Eskay Creek Project

NI 43-101 Technical Report and Feasibility Study

British Columbia, Canada

Effective Date: September 6, 2022
Amended & Restated Report Date: September 19, 2022*

Prepared for: Skeena Resources Limited

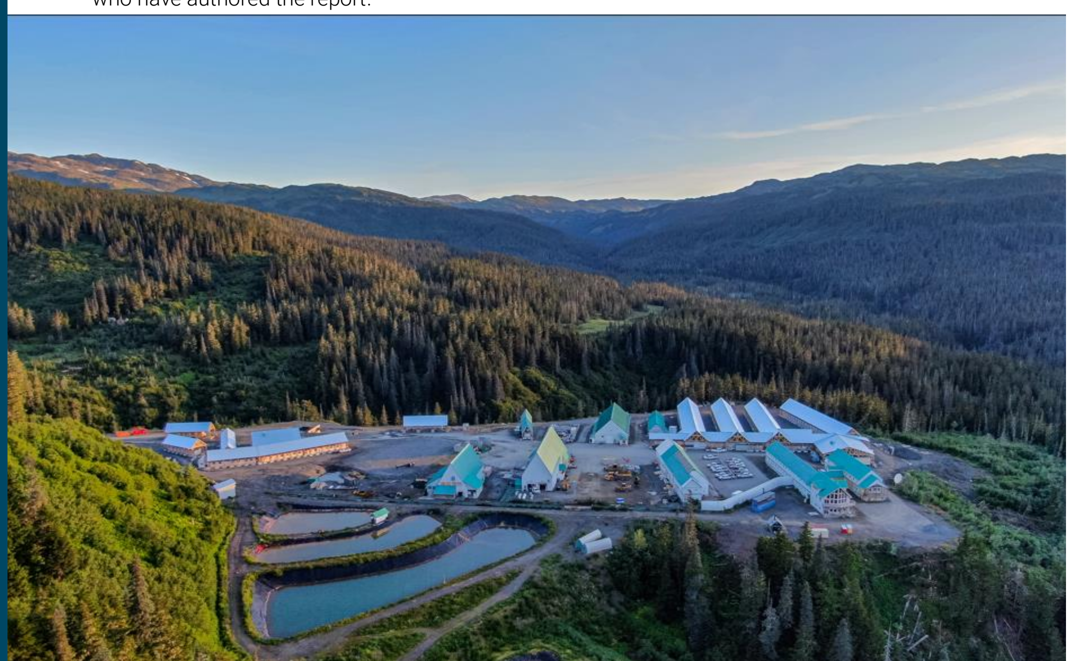
Suite #650 -1021 West Hastings Street
Vancouver, BC V6E 0C3 Canada

Prepared by: Ausenco Engineering Canada Inc.

1050 West Pender, Suite 1200
Vancouver, BC, Canada

List of Qualified Persons: Kevin Murray, P.Eng., Ausenco Engineering Canada Inc. • Mohammad Ali Hooshiar Fard, P.Eng., Ausenco Engineering Canada Inc. • Gerry Papini, P.Geo., Ausenco Sustainability Inc. • Davood Hasanloo, MASc, P.Eng., Ausenco Sustainability Inc. • Peter Mehrfert, P.Eng., Ausenco Engineering Canada Inc. • Sheila Ulansky, P.Geo., SRK Consulting (Canada) Inc. • Rolf Schmitt, P.Geo., ERM • Willie Hamilton, P.Eng., AGP Mining Consultants Inc. • Ian Stilwell P.Eng., BGC Engineering Inc. • Catherine Schmid, P.Eng., BGC Engineering Inc.

***Notice to Reader:**
This report replaces the "Eskay Creek Project NI 43-101 Technical Report and Feasibility Study," filed on September 14, 2022 and reflects certain housekeeping and clarifying amendments along with conforming changes to the certificates of the qualified professionals who have authored the report.



CERTIFICATE OF QUALIFIED PERSON

Kevin Murray, P. Eng.

I, Kevin Murray, P. Eng., certify that:

1. I am employed as a Manager Process Engineering with Ausenco Engineering Canada Inc., with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC Canada, V6E 3S7.

