2,075	$2,344	$2,793
20.0%	$1,438	$1,707	$1,977	$2,246	$2,695

Pre-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
(20.0%)	60.5%	66.4%	72.0%	77.5%	86.1%
(10.0%)	58.4%	64.4%	70.2%	75.7%	84.5%
--	56.4%	62.5%	68.3%	73.9%	82.8%
10.0%	54.3%	60.5%	66.4%	72.1%	81.0%
20.0%	52.1%	58.4%	64.4%	70.2%	79.3%

Pre-Tax NPV Sensitivity To Total Capex

Total Capex	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
(20.0%)	$1,728	$1,998	$2,267	$2,536	$2,985
(10.0%)	$1,682	$1,951	$2,220	$2,490	$2,939
--	$1,635	$1,904	$2,174	$2,443	$2,892
10.0%	$1,588	$1,858	$2,127	$2,396	$2,845
20.0%	$1,542	$1,811	$2,080	$2,350	$2,799

Pre-Tax IRR Sensitivity To Total Capex

Total Capex	Gold Price (US$/oz)				
	$1,250	$1,400	$1,550	$1,700	$1,950
(20.0%)	67.9%	74.8%	81.4%	87.7%	97.8%
(10.0%)	61.6%	68.1%	74.3%	80.2%	89.6%
--	56.4%	62.5%	68.3%	73.9%	82.8%
10.0%	51.8%	57.6%	63.2%	68.5%	76.9%
20.0%	47.9%	53.4%	58.7%	63.7%	71.7%

Post-Tax NPV Sensitivity To Metal Prices

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Silver Price (US$/oz)	$18.00	$924	$1,096	$1,267	$1,439	$1,723
	$20.00	$990	$1,162	$1,333	$1,504	$1,789
	$22.00	$1,056	$1,228	$1,399	$1,570	$1,854
	$24.00	$1,122	$1,293	$1,465	$1,635	$1,920
	$26.00	$1,188	$1,359	$1,530	$1,701	$1,985

Post-Tax IRR Sensitivity To Metal Prices

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Silver Price (US$/oz)	$18.00	41.9%	47.1%	52.1%	56.8%	64.0%
	$20.00	43.8%	48.9%	53.8%	58.4%	65.5%
	$22.00	45.7%	50.7%	55.5%	60.0%	66.9%
	$24.00	47.5%	52.4%	57.2%	61.5%	68.3%
	$26.00	49.3%	54.1%	58.8%	63.0%	69.7%

Post-Tax NPV Sensitivity To Discount Rate

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Discount Rate	0.0%	$1,623	$1,871	$2,118	$2,365	$2,777
	3.0%	$1,252	$1,450	$1,647	$1,845	$2,173
	5.0%	$1,056	$1,228	$1,399	$1,570	$1,854
	8.0%	$821	$961	$1,100	$1,240	$1,471
	10.0%	$695	$817	$940	$1,062	$1,265

Post-Tax IRR Sensitivity To Discount Rate

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Discount Rate	0.0%	45.7%	50.7%	55.5%	60.0%	66.9%
	3.0%	45.7%	50.7%	55.5%	60.0%	66.9%
	5.0%	45.7%	50.7%	55.5%	60.0%	66.9%
	8.0%	45.7%	50.7%	55.5%	60.0%	66.9%
	10.0%	45.7%	50.7%	55.5%	60.0%	66.9%

Post-Tax NPV Sensitivity To FX (CAD:USD)

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
FX (CAD:USD)	0.68	$1,367	$1,563	$1,759	$1,955	$2,281
	0.73	$1,201	$1,384	$1,567	$1,749	$2,053
	0.78	$1,056	$1,228	$1,399	$1,570	$1,854
	0.83	$929	$1,090	$1,251	$1,412	$1,679
	0.88	$815	$968	$1,119	$1,272	$1,524

Post-Tax IRR Sensitivity To FX (CAD:USD)

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
FX (CAD:USD)	0.68	54.2%	59.5%	64.3%	68.9%	76.2%
	0.73	49.8%	54.9%	59.8%	64.2%	71.3%
	0.78	45.7%	50.7%	55.5%	60.0%	66.9%
	0.83	42.0%	46.8%	51.5%	56.0%	62.8%
	0.88	38.5%	43.3%	47.8%	52.2%	59.1%

Post-Tax NPV Sensitivity To Opex

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Opex	(20.0%)	$1,253	$1,425	$1,596	$1,767	$2,051
	(10.0%)	$1,154	$1,326	$1,497	$1,668	$1,953
	--	$1,056	$1,228	$1,399	$1,570	$1,854
	10.0%	$957	$1,129	$1,300	$1,471	$1,756
	20.0%	$859	$1,030	$1,202	$1,373	$1,657

Post-Tax IRR Sensitivity To Opex

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Opex	(20.0%)	50.6%	55.4%	60.0%	64.3%	71.0%
	(10.0%)	48.2%	53.1%	57.8%	62.2%	69.0%
	--	45.7%	50.7%	55.5%	60.0%	66.9%
	10.0%	43.1%	48.2%	53.2%	57.8%	64.8%
	20.0%	40.4%	45.7%	50.8%	55.5%	62.7%

Post-Tax NPV Sensitivity To Total Capex

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Total Capex	(20.0%)	$1,149	$1,321	$1,492	$1,663	$1,948
	(10.0%)	$1,103	$1,274	$1,446	$1,617	$1,901
	--	$1,056	$1,228	$1,399	$1,570	$1,854
	10.0%	$1,009	$1,181	$1,352	$1,523	$1,808
	20.0%	$963	$1,134	$1,306	$1,477	$1,761

Post-Tax IRR Sensitivity To Total Capex

		Gold Price (US$/oz)				
		$1,250	$1,400	$1,550	$1,700	$1,950
Total Capex	(20.0%)	56.8%	62.5%	68.0%	73.1%	80.9%
	(10.0%)	50.7%	56.1%	61.2%	66.0%	73.3%
	--	45.7%	50.7%	55.5%	60.0%	66.9%
	10.0%	41.4%	46.1%	50.6%	54.9%	61.4%
	20.0%	37.6%	42.1%	46.4%	50.5%	56.7%

22.6 QP Comments on "Item 22: Economic Analysis"

Based on the assumptions and parameters presented in this Report, the PFS shows positive economics.

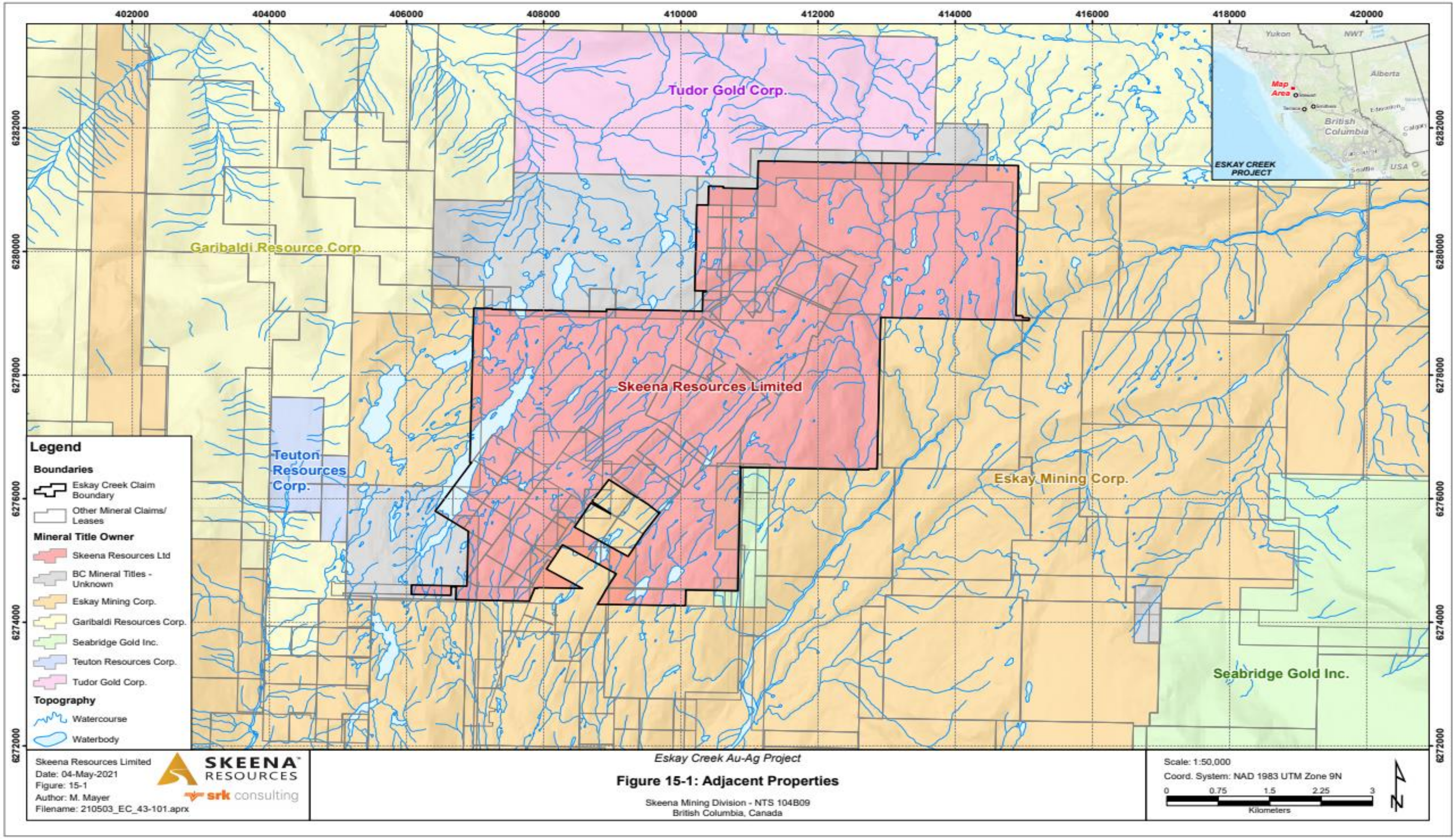
23 ADJACENT PROPERTIES

Notable third-party properties in the Iskut River region are summarized in Table 23-1. Adjacen properties to the Eskay Creek Project are shown in Figure 23-1. The information listed has been taken from documents readily available on the respective company websites and BC MINFILE. Although the information below was publicly disclosed by the Owner or Operator of the adjacent properties, the QP has not audited the associated technical data and the information is not necessarily indicative of the mineralization on the Property that is the subject of this Technical Report.

Table 23-1: Summary Table of Notable Third-Party Properties in the Iskut River Region

Project Name	Owner/Operator	Status	Year	Classification	Cut-off Grade	Tonnes (000)	Average Grades				Source
							Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	
Brucejack	Pretium Resources Inc.	In Production	2020	Proven & Probable	$180/t	15.7	8.4	59.6	-	-	Shaw et al., (2020)
KSM	Seabridge Gold Inc.	Development Project	2020	Proven & Probable	$9/t	2,198	0.55	2.6	0.21	42.6	Threlkeld et al., (2020)
Galore Creek	Teck Resources Ltd./NOVAGOLD Resources Inc. JV	Development Project	2011	Proven & Probable	$11.96/t	528	0.32	6.02	0.59	-	Gill et al., (2011)
Schaft Creek	Teck Resources Ltd./Copper Fox Metals Inc. JV	Development Project	2013	Proven & Probable	$6.6/t	940.8	0.19	1.72	0.27	0.018	Farah et al., (2013)
Red Mountain	Ascot Resources Limited	Development Project	2020	Proven & Probable	3.11 to 4.1 g/t AuEQ	2,544	6.52	20.6	-	-	Bird et al., (2020)
Project Name	**Owner/Operator**	**Status**	**Production Years**	**Million Tonnes**	**Historical Production**				**Au (Moz)**	**Ag (Moz)**	
					Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Mo (%)**			
Snip	Skeena Resources Limited	Past Producer, Exploration	1991-1999	1.308	24.53	9.31	0.02	-	1.03	0.39	BC MINFILE (2018)
Johnny Mountain	Seabridge Gold Inc.	Past Producer, Exploration	1988-1990, 1993	0.227	12.38	19.14	0.44	-	0.09	0.14	BC MINFILE (2018)
PROJECT NAME	**Owner/Operator**	**Status**	**Comments**								
E & L	Garibaldi Resources Inc.	Exploration	Plans for 2021 include two airborne geophysical surveys to expand the footprint of the flagship E&L sulphide zone at Nickel Mountain and establish new discoveries along the 15km strike length between E&L and the nickel-copper outcrops identified at Mount Shirly in 2020. It will also explore for Eskay Creek type gold-silver rich volcanogenic-massive-sulphide (VMS) mineralization.								Company website
KSP	QuestEx Gold and Copper Limited.	Exploration	The Inel prospect located on the 312 km^2 KSP property has 38,000m of drilling in 305 drill holes and 1,240m m of underground development. During 2021 QuestEx plans to drill +2,000m meters for infill drilling and testing of exploration targets as well as develop an independent 43-101 compliant mineral resource.								Company website
Corey and SIB	Eskay Mining Corporation.	Exploration	2021 exploration work will consist of a property wide SkyTEM survey across its 100% owned consolidated Eskay Precious metal rich VMS Project. A +30,000 m drill program is anticipated to commence in June.								Company website
Treaty Creek	Tudor Gold Corp./American Creek Resources Ltd./Teuton Resources Corp.	Exploration	Maiden Resource estimate was released in March 2021 containing a pit constrained resource of 609.8 M tonnes at 0.65 g/t Au, 3.2 g/t Ag and 0.06% Cu (Measured and Indicated) and 139.4 Mt of 0.72 g/t Au, 3.6 g/t Ag and 0.004% Cu (Inferred) at a 0.3 g/t AuEQ cut-off grade as well as Bulk Underground resource of 205.9 Mt at 0.7 g/t Au, 4.6 g/t Ag and 0.07 Cu (Measured and Indicated) and 172.3 Mt of 0.72 Au, 4.4 g/t Ag and 0.006% Cut (Inferred) at a cut off grade of 0.46 g/t AuEQ.								Company website
Kirkham	Metallis Resources Inc.	Exploration	2021 Exploration work consists of geological mapping/prospecting, surface rock-chip and soil sampling, 15 line kms of Induced Polarization ("IP") survey and 5000m of drilling on the Kirkham Property.								Company website

Figure 23-1: Adjacent Properties



24 OTHER RELEVANT DATA AND INFORMATION

Skeena will fund the Project with a combination of equity and financing.