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "**Technical Report**"), that has an effective date of September 6, 2022 (the "**Effective Date**"), and an amended and restated report date of September 19, 2022.

3. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering. I am a member in good standing of Engineers and Geoscientists British Columbia, License# 32350 and Northwest Territories Association of Professional Engineers and Geoscientists' Registration# L4940.

4. I have practiced my profession for 22 years. I have been directly involved in all levels of engineering studies from preliminary economic analysis (PEA) to feasibility studies including being a Qualified Person for flotation projects including Ero Copper Corp.'s Boa Esparenca Feasibility Study and NorZinc Ltd.'s Prairie Creek PEA. I have been directly involved with test work and flowsheet development from preliminary testing through to detailed design and construction including my direct experience at Red Lake Gold Mine, Porcupine Gold Mine and Éléonore Gold mine while working for Goldcorp/Newmont.

5. 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.
6. I have not made a site visit to the Eskay Creek Project.
7. I am responsible for Sections 11.1-1.3, 1.16, 1.17.1-1.17.3, 1.17.8-1.17.10, 1.18, 1.20, 1.21, 1.22, 1.23.1, 1.23.6, 1.23.7, 2, 3, 17, 18.1-18.5, 18.14, 19, 21.1.1, 21.1.4-21.1.6, 21.1.7, 21.1.8-21.1.12, 21.2.1, 21.2.3, 21.2.4, 22, 24, 25.8, 25.9.1, 25.9.5, 25.11-25.13, 25.14.1.1, 25.14.1.5, 25.14.1.8, 25.14.2.5, 25.14.2.6, 25.14.2.8, 26.1, 26.5, 26.6, and 27 of the Technical Report.
8. I am independent of Skeena Resources Limited as independence is described by Section 1.5 of the NI 43-101.
9. I have had no previous involvement with the Eskay Creek Project.
10. 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.
11. 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: September 19, 2022

"*Signed and Sealed*"

Kevin Murray, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Mohammad Ali Hooshiar Fard, P.Eng.

I, Mohammad Ali Hooshiar Fard, P.E., certify that:

1. I am a Professional Engineer, currently employed as Geotechnical Engineer, with Ausenco Engineering Canada Inc., with an office at 1050 W Pender St, Vancouver, BC V6E 3S7.

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "**Technical Report**"), that has an effective date of September 6, 2022 (the "**Effective Date**"), and an amended and restated report date of September 19, 2022.

3. I graduated from Sharif University of Technology with BSc and MSc in Materials Science and Engineering in 2003 and 2006, respectively, and the University of Alberta in 2011 with a PhD in Materials Engineering. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia (No. 40965) and Engineers Yukon. I have practiced my profession for 18 years with experience in designing tailings and waste rock storage facilities as well as managing geotechnical field investigation and lab testing programs for mining projects across the globe. A summary of the more recent portion of my professional career is as follows:

   - Geotechnical Mining Engineer, Ausenco, Canada 2018–present
   - Geotechnical Mining Engineer, AECOM, Canada 2013–2017
   - Senior Geotechnical Consultant, SRK Consulting Inc., Canada 2011–2013

4. 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.

5. I have not made a site visit to the Eskay Creek Project.

6. I am responsible for Sections 1.17.4, 1.17.6, 16.4.2, 16.11.2, 16.11.3, 18.6-18.9, 18.11, 20.2.3, 21.1.3, 25.9.2, 25.10.2, and 25.14.1.6 of the Technical Report.

7. I am independent of Skeena Resources Limited as independence is described by Section 1.5 of the NI 43-101.

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

9. 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.

10. 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: September 19, 2022

"*Signed and Sealed*"

Mohammad Ali Hooshiar Fard, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Gerry Papini, P. Geo.

I, Gerry Papini, P. Geo., certify that:

1. I am a Professional Geoscientist, currently employed as Hydrogeologist, with Ausenco Engineering Canada Inc., with an office at 18th Floor, 4515 Central Boulevard | Burnaby, BC, V5H 0C6.

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "Technical Report"), that has an effective date of September 6, 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.

3. I graduated from the University of Cape Town with a Master of Science degree in Environmental Geochemistry in 1987.

4. I am a Professional Geoscientist, registered with Engineers and Geosciences of British Columbia, member number 141389.