Skeena will provide management for the overall Project and Construction Management and retain an engineering house for overall engineering coordination and procurement assistance.

The intent is to package the majority of the Project scope into discrete design build or design supply contracts with a focus on modularization of the facilities.

Currently, Skeena is not contemplating any self-perform scope for the Project; however, the mine pre-production pit development may be a candidate for an Owner self-perform scope.

There are no plans for build–own–operate (BOO) or build–own–operate and transfer (BOOT) contracts.

Temporary construction infrastructure will be minimized by using the existing infrastructure at the historical site and by installing and commissioning selected permanent infrastructure early in the construction program, such as security, emergency response and medical facilities; communications and permanent accommodation.

Existing mine permits will allow for some work to be executed before the *Mines Act* and *Environmental Management Act* permits are issued approving the full mine plan construction and operation.

After receipt of the provincial and federal permits approving the mine plan and activities, the balance of the construction program will be completed within approximately 18 months, followed by a five month production ramp-up period.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

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

Information from legal experts and Skeena's in-house experts support that the tenure held is valid and sufficient to support a declaration of Mineral Resources and Mineral Reserves.

On December 18, 2017, Skeena and Barrick entered into an Option Agreement on the Eskay Creek Property. On October 5, 2020, Skeena and Barrick agreed to amend the terms of the original option agreement on the Eskay Creek Property. Skeena acquired 100% ownership of Eskay Creek.

Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under BC regulations. All statutory annual reporting obligations have been met.

Royalties are payable on a number of the claims. Barrick retains a 1% NSR royalty on tenements otherwise not subject to royalty payments.

Skeena holds an interest in two surface leases and the Eskay Road access. Skeena will need to acquire surface rights in support of any future mining operations.

Skeena currently holds two water licences. Skeena anticipates needing to apply for additional Water Licences under the BC Water Sustainability Act for the proposed Project.

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

25.3 Geology and Mineralization

The Eskay Creek deposit is generally classified as an example of a high-grade, precious metal-rich epithermal volcanogenic massive sulphide (VMS) deposit; however, it has also been suggested to be an example of a subaqueous hot spring gold–silver deposit.

The understanding of the Eskay Creek deposit settings, lithologies, mineralization, and the geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources and Mineral Reserves.

There is significant remaining exploration potential in the Eskay Creek deposit and environs. Skeena considers that well-defined, mineralized syn-volcanic feeder structures that propagate through the volcanic pile have not been sufficiently explored at depth and along strike. The underexplored Lower Mudstone is situated ~100 m stratigraphically below the more well-known Contact Mudstone and represent a horizon with potential to host similar exhalative style mineralization. Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed to date are appropriate for the style of the deposits in the Project area.

Sampling methods are acceptable for Mineral Resource and Mineral Reserve estimation.

Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards at the time the information was collected.

The quantity and quality of the logged geological data, collar, and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation.

No material factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource or Mineral Reserve estimation.

The sample preparation, analysis, and security practices are acceptable and meet industry-standard practices at the time that they were undertaken and are sufficient to support Mineral Resource and Mineral Reserve estimation.

The Eskay Creek mine initiated QA/QC measures into their sample stream in 1997. With progressive years the QAQC protocol became more comprehensive and detailed. QA/QC submission rates meet industry-accepted standards at the time of the campaign. The QA/QC programs did not detect any material sample biases in the data reviewed that supports Mineral Resource and Mineral Reserve estimation.

The data verification programs concluded that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource and Mineral Reserve estimation.

25.5 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralization type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralization styles found within the various mineralized zones.

Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass, including individual tests to assess variability.

Recovery factors estimated are based on proven metallurgical testwork procedures, appropriate to the mineralization types and selected processing route. A result of the 2021 PFS testwork program was a modified process flowsheet –involving sequential stages of milling and flotation, or an 'MF2' type flowsheet. This was done to isolate soft minerals (including clays) which were impacting the flotation kinetics. In the 2021 PFS flowsheet, a separate fines flotation circuit is now included producing a small portion of the final concentrate.

Results from the 2021 PFS testwork program were used to develop new recovery models for both payable gold and silver as well as penalty elements (arsenic, antimony, and mercury). Variability test results indicated a range of recovery versus final concentrate grade curves – which were attributed to different levels and compositions of NSG as well as pyrite.

Additional testwork is warranted to improve the confidence in the metallurgical performance estimates. An expanded variability testwork program to develop geometallurgical models based on mineral composition should be conducted.

Mineral assemblages will need to be related back to 'proxy' ICP assays so that block models can be populated with metallurgical performance estimates.

25.6 Mineral Resource Estimates

The Mineral Resource estimation for the Project conforms to industry-accepted practices and is reported using the 2014 CIM Definition Standards.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to the density values applied to the mineralized zones; changes to geological shape and continuity assumptions; potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures; changes to the input values used to generate the AuEQ cut-off grade; changes to metallurgical recovery assumptions; changes in assumptions of marketability of final product; changes to the conceptual input assumptions for assumed open pit operations, changes to the input assumptions for assumed underground operations; variations in geotechnical, hydrogeological and mining assumptions; changes to environmental, permitting and social license assumptions.

25.7 Mineral Reserve Estimates

The Mineral Reserve estimation for the Project conforms to industry-accepted practices and is reported using the 2014 CIM Definition Standards.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes to recovery assumptions based on further metallurgical testwork and determination of mill feed blending; changes to marketing terms due to future negotiations; effective execution of water diversion to allow access to northern portion of the pit; effective excavation and control of open pit slopes, and maintaining bench advance rate by dealing with ore separation near underground workings and management of snow and rain conditions.

25.8 Mine Plan

25.8.1 Geotechnical Considerations

The current geotechnical dataset is considered adequate for PFS-level designs. The Project area is within a region that is seismically active, and seismicity is incorporated into design considerations. Rock quality varies from good to extremely poor, and is generally related to lithology, and the degree of, and proximity to, local and regional faulting; and rock quality can change rapidly over short distances. Inter-ramp slope angle recommendations range from 34–46º.

The proposed North pit will intersect and mine into the historical underground workings at approximately mid-slope height on the mid to north side of the pit. This will result in increased risks for safely mining in this area and prescriptive plans will need to be developed to adequately mitigate these risks to acceptable levels.

25.8.2 Hydrological Considerations

The regional groundwater regime is most likely controlled by the regional groundwater flow system, and from seasonal snow melt. The regional faults likely provide high permeability recharge pathways and groundwater storage areas; however, the rock units themselves are highly fractured and even away from major faults constitute fractured aquifers. Faulted andesite most likely provides the highest permeability and highest storage capacity of all the rock units.

The planned ultimate pit bottom will be at 714 masl, and therefore only about 50 m of flooded working is likely to require dewatering.

Pit stability can be managed by progressive dewatering of the ground behind the pit slope with vertical or horizontal boreholes. Mudstones may require special attention as matrix pore pressures could remain elevated despite successful dewatering.

25.8.3 Mine Plan

Each pit phase was designed to accommodate the proposed mining fleet. Mining will occur on 8 m benches with catch benches spaced either 8 or 16 m vertically depending on lithology type. The haul roads will be 30.2 m in width with a road grade of 10%.

The mine schedule plans to deliver 26.4 Mt of mill feed grading 3.37 g/t Au and 94.4 g/t Ag over a mine life of 10 years. Waste tonnage totalling 212 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is estimated at 8.0:1. The mine schedule assumed a maximum of 2.9 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. A maximum descent rate of eight benches per year per phase was applied.

The proposed mine life includes three years of pre-stripping and 10 years of mining. Mill feed will be stockpiled during the pre-production years, with three grade stockpiles envisaged. A technical sample will be mined in Year -3 so that process performance of the mill can be evaluated on a bulk sample.

The mine equipment fleet is anticipated to be leased to lower capital requirements.

There will be three WRSFs that will store the NAG waste. PAG waste will be sent to the TMSF to be submersed below water.

25.9 Recovery Methods

The plant will process material at a nominal rate of 2.9 Mt/a for Years 1 to 4 and 2.7 Mt/a for the remaining years with an average head grade of 3.2 g/t Au and 94 g/t Ag. The ore becomes harder and more competent after the first four years of operation.

The plant is designed to operate two shifts per day, 365 days per year with an overall plant availability of 92%.

The process plant flowsheet designs were based on testwork results and industry-standard practices.

The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

25.10 Infrastructure

25.10.1 Site Facilities

Infrastructure to support the Project will consist of site civil work, site facilities/building, a water system, and site electrical. Site facilities will include both mine and process facilities:

- Mine: administration offices, truck shop and warehouse, tire repair shop, mine workshop, mine dry, fuel storage and distribution, mobile equipment, temporary camp for accommodating construction crew, permanent camp facility and miscellaneous facilities;

- Process: process plant, crusher facility, process plant workshop and assay laboratory;

- Services: security, information technology, potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

Multiple options for the export of concentrate were studied, with two options through Stewart identified as preferred. Both transportation options have similar overall logistics costs for the movement of concentrate from the mine into a ship. The bulk carrier vessel would be the same in each case and would transport the concentrate to a terminal facility nearest the preferred smelter location in southern China.

Construction materials and mine consumables would be transported to site via existing highways and access roads. However, some specialized equipment may come through the SBT site, which has a general cargo dock.

25.10.2 Tom MacKay Tailings Storage Facility

The existing TMSF was selected as the preferred NAG and PAG tailings and PAG waste rock storage option since it is permitted as a waste storage facility and is currently still one of the Best Available Technologies (BAT) for storage of PAG materials. The TMSF will have sufficient capacity to contain 76.7 Mt of NAG and PAG tailings and PAG waste rock and will be constructed in two phases over the LOM based on storage and operating criteria.

The operational plan of the TMSF is to deposit slurry tailings at the south end of the facility and PAG waste rock at the north end of the facility. PAG waste rock deposition will use a causeway approach, depositing waste across the facility from west to east then the upper 5 metre will be removed and deposited on the south side of the causeway for the PAG waste rock is submerged a minimum of 3 m below the water surface.

Tailings will be slurried from the process plant to the TMSF by way of a pipeline, which would extend onto the TMSF to a floating barge with a weighted spigot located near the bed of the tailings to promote settling of the tailings.

25.10.3 Water Supply and Management

Pit dewater will be sent directly to a WTP, then to D7 polishing ponds, and finally to Ketchum Creek during pre-production. The water treatment plant's maximum capacity has been designed to accommodate the pit water with additional treatment capacity. The WTP has a capacity of approximately 150 L/s, which supports pre-production operations. Once the tailings pipeline is installed and operations begin, pit water will report to the tailings mixing tank at the plant and sent with the tailings in the tailings transportation pipeline to the TMSF. As the open pit becomes larger, pit dewatering flow rates will increase. The pit dewater flow to the tailings mixing tank will range from 65.5 to 376.3 L/s during the mine life.

The WDW water management includes both contact and non-contact water management structures. The facility is located in a relatively small watershed. The non-contact water will pass underneath the facility in a rock drain that converts to 2 solid wall HDPE pipes that discharge water directly into Tom MacKay Creek. The surface contact water from the WRSF will be conveyed in both temporary and permanent diversion channel to contact water 5 Pond to remove sediment 10 microns and above prior to releasing water into Tom MacKay Creek. The contact water management system was designed for 1:200 year event and the non-contact water management system for 1:475 year event.

For TMSF there are no diversion works. There will be inflow of water into the TMSF from direct rainfall and snow and runoff from the surrounding catchment into the TMSF. To maintain flow from TMSF during operations there is a penstock that will release a flow to maintain stream flow in Tom MacKay Creek. However, during peak runoff (May through October) a portion of the flow will be held to increase the water level within TMSF to ensure there is a minimum of 3 m of water cover over the tailings and waste rock during operations.

The industrial water requirements will come from the TMSF, which are estimated to be 113 L/s to be used in mineral processing. Fresh/fire water will be pumped from a local fresh water supply well into a fresh/fire water tank. Water for the accommodations camp will be from a well and treated to become potable.

25.10.4 Power

Project power will be provided through a 20 km long 69 kV overhead transmission line. The source of power will be from the Volcano Creek 287 kV substation. The estimated power demand for the Project is 21 MW.

25.11 Environmental, Permitting and Social Considerations

25.11.1 Environmental Considerations

The Project will be designed, constructed, operated, and decommissioned to meet all applicable BC environmental and safety standards and practices. Skeena will develop and implement an EMS that defines the processes by which compliance will be met and demonstrated. The EMS will include ongoing monitoring and reporting to relevant parties at the various project stages.

The main waste management issue for the Project is the prevention and control of ML/ARD from the tailings and waste rock.

Site water management will be a critical component of the Project design. Strategies for water management include collecting surface water from disturbed areas (mine-contact) to manage surface water erosion; recycle mine-contact water whenever possible; treat mine-contact water as required; and monitor water quality to meet discharge standards prior to discharge.

25.11.2 Closure Considerations

The mine closure strategy for the mine will be to have a stable, revegetated site with best mitigation of potential ML/ARD and water quality risks that is consistent with the Tahltan and Skeena's agreed Social and Environmental Design Principles and post-mining end land uses. A Closure and Reclamation Plan will be developed during the permitting process to achieve end land use objectives (e.g., wildlife habitat), in consideration of Indigenous interests. Closure planning will include Indigenous groups and stakeholders to determine post-mining land use objectives and supporting strategies, including addressing regulatory requirements. Achieving the desired outcomes will be an iterative process during the design and permitting process and incorporate social, environmental, engineering, technical and Tahltan criteria.