5. I have practiced my profession continuously since 1988 and have been involved in: hydrogeological investigations and modelling for mining development projects in British Columbia since 2006.

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 30, 2022 for a visit duration of two days.

8. I am responsible for Sections 1.15.2, 1.17.5, 16.4.1, 16.4.3, 16.4.4, 18.12, 25.7.2, and 25.9.3 of the Technical Report.

9. I am independent of Skeena Resources Limited 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: September 19, 2022

"Signed and Sealed"

Gerry Papini, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Davood Hasanloo, P. Eng.

I, Davood Hasanloo, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Senior Water Process Engineer, with Ausenco Engineering Canada Inc., with an office at 1050 W Pender St, Vancouver, BC V6E 3S7.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "**Technical Report**"), that has an effective date of September 6, 2022 (the "**Effective Date**"), and an amended and restated report date of September 19, 2022.
3. I graduated from the Chamran University with a Bachelor of Science in Civil engineering in 2006 and University of British Columbia with a Master of Applied Science degree in Hydrotechnical Engineering.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 42250.
5. I have practiced my profession continuously since 2009 and have been involved in hydrotechnical analysis and water resources engineering related to mining projects dealing with sitewide water management and water management design.
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 Project.
8. I am responsible for Sections 1.17.7, 18.10, 18.13, and 25.9.4 of the Technical Report.
9. I am independent of Skeena Resources Limited 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: September 19, 2022

"Signed and Sealed"

Davood Hasanloo, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Peter Mehrfert, P. Eng.

I, Peter Mehrfert, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Principal Process Engineer, with Ausenco Engineering Canada Inc., with an office at 1050 W Pender St, Vancouver, BC V6E 3S7.
2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "Technical Report"), that has an effective date of 6 September 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.
3. I am a graduate of the University of British Columbia in 1996 where I obtained a Bachelor of Applied Science in Mining and Mineral Process Engineering.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 100283.
5. I have practiced my profession continuously for 27 years and have been involved in the design, evaluation and operation of mineral processing facilities during that time. Approximately half of my professional practice has been the supervision and management of metallurgical test work related to feasibility and prefeasibility studies of projects involving flotation technologies. Previous projects that I have worked on that have similar features to Eskay Creek (include gold bearing bulk sulphide flotation and fine regrinding) are: Springpole, Peñasquito and Spanish Mountain Gold.
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 1.12, 1.23.4, 13, 25.4, and 26.3 of the Technical Report.
9. I am independent of Skeena Resources Limited as independence is described by Section 1.5 of the NI 43-101.
10. I have been involved with the Eskay Creek project since June 2022, during which I supervised recent metallurgical test programs and analyzed results from past metallurgical programs and technical 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: September 19, 2022

"Signed and Sealed"

Peter Mehrfert, P. Eng.

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 (Canada) Inc., 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 Feasibility Study"*, (the "**Technical Report**"), that has an effective date of September 6, 2022 (the "**Effective Date**"), and an amended and restated report date of September 19, 2022.
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 1.5-1.11, 1.13, 1.23.2, 1.23.3, 4.1-4.3, 4.6, 4.10, 6-12, 14, 25.1, 25.2, 25.3, 25.5, 25.14.1.2, 25.14.1.3, 25.14.2.1-25.14.2.3, and 26.2 of the Technical Report.
9. I am independent of Skeena Resources Limited as independence is described by Section 1.5 of the NI 43-101.
10. In 2021 Skeena submitted a Prefeasibility Report for which I was the Qualified Person for the Eskay Creek Project Resource Model.
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: September 19, 2022

"*Signed and Sealed*"

Sheila Ulansky, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Rolf Schmitt, P. Geo.