25.11.3 Permitting Considerations

The proposed Project is anticipated to undergo a concurrent EA/IA. Since the Eskay Creek Mine has two existing Certificates, one or both will be amended through a substituted EA/IA process. For the proposed Project, Skeena will undertake a substituted process to amend an existing EAC or obtain a new EAC. The process to follow for the EA/IA is being developed with the provincial and federal regulators, the Tahltan Nation and Skeena, based upon the legislative steps, criteria, and procedures. In addition to obtaining the EAC, the Project will require permits and authorizations in accordance with provincial and federal legislation and regulations prior to construction and operation. No permits for Project commercial development will be issued before an EAC is obtained. Consequently, Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction.

No technical or policy issues are anticipated for obtaining the required project permits and approvals, given the previous long mining history.

25.11.4 Stakeholder Considerations

Community and socio-economic impacts of the Project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers.

Provisions for consultation with Indigenous Nations and the public are a component of the provincial and federal legislation for both the EA processes and permitting activities. Skeena is developing an EP for the Project as required by the provincial and federal EA processes. The EP will be submitted with the IPD to begin the EA process. Ongoing and future engagement and consultation measures by Skeena are driven by best practices as well as Skeena's internal company policies. These measures will at a minimum comply with federal and provincial regulations.

Skeena recognizes engagement and support of the Project from Indigenous Nations from initial project design until post-closure is critical for the success of the Project. Skeena is and will consult with local Indigenous Nations to gain that support, yet also recognizes this is part of the EA process at both the provincial and federal level.

The Project is located within the traditional territory of the Tahltan Nation and the asserted territory of the Tsetsaut Skii Km Lax Ha. The historical environmental process and subsequent expansions included consultation with the Iskut Band, Tahltan Band, and the Tahltan Central Government. Project traffic will use Highways 37 and 37A which pass through the Nass Area and Nass Wildlife Area (as defined by the Nisga'a Final Agreement) and the traditional territory of the Gitanyow Nation.

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will form a project specific working group at the early stages of the EA process, which will include representatives from many government groups. Skeena will consult with the working group on project-related developments during the EA process. Skeena will consult with the public and relevant stakeholder groups, including tenure holders, businesses, economic development organizations, businesses, and contractors (e.g., suppliers and service providers), and special interest groups (e.g., environmental, labour, social, health, and recreation groups), as appropriate.

25.12 Markets and Contracts

The concentrate as proposed is a complex gold concentrate with relatively low gold content and elevated levels of arsenic, mercury, and antimony. Deleterious element assays are notably elevated in the first few years of mine life (arsenic in Years 1 and 2 and mercury in Years 1 to 3) before dropping to values which fall within typical industry expectations.

Concentrate quality parameters are based on the results of ICP analysis of gold–silver concentrates produced during the variability flotation testwork at BaseMet.

Ausenco provided the expected concentrate composition and tonnage at Eskay Creek to four concentrate marketing specialists (WoodMac, Open Minerals, Hartree, and Trafigura). Gold smelter contract estimates were received from all four marketing specialists and one copper smelter contract estimate was provided by WoodMac. For the purposes of the 2021 PFS, the smelter terms provided by Open Minerals were used in the economic analysis.

Based on the predicted analysis, the Eskay Creek concentrates will be readily saleable. The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across several buyers since individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable effluent limits. An alternative option is to sell the concentrate to traders who may be able to buy all concentrate and spread distribution across a range of end customers, potentially including a mix of gold and copper smelters. Expectations of NSR may be achieved and penalties for deleterious elements may be minimized. Concentrate grades for gold, silver, mercury, antimony, and arsenic are expected to vary throughout the life of mine which will impact the marketability and net revenue. Concentrate volumes are expected to decrease over the mine life as the feed grade decreases. This results in an easier blending of the deleterious elements out of the concentrate over time.

The most likely market for the concentrate is China, where the material will be imported as a gold concentrate (exceeding the minimum gold content criterion) and will therefore not be subject to arsenic import limits that would be imposed on base metal concentrate imports. The Chinese market offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate. Chinese gold smelters can typically monetize antimony at the levels found in the Eskay Creek concentrates.

No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

The base case for logistics is moving the concentrate by bulk bags to Stewart, where they will be loaded into containers for export via container vessels to China. The projected overall transport cost is US$137/t.

The economic analysis included in the 2021 PFS is based on a two-year average of gold and silver prices as of March 31, 2021. The estimated prices are based on the daily closing price of gold and silver prices from the LBMA.

25.13 Capital Cost Estimates

The estimate is based on assumptions of an exchange rate of US$0.78:C$1.00, is expressed in Canadian dollars, has a base date of Q1, 2021, and has an accuracy range of -20% to +30%.

The costs can be broken down as follows:

- Initial capital costs: include the costs required to construct all of the surface facilities, and open pit development to commence a 2.9 Mt/a operation. The initial capital cost is estimated to be $487.9 M;
- Sustaining capital costs: include all the costs required to sustain operations, with the most significant component being open pit mine development. Sustaining capital costs total $47.4 M over the LOM;
- Closure costs: include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $92.4 M.

25.14 Operating Cost Estimates

Operating cost estimates are based on a combination of first-principal calculations, experience, reference projects and factors. Operating costs include provision for mining, processing, process contingency, G&A, and water treatment.

Over the LOM, process costs will average $145.34 M/a, and $55.01/t processed.

25.15 Economic Analysis

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. The cashflow model included key assumptions:

- Construction period of three years;
- Mine life of 9.8 years;
- Base case gold price of US$1,550/oz and silver price of US$22/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.78 (US$/C$)
- Cost estimates in constant Q2 2021 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 2% NSR; NSR payments will total C$109 M in royalty payments over LOM;
- Capital costs funded with 100% equity (i.e. no financing costs assumed);
- All cash flows discounted to start of construction;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold concentrate into the international marketplace;
- No contractual arrangements for smelting or refining currently exist.

The Project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%);
- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

Total tax payments are estimated to be C$1,145 M over the LOM.

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$2,174 M, the internal rate of return IRR is 68.3%, and payback is 1.3 years. On an after-tax basis, the NPV5% is C$1,399 M, the IRR is 55.5%, and the payback period is 1.4 years.

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, capital costs, and operating costs. Analysis revealed that the Project is most sensitive to changes in metal prices and head grade and then, to a lesser extent, to operating costs and capital costs. The project economics are less sensitive to head grades due to the impact of variable mineralogy, lower concentrate grades and penalty elements on concentrate net smelter returns.

25.16 Risks and Opportunities

25.16.1 Risks

25.16.1.1 Geology and Resource Estimates

The current understanding of the distribution variability of elements that can be deleterious in concentrates is based on incomplete data, as epithermal and base metal elements were only selectively sampled in the legacy drill programs. It is expected that information obtained from the planned drill programs will provide more complete data on elemental distributions within key lithologies and domains, which in turn is likely to affect the domain and grade-shell outlines as interpreted in the current Mineral Resource estimate. The risk is that the variability is much higher than currently estimated, and that the model underestimates the deleterious elemental tonnages and grades that the 2021 PFS mine plan and concentrate marketability assumptions are based on.