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

1. I am a Professional Geoscientist, 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 Feasibility Study,"* (the "Technical Report"), that has an effective date of September 6, 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.
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 Geoscientist, registered with Engineers and Geoscientists 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 1.4, 1.19, 1.23.8, 1.23.9, 4.4, 4.5, 4.7-4.9, 5, 20.1-20.2.2, 20.3-20.6, 23, 25.10.1, 25.10.3-25.10.5, 25.14.1.7, 25.14.2.7, and 26.7 of the technical report.
9. I am independent of Skeena Resources Limited as independence is described by Section 1.5 of the NI 43-101.
10. I was previously responsible for the environmental, permitting and social content of the Eskay Creek Project Prefeasibility Study filed on September 1, 2021.
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: September 19, 2022

"*Signed and Sealed*"
Rolf Schmitt, M. Sc., 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., 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 Feasibility Study"*, (the "Technical Report"), that has an effective date of September 6, 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.
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 and Engineers and Geoscientists British Columbia, license number 20429.
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 1.14, 1.15.3, 1.23.5.2, 15.1, 15.3-15.8, 16.1, 16.2, 16.5-16.10, 16.11.1, 16.12-16.16, 21.1.2, 21.2.2, 25.6, 25.7.3, 25.14.1.4, 25.14.2.4, and 26.4.2 of the Technical Report.
9. I am independent of Skeena Resources Limited 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 the Preliminary Economic Assessment, Prefeasibility and Feasibility 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: September 19, 2022

"*Signed and Sealed*"

Willie Hamilton, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Ian Stilwell, P. Eng.

I, Ian Stilwell, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Principal Geotechnical Engineer, with BGC Engineering Inc., with an office at 234 St. Paul Street, Kamloops, BC, Canada, V2C 6G4
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 September 6, 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.
3. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in Geological Engineering in 1995.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 27316.
5. I have practiced my profession continuously since 1995 and specialize in geotechnical open pit and waste dump design and provide operational support for open pit mining operations. I have worked at mining operations and projects throughout Canada, the United States, Mexico, South America, Africa and Asia.
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 September 1, 2021, for a visit duration of three days.
8. I am responsible for Sections 1.15.1, 1.23.5.1, 15.2, 16.3.1-16.3.7, 16.3.9, 25.7.1, and 26.4.1 of the Technical Report.
9. I am independent of Skeena Resources Limited 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: September 19, 2022

"*Signed and Sealed*"

Ian Stilwell, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Catherine Schmid, M.Sc., P. Eng.

I, Catherine Schmid, M.Sc., P. Eng., certify that:

1. I am a Professional Engineer, currently employed as Senior Geotechnical Engineer, with BGC Engineering Inc., with an office at 234 St. Paul Street, Kamloops, BC, Canada, V2C 6G4

2. This certificate applies to the technical report titled, *"Eskay Creek Project N.I. 43-101 Technical Report and Feasibility Study"*, (the "Technical Report"), that has an effective date of September 6, 2022 (the "Effective Date"), and an amended and restated report date of September 19, 2022.

3. I graduated from Queen's University with a Bachelor of Applied Science degree in Geological Engineering in 2002 and a Master's of Science in Engineering in 2005.

4. I am a Professional Engineer, registered with Engineers and Geoscientists of British Columbia, member number 33195.

5. I have practiced my profession continuously since 2002 and specialize in underground rock mechanics and provide operational support for underground mine and tunneling operations. I have worked at mining operations and projects throughout Canada, the United States, Africa and Europe.

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 September 1, 2021, for a visit duration of three days.

8. I am responsible for Section 16.3.8 of the Technical Report.

9. I am independent of Skeena Resources Limited 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: September 19, 2022

"Signed and Sealed"

Catherine Schmid, M.Sc., P. Eng.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Skeena Resources Limited (Skeena) by Ausenco Engineering Canada Inc. (Ausenco), Ausenco Sustainability Inc. (collectively referred to herein as Ausenco), SRK Consulting (Canada) Inc., AGP Mining Consultants Inc. (AGP), BGC Engineering Inc. (BGC), and Environmental Resources Management (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 purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Overview

This Report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Skeena Resources to summarise the results of the NI 43-101 Technical Report and Feasibility Study on the Eskay Creek Project. The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1. The Report supports disclosures by Skeena in a news release dated September 8, 2022, entitled, "Skeena Completes Robust Feasibility Study for Eskay Creek After-Tax NPV (5%) of C$1.4 B, 50% IRR and 1 Year Payback."

The NI 43-101 responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Skeena Resources to manage and coordinate the work related to the N.I. 43-101 and for:
  - metallurgical testwork management;
  - developed the feasibility-level design and cost estimating of the process plant and surface infrastructure;
  - complete the feasibility-level design and bulk material estimates of the Tom MacKay storage facility (TMSF);
  - complete the feasibility-level design and bulk material estimates of the waste rock storage facility (WRSF);
  - water management (hydrogeological, hydrology, surface water management, site wide water balance;
  - site-wide geotechnical and hydrogeological investigations;
- SRK Consulting (Canada) Inc. (SRK) was commissioned to complete the mineral resource estimates.
- Environmental Resources Management (ERM) was commissioned to support environmental planning, assessment, licensing, and permitting, as well as the feasibility-level design and bulk material estimates of the water management structures.
- AGP Mining Consultants Inc. (AGP) was commissioned to design the open pit mine plan, mine production schedule, and mine capital and operating costs.

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

1.2 Reliance on Other Experts

The QP's have fully relied upon and disclaim responsibility for taxation information derived from MNP LLP who was retained for this information. Communications on August 31 2022, have been the basis for taxation. The QP's have also relied on Open Mineral AG for data on potential smelters, treatment charges, penalties, and net gold and silver payable information. This information was presented in a document in August 2022.

1.3 Property Description and Location

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.

1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

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 approximately 40 km west of the mine site and Bob Quinn, approximately 37 km northeast of the Project.

The mean annual total precipitation at the former project site is estimated to be 2,020 mm. Approximately 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 the actual operation will remain so.

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (masl), 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.

1.5 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 Limited. 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 5,798.86 ha, consisting of 49 mineral claims (3,968.58 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.), a 1% NSR payable to David A. Javorsky, a 2% NSR payable to Eagle Plains Resources and a 2% NSR payable to Joseph Vandervoort. There is also a 1% royalty payable to Barrick on all 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.

1.6 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 and prefeasibility technical studies.

1.7 Geology & 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 (Mount Madge Sedimentary unit), located between the footwall Eskay Rhyolite member and the hanging wall Willow Ridge mafic unit. The stratiform hosted zones include the 21A Zone (characterized by arsenic–antimony–mercury sulphides), the 21C Zone, 21B Zone, the 21Be Zone, the 21E Zone and the NEX 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. The Lower Mudstone (Datum Mudstone) and Even Lower Mudstone (Spatsizi Formation) are located stratigraphically below the footwall Eskay Rhyolite member and Dacite respectively. These mudstones are part of the Lower Package (LP) Zones.

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 21Be Zone, the 21E Zone, the NEX Zone, the WT 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, 21Be, 21C, NEX, WT 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 23 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.

1.8 Deposit Types

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.

Features that would classify Eskay Creek as a VMS deposit (Roth et al., 1999) include:

- It formed on the seafloor in an active volcanic environment with a rhyolite footwall and basalt hanging wall.
- There is a chlorite–sericite alteration in the footwall, and sulphide formation within a mudstone unit at the seafloor interface.
- Unlike many VMS deposits, Eskay Creek has 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 sulfosalts, are more typical of an epithermal environment with low formation temperatures.

Features that would classify Eskay Creek as a subaqueous hot spring gold–silver deposit (Alldrick, 1995) include:

- broad hydrothermal systems marked by widespread sericite–pyrite alteration;
- evidence of a volcanic crater or caldera setting; and
- accumulations of felsic volcanic strata.

1.9 Exploration

The exploration programs from 2018 to 2021 completed by Skeena are overviewed in this technical report. A summary is as follows:

1.9.1 2018 – 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 km2 survey area.

1.9.2 2019 – Mapping and Grab Sampling Program

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.

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. 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.

1.9.3 2020 – 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 over the axis of the Eskay Creek anticline from the Bowser Basin south to the Tom MacKay Zones using the DIAS32 system in the UTM Zone 9N WGS84.

Dias Airborne Limited of Saskatoon, SK, flew an airborne magnetic gradiometry survey over 5 days in 2020 using the QMAG full tensor magnetic gradiometer (FTMG) system. Forty-meter line spacing for a total of approximately 1060 line kilometres were completed, which included 965 km of survey lines and 95 km of tie lines.