25.16.1.2 Mineral Resource Risk

Risk management was incorporated into the Mineral Resource estimates by means of identifying, assessing, and controlling variability in the model in advance of selecting and delineating appropriate resource classification categories. Several factors, including grade range and continuity, domain thickness, and geological trends are inherently variable in geological models. In addition, the distance between drill core samples, the direction between samples, and volume above a cut-off grade vary spatially within the mineralized bodies. With sound knowledge of the nature and arrangement of the supporting data, categories were quantified and delineated into areas of similar confidence. Drill sample spacing varies by mineralized domain and the classification of Mineral Resource estimates was assigned by the level of confidence, primarily based on drill core sample spacing. Higher confidence at Eskay Creek is associated with closer-spaced drilling and lower confidence is associated with widely-spaced drilling.

Risk assessment defined herein considers the payable elements, gold and silver, which are appropriately defined for reasonable prospects of eventual economic extraction. For these elements, risk is associated with all levels of classification in the Mineral Resource estimates; however, the greatest risk is associated with Inferred Mineral Resource. There is also a risk associated with the suite of deleterious elements (arsenic, mercury, and antimony) that are associated with gold and silver mineralization. The deleterious elements lack full assay coverage and are not fully understood in terms of revenue and environmental impacts.

25.16.1.3 Mining

Mining through voids during open pit operations is a generally manageable risk where such voids are known to exist. However, unidentified voids may exist, and present a risk to mine and production plans if alternate schedules have to be derived, or new safety measures implemented.

It is probable that unfavorably oriented geological structures are present locally within various slope pit sectors resulting in local instability, particularly given the size and extents of the pit and the observed variability in discontinuity orientations; it is assumed at present that small bench-scale failures developed along these features can be managed with careful blasting techniques and regular berm maintenance/clearing, wherever access is possible.

Both seismic loading and multi-bench-scale to pit-scale structures have the potential to significantly affect overall pit slope stability. The current status and impact of these features are largely unknown. The inclusion of hypothetically adversely-oriented faults and bedding planes in the stability analyses indicates potential FOSs of <1.0, particularly with seismic loading applied. Additional geotechnical investigations are warranted to further determine the location and character of inter-ramp to global-scale conditions and features that may impact stability and mining outcomes.

The sampling program designed to segregate PAG and NAG waste rock must be adhered to during mining operations to minimize economic and water quality impacts.

The WRSF design assumes that no geomembrane liner will be required based on a limited geochemical program. If, with further data, such a liner is required or other mitigation measures are required this will affect the mining capital cost estimate.

More detailed geotechnical information is required to support the assumption in the 2021 PFS that mining will extend across Tom Mackay Creek. These data will be used to develop a more detailed water diversion tunnel design and strategy. Geotechnical information may require realignment of the tunnel to avoid potentially problematic material or need additional support requirements which may alter the cost attributed to the tunnel. There is a risk that this design could result in mining capital and operating cost increases.

Detailed operating procedures will need to be established to ensure the PAG rock exposure to air is minimized when placing PAG material into the TMSF.

The support equipment fleet will be responsible for the usual road, pit and WRSF maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management. This is considered an important, but manageable operating risk to meet production targets.

The mill feed blend will require close management of deleterious elements and the effect on resulting mill performance. Proper tracking of deleterious elements in the mine dispatch system will be necessary to ensure the mill can be fed a blend of material that can be processed with lower risk to mill recovery and concentrate revenue.

25.16.1.4 Process

Flow sheet development, locked cycle, and variability testwork has shown the process flow sheet to be robust and stable at a pre-feasibility level under laboratory conditions. The ability to produce a saleable gold concentrate has been confirmed.

Although the Project is considered viable, there process design assumed for the 2021 PFS has some risks identified that could impact delivery or economics and these need to be managed and mitigated by additional testwork and studies. The key aspects of the Project presenting most execution risk are:

- Variability in samples need to be tested to ensure there is a reasonable 3D range of test results covering the mineralization that will be mined in the envisaged LOM open pit mine plan;
- On time delivery of critical packages (crushers, mill, flotation circuits);
- Piloting data should be obtained to confirm that the DFR cells will perform as projected;

- Further flotation density optimization testwork is required with Woodgrove to validate process design criteria of DFR cells as current testwork suggests very low pulp density (20%) in order to achieve good results;

- Sufficient data on settling kinetics should be obtained to support DFR performance assumptions. This should include flocculant optimization studies;

- Flotation data should be collected to confirm that the low pulp densities required for successful DFR operation can be achieved;

- Smelter contract terms will change over the development of the Project. Selecting the best smelter terms at this phase will improve the economics.
    - Selection of the best smelter terms are complex and requires a techno-economic model to optimize concentrate grade and recovery for the available terms.

Depending on results, the testwork could indicate that the selected DFR parameters are too optimistic for the mineralization to be treated, or that there can be improvements to the assumptions in the 2021 PFS resulting in lower operating costs and better recovery performance.

The smelter terms that can be obtained over the LOM, including payability and penalty assumptions, are likely to be more complex than currently represented in the 2021 PFS. A focused study that will evaluate projected concentrate grades (payable and penalty) and recovery forecasts is required to provide additional support for assumptions as to smelter terms that would be available to the project. This study could impact project economics negatively if the study indicates higher penalties or less favourable terms than assumed in the 2021 PFS. Conversely, there could be a positive impact if lower penalties and more favourable terms are indicated than assumed in the 2021 PFS.

Solid/liquid separation issues could increase process costs due to larger thickeners and filters and use of flocculant.

Higher mass pull to final concentrate might result without careful control of the grinding pulp chemistry (e.g., stainless-steel media).

25.16.1.5 Infrastructure

A portion of the access road passes through topography which is known to have an elevated geohazard (e.g., avalanche) risk. There is potential for geohazard events to temporarily halt movement along the access corridor. This could affect supply logistics and could result in temporary halts to mining and/or processing operations.

Until there is an agreement in place to connect the powerline at Volcano Creek, there is a risk that the power would have to come from onsite liquified natural gas power generation or a powerline from Bob Quinn which is farther away than Volcano Creek. This would affect the power capital and operating cost assumptions as envisaged in the 2021 PFS.

A WTP at the discharge of the TMSF has not been included in the scope of the 2021 PFS. Further testing will be done in the next phase to confirm there is no requirement for water treatment. If required, this would affect the water-related capital and operating cost assumptions in the 2021 PFS.

A PAG waste rock deposition plan into the TMSF was developed for the 2021 PFS. A detailed operating procedure will need to be established in the next phase to ensure the PAG waste rock exposure time to air is minimized when placing PAG material into the TMSF to prevent acidification and metal leaching. A change in the deposition plan for the PAG waste rock could result in capital and operating cost increases.

Additional testing is required to understand settling times of tailings and fine particle material from the waste rock placed into TMSF. Currently, the practice is to place the tailings at the south end of the facility to allow for additional settling time and the waste rock has assumed the fine grain particles will settle more quickly. A turbidity fence is used to reduce the potential for turbid water to discharge from the facility. If the settling tests indicate that additional measures are required to prevent the downstream migration of suspended solids, this could result in capital and operating cost increases.

Deposition of the PAG waste rock during winter operations needs further study to ensure there is a sufficient ice-free zone on the causeway to deposit the PAG waste rock. The dragline or an additional dragline could be used to keep an ice-free zone for deposition operations. If needed, this would result in capital and operating cost increases.

25.16.1.6 Environmental, Permitting and Social

The provincial and federal regulatory processes under recent legislative changes may influence overall timelines to amend the existing permits and obtain new permits for the Project. Additional work is underway to support permit amendments and new permit applications, including environmental baseline data collection, mine plan details, and environmental assessment and consultations.

The current permits for the Eskay Mine do not consider operations at the scale contemplated in the 2021 PFS. Additional work will be required to support permit updates and amendment applications, which will include environmental baseline data collection, environmental assessment and proposed mine plan and reclamation and closure plan.

The Project is within the territories of Indigenous groups. Agreements with such groups that may be affected by the envisaged project remain to be negotiated. If such agreements include royalty or similar payments, this could result in changes to the assumptions made in the economic analysis. Skeena actively engages with communities of interest and Indigenous Peoples to understand potential Project effects and plan mitigative approaches collaboratively.

25.16.2 Opportunities

25.16.2.1 Exploration

Exploration activities may result in definition of additional mineralization that could support Mineral Resource estimates.

Upside potential for further mineralization discoveries exists within the Lower Mudstone and Even Lower Mudstone units in the Lower Package. These units are typically at depth below the current limits of the proposed open pit.

25.16.2.2 Resource Estimation

There is upside Project potential if mineralization currently classified as Inferred can be upgraded to higher confidence categories. There is also potential for mineralization that is currently outside the estimate boundaries, or discovery of previously unknown mineralization, to be included in estimation with support of drilling and testwork.

25.16.2.3 Mineral Resource Opportunity

The most significant upside for Eskay Creek is the potential for; 1) conversion of Inferred Mineral Resources to Indicated Mineral Resources, and possibly, with additional work, to Mineral Reserves in the future; 2) upgrade of unclassified resources within the Low-Grade Envelope to Inferred Resources, and 3) discovery of additional mineralization that may support Mineral Resource estimation.

The 2021 drill program should focus on identifying mineralization that is proximal and lateral to currently defined mineralization domains, to generate maximum ounces with minimal cost. In addition, potential to increase the Mineral Resources at the margins, and within the low-grade envelope is possible with additional infill drilling and mineralization domain development. Furthermore, with the recent discovery of the Lower Mudstone and Even Lower Mudstone units in the Lower Package, which mostly occurs below the level of the current resource pit shell, additional drilling is suggested to better define the full extents of these domains.

25.16.2.4 Mining

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value. Additional hardness testing is likely to be available in the next study stage to inform more detailed throughput management and potentially higher value.

There is potential for improved slope design, when additional geotechnical data such as waste rock strength and joint orientations, are available from drill testing. Steeper pit slopes would reduce the cost associated with waste stripping and provide an opportunity to improve economics.

Slightly higher bench heights could provide an opportunity to better match blasting performance with mine productivity. This will be dependent on the ability to separate ore near underground workings. Higher mine production rates could result in lower mine operating costs and also lower risk to the achieve the mine schedule.

As the metallurgical and marketing information is better understood, the use of stockpiles will likely be modified to allow for improved blending of mill feed material. Stockpile space is fairly limited near the crusher, so a location for lower value material would be useful to ensure high value stockpiles have adequate capacity. This could result in better process performance and improved project economics.

Ongoing test work results will be monitored to see if a portion of the PAG waste material can be effectively neutralized by blending with NAG waste. The ability to blend a portion of this material could result in less PAG material being sent to the TMSF and therefore lower waste haulage and deposition costs.

25.16.2.5 Process

Higher gold and silver recoveries may be obtained from lower head grade samples with optimized flotation conditions. Higher recoveries will enhance revenue from lower grade areas of the deposit.

Pre-concentration by screening and/or bulk sorting might reject waste material and increase plant feed grade. Waste material may also be upgraded with these technologies and converted to ore grade.

Incorporation of gravity concentration in the grinding circuit may provide an opportunity to remove free gold prior to flotation and direct it to final concentrate. This may potentially increase gold recovery and reduce operating costs.

An improved project schedule may be achievable due to shortened equipment leads times, fewer bulk materials, and resulting reduction in construction and installation of the DFR cells. A reduction in the project schedule will reduce capital costs.

Investigations into the geometallurgy may lead to flowsheet optimization for the various geological and mineralogical zones for the comminution, flotation, and regrinding configurations.

Albino Lake is a subaqueous repository for mine waste rock and tailings used by the previous operators. Initial drilling has indicated elevated gold values in this material. Testwork is required to determine gold can be economically extracted as part of an overall evaluation. This material could be incorporated into the mine plan and potentially result in an improvement in Project economics.

25.16.2.6 Environmental and Social

Potential environmental and social opportunities within this Project include the following:

- Collaboration with Indigenous Peoples to develop the Project Closure and Reclamation Plan to meet long term Indigenous End Land Use objectives will gain support for the Project and reduce post-closure cost estimate uncertainty;
- Rationalization of regulatory timeframes in a project charter agreement with regulators and Indigenous peoples can support predictable Project permitting timelines in parallel with testing programs and site development;
- Geochemical baseline studies to refine NPAG/PAG classifications and material segregation may help optimize waste management costs, design, and complexity. Commenced early, geochemical studies improve regulator confidence in modelled outcomes of post-closure environmental management.
- Assessment of energy efficiencies and fleet/machinery composition may present opportunities to reduce emissions over mine life; and
- Incorporation of Indigenous perspectives and values on how mining development occurs on the landscape and its effects on land and water will be pursued throughout the Project life resulting in a Project viewed as meeting sustainability goals by Indigenous communities.

25.16.2.7 Infrastructure

The TMSF has significant expansion capability (> 20Mm^3 of waste materials) if additional mineralization that could support incorporation in the mine plan is discovered. The capital and operating costs would be significantly less than constructing a new storage facility.

25.16.2.8 Marketability

There is upside potential for the Project if the planned drill programs more comprehensively document deleterious elemental distributions such that the levels of these elements, in particular arsenic and mercury, can be minimized in the concentrate to below smelter penalty thresholds.

25.17 Conclusions

Based on the assumptions and parameters presented in this Report, the 2021 PFS shows positive economics. The 2021 PFS supports that more detailed studies are warranted.

26 RECOMMENDATIONS

26.1 Introduction

The recommended work program is divided into two phases. The phases can be conducted concurrently, but some portions of the phase 1 work plan would be incorporated into the phase 2 recommendations.

The first recommendations phase totals approximately $11.49 M and would be completed in support of more detailed studies. The program will consist of drilling; determination of whether bulk ore sorting could potentially be implemented at the pre-mining stage; a study to determine if a relationship between rock mass structure and head grade exists; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second phase is estimated at approximately $4.6 M, and will consist of project environmental, permitting, and social de-risking activities. The program will consist of baseline and targeted environmental studies; environmental assessment; support work for operating permits and supporting applications; consultation and negotiations with Indigenous groups; other stakeholder engagement; and updates to the water balance.

26.2 Phase 1

The planned Phase 1 work program is set out in the following subsections, by major discipline area.

26.2.1 Drilling

A total of 209 drill holes for a total of 14,267.27 m have been drilled at Eskay Creek since the database supporting the Mineral Resource estimate was closed out.