1.9.4 2021- Eskay Rift-Basin Reconstruction and Targeting Project

From April 19 through May 3, 2021, relogging of diamond drill core was undertaken to establish an informal stratigraphy for strata that host the Eskay deposits. Relogging of drill core and resulting graphic logs were completed for 26 representative drill holes totalling approximately 7,439 m. Eighty-nine samples were collected for whole rock analysis to characterize lithofacies and alteration types.

1.9.5 2021 – Geochemical Soil Sampling Program

Inherited soils data collected by previous operators demonstrated strong correlations between Au-Ag mineralization exposed at surface and B-Horizon Au soil anomalies. Unfortunately, the historical soils coverage was discontinuous across the property, particularly along the Eastern Limb of the Eskay Anticline. In addition, the data collected by previous operators is poorly documented, generally lacks any quality assurance/quality control checks and is therefore of uncertain quality.

During the summer of 2021, Skeena collected 4,367 soil samples. The soil sampling program covered the majority of the lease boundaries, apart from areas defined as Bowser Basin geological units. The sampling entailed 116 line kilometres and was completed on a systemic 25-m x 100-m grid. Given the surficial footprint criteria for a near surface bulk tonnage target, these soil grid parameters permitted adequate coverage to detect an economic target.

1.9.6 2021 – Regional Mapping and Grab Sampling

From June through August 2021, Skeena collected 2,296 rock samples throughout the property, apart from areas defined as the Bowser Basin geological unit, to assist in the characterization of the lithofacies and alteration types. In addition, geological field mapping and prospecting activities were completed over the entirety of the property with additional focus on geochemical anomalies reported in historical soil grids, grab rock samples and diamond drilling. The samples were collected to ensure coverage at outcrops that had no previous data recorded nearby. The most mineralized or altered parts of the outcrops were sampled.

1.9.7 Exploration Potential

There is remaining exploration potential in the Eskay Creek deposit. Several areas have been selected for drill targeting based on the geochemical soil sampling, and grab rock sampling campaigns along the Eskay Trend.

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, 21A Zone, 23 Zone, 21C Zone and in the mudstones of 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 style mineralization.

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

1.10 Drilling

Surface drilling has been carried out by multiple operators, with the first drilling on the property by Unuk Gold in 1934. Data collected prior to Skeena's project interest is discussed in Section 6.

Since 2018 to the end of 2021 Skeena has drilled 913 surface drill holes totalling 128,362.89 m. Table 1-1 summarizes the surface drilling Skeena has completed on the Eskay Creek Project from 2018 to 2021.

Table 1-1: Drill Summary Table of Drilling Undertaken by Skeena

Period of Work	Area of Work	Number of Holes	DDH #'s	Metres Drilled
2018	21A / 21C / 22 Zones	46	SK-18-001 to SK-18-043; SK-18-048 to SK-18-051	7,737.45
2019	21A / 21B / 21E / HW Zones	203	SK-19-044 to SK-19-047; ~SK-29-052 to SK-19-247	14,091.87
2020	21A / 21B / 21C / 21E / HW / PMP / WT / MAC / 22 Zones	473	~ SK-20-248 to SK-20-788	79,992.79
2021	22 / 21A / 21C / 21B / 21E / PMP / HW / NEX / Albino Lake / Tom MacKay / 23 Zone / East Dacite / Eskay Porphyry	191	~ SK-21-789 to SK-21-997	26,610.78

1.11 Drill Hole Data Verification

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.

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. 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 ranges after checking for data entry errors, then repeat assay were requested. The laboratory was instructed to retrieve five pulp samples before and after the QC failure. Prep and pulp duplicate data were also monitored, with Skeena reporting any concerns to the laboratory manager.

1.12 Mineral Processing & Metallurgical Testing

1.12.1 Previous Programs

As part of the 2019 PEA and 2021 PFS, testwork programs were completed by Blue Coast Research in Parksville BC and Base Metallurgical Laboratories Ltd. in Kamloops BC respectively. The outcome of this work was a modified circuit design, incorporating two stages of milling and flotation – or an MF2 flowsheet. This avoided overgrinding softer minerals present at different levels in the Eskay Creek samples as well as isolating a slimes fraction to a separate flotation circuit.