Skeena plans to drill a further 98 drill holes (approximately 16,500 m), using skid-mounted drill rigs and helicopter support. This program is estimated with all-in drilling costs of $475/m, to be approximately $8.0 M. At program completion, the intent is to update the block model and resource estimate.

26.2.2 Sampling and QA/QC

The QA/QC measures implemented in the 2018–2019 drill programs should be retained for future drill campaigns.

Lithological, alteration, mineralization and structural data captured during these programs should continue to be used to refine geological understanding and interpretations and inform the resource modelling process.

The current SG sampling process at Eskay Creek is to conduct on-site density determinations using the water displacement method. Future drill programs should adopt a method of independently analysing a percentage of the SG samples.

With the recently completed LiDAR survey, there is the opportunity of incorporating the results into future structural modelling interpretations. The recent LiDAR results will also be used as the final topographic surface in future models.

Geotechnical inspections of the underground workings will need to be completed to determine rock conditions immediately adjacent to, and within, the mined-out solids; measurements that are needed for adjusting the depletion buffer zone appropriately.

26.2.3 Metallurgy

Sample selection for future mining studies should reflect mineralization that would be treated in the first five years of the mine life. Variability samples are required to understand the responses of the various mineralized zones to flotation kinetics and contaminant correlations.

For the MF2 flowsheet selected for the PFS study, additional testwork is warranted to improve the confidence in the metallurgical performance estimates. In addition, an expanded variability testwork program will be necessary to develop geometallurgical models based on mineral composition. Mineral assemblages will need to be related back to 'proxy' ICP assays so that block models can be populated with metallurgical performance estimates.

The list of recommended testwork includes:

- Variability testing program for comminution and flotation geometallurgical modelling;
- Woodgrove DFR bench-scale testing to confirm mass pull performance vs. lower % solids;
- Locked cycle testing on a number of composite samples to confirm circuit stability;
- Pilot plant testing to generate representative samples of final concentrate for customer/smelter evaluation;
- Vendor testing for solid/liquid separation evaluation.

This will require approximately 1.5 t of half core samples. It is expected the next phase of testwork will cost approximately $500 k with pilot plant work to cost an additional $100 k.

26.2.4 Materials Handling

Material handling test work is recommended for design of bins, chutes, conveyors, and stockpile drawdown. This program is estimated at $65 k.

26.2.5 Mine Geotechnical

A nominal six to eight hole geotechnical drilling and rock mass characterization program is proposed to support more detailed studies, including targeted drilling of current data voids, particularly the portions of the higher wall sectors, North Pit northern sector slopes and diversion tunnel, to include discontinuity orientation measurements (where possible), sampling for additional laboratory strength testing, and televiewer surveys. The core holes should be drilled using a triple tube core barrel to preserve the integrity of the core while drilling and retrieving. The program, assuming eight holes are completed, is estimated at $1.2 M, based on 2,000 m of drilling at $600 /m.

Core orientation (using the ACT, EZ-Mark, or equivalent systems) and/or optical or acoustic televiewing of select holes will be needed to determine discontinuity data. Point load tests should be completed at regular intervals of drill core (~once per run to domain intercept scale). Additional laboratory testing is recommended, including uniaxial compressive strength testing (with strain measurements), tri-axial strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill materials. The combined orientation and testwork program is estimated at $50 k.

The following mining geotechnical evaluation tasks are recommended:

- Review and compilation of geotechnical data; updating the existing 3D lithological and/or structural models to incorporate the results of any additional exploration drilling and/or an improved understanding of the deposit geology.
- Laboratory testing to investigate anisotropic/heterogeneous rock mass strengths should be investigated, defined, and utilized as appropriate to capture the conditions in directions parallel to structural fabric and orientations, and with respect to pit slope sector orientations.
- Updating of geotechnical domains, slope designs sectors, stability models, slope design recommendations.
- Preliminary diversion tunnel and portal design including stability assessments and ground support designs.

These studies are estimated at $400 k.

26.2.6 Mine Studies

The following areas should be addressed during more detailed studies. These studies are collectively estimated at $400 k.

26.2.6.1 Grade Control

The 2021 PFS assumed that RC and blasthole sampling would be the preferred grade control methods. Sample sizes, methodology of sample selection and assaying procedures need to be defined to properly assess the cost of grade control.

26.2.6.2 Geology Model Improvement

The 2021 PFS model has a 0.2 m buffer around the old stopes which is modelled as waste. This needs to be examined further to confirm whether this assumption is valid and assess the impact on the overall mine plan.

Currently the mine plan assumes a split of waste material between PAG and NAG by lithology. Further studies need to be completed to increase confidence in the grouping of waste categories to ensure waste is managed in a suitable manner, with a resulting potential reduction in water treatment costs.

26.2.6.3 Dewatering Requirements

A proper understanding of pumping requirements and the hydrogeology is critical. Further work assessing this is recommended.

26.2.6.4 Pit Slope Sensitivity

A detailed examination of the slopes to reduce stripping while still providing a safe work environment is required. Detailed mapping of the slopes and recommendations and further analysis is required.

26.2.6.5 Mining Schedule Optimization

A review of the mining schedule and design should be completed with updated metallurgical inputs resulting from ongoing and planned testwork. Hardness information should be incorporated into the schedule in the next stage so that mill throughput is better managed.

26.2.7 Hydrological

Hydrogeological testing (packer testing, profile tracer testing) and instrumentation (i.e. piezometers) should be installed in select holes to provide basic data for groundwater modelling and excavation dewatering/depressurization simulations. This program is budgeted at $75 k.

26.2.8 Water Treatment

After the site-wide water balanced has been further evaluated and ARD parameters are better understood, water treatment testwork should be conducted to confirm that impurities can be removed from water prior to discharge to the environment. This work is estimated at $50 k.

26.2.9 Infrastructure Geotechnical, Construction Borrow Materials, and Hydrological

Regional and local metrological data should be collected to support development of site climate data and hydrological parameters. Such data should be reviewed to ensure that they are statistically reliable for use by the Project, including effects of location and elevation. This should include:

- Examination of data from Seabridge Gold’s weather station for their KSM project;
- Data sets from long-term public regional weather stations.

A weather station should be installed at the Project to provide a correlation between the Eskay Creek and KSM project data sets.

Field mapping, geotechnical sample collection (boreholes, tests pits) and laboratory studies should be conducted to identify borrow material sources for construction activities, and provide information for support of the WRSF, plant site, ancillary facilities locations, and the TSF design.

This program is estimated at $1.3 M.

26.3 Phase 2

The second work phase will focus on project environmental, permitting, and social de-risking activities, which will include:

- Baseline and targeted environmental studies. As the majority of baseline information was collected prior to the 1994–2008 mining operation, Skeena's focus will be on re-establishing the baseline data for the project area. This would include studies such as habitat assessment, endangered or threatened species, and cross-checking sites selected to host infrastructure to ensure selected sites will have the minimal disturbance possible. Many of the studies are likely to have requirements for seasonal data collection;
- Environmental assessment;
- Documenting the required data to support applications for operating permits and completion of such applications;
- Consultations and negotiations with Indigenous groups;
- Other stakeholder engagement and consultation;
- Update water balance to better understand makeup requirements, distribution of site flows, site water quality and water treatment requirements.

A budget of approximately $4.6 M is recommended.

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