The 2019 program was completed on a limited number of samples from 21A, 21C and 22 ore zones while the 2021 program included a wider range of samples for variability testing and from a greater number of ore zones.

Testwork into cyanide leaching, gravity recovery and concentrate hydrometallurgical retreatment resulted in these options being excluded from the final flowsheet, which generates a saleable precious metal concentrate from both coarse and fine flotation circuits.

Work was also completed to estimate regrind mill power requirements and dewatering of tailings and final concentrate.

1.12.2 Feasibility Study Program

The FS program was completed by Base Metallurgical Laboratories Ltd. over the period June 2021 to August 2022, focussing on FS flowsheet conditions. A bulk sample was processed through a pilot plant to generate sufficient sample mass for regrind mill evaluation and additional thickener and filter testing. A larger variability sample program was tested to generate results for recovery modelling. Two main lithologies: Rhyolite and Hanging Wall/Mudstone were modelled separately due to their different response.

Additional comminution testing was conducted on both Rhyolite and Mudstone samples as well as regrind mill specific energy testing (both HIGmill and IsaMill) was done on samples of rougher concentrate and deslimed rougher tailings. Dewatering tests on the final concentrate identified the need to supplement drying after pressure filtration for some of the samples, in order to reach Transportable Moisture Limit (TML) levels of water content.

The variability testing provided insight into methods to mitigate cleaner circuit losses, particularly on Hanging Wall/Mudstone samples. Repeat cleaner tests were conducted on several samples from the variability testing to demonstrate improved metallurgical performance when grind size targets and collector addition rates were tightly controlled. After this improved repeat testing, locked cycle tests were conducted on several samples including a year 1-5 composite to confirm closed circuit performance for recovery modelling and equipment sizing.

For mine planning purposes, a series of recovery models were developed from the 2022 FS variability results, for each major rock type. The recovery equations developed are acceptable for use in the MRMR estimates and mine plan used in financial modelling. Within each rock type, concentrate quality could be reliably estimated from feed grades and was found to vary based on gold and sulphide mineral contents, as well as lithology. The recovery models developed were based on performance at different cleaner circuit operating points for each mining period in order to maximize NSR.

With higher-grade material processed in the first three years, although arsenic, antimony, and mercury levels are expected to be elevated in the final concentrate, the concentrate saleability is not impacted. Grades of gold in concentrate are expected to be 60 g/t in Year 1 and decrease to 18 g/t in Years 8 and 9. Overall gold recovery for the first nine years is 84% to a 37g/t Au concentrate. Silver recoveries average 88% over the mine life, with concentrate grades of 1,024 g/t Ag. Sulphur levels in final concentrates are expected to be between 18% and 26% at selected cleaner operating points.

1.13 Mineral Resource Estimates

The Mineral Resource estimate is primarily based upon legacy drilling completed by the previous operator; however, additional holes drilled by Skeena since 2018 have been included. The database used in estimation contains 7,583 historical holes and 826 completed surface holes drilled by Skeena from 2018 to August 2021. The close out for the database was September 10, 2021, once all assays were received for the last hole from Phase 3.

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, 91 solids were created for the 2022 estimate including 90 mineralization 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:

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.

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.

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 10 x 10 x 5 m parent block sizes, with sub-block sizes of 5 x 5 x 2.5 m; and 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 2.5 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 4.5–600 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 2022 estimate is a combination of lithology type and Zone, with the mean SG value selected from each ore Zone, or, if outside of the ore Zones, then average SG values within lithology type.

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. Five estimation domains below the bottom of the optimized resource pit were reported as resources potentially amenable to underground mining methods (22, HW, NEX, WT 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, except for the small faults of the 21C Zone, the Even Lower Mudstone and Footwall Andesite where Inverse Distance to the second power was used (ID2). Gold and silver grades within the mineralization domains were estimated in three successive passes with increasing search radii based on variogram ranges. A fourth validation pass was used for validation purposes only. A hard boundary was applied within a 1 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 ID2 and nearest-neighbour (NN) methods, and swath plots. No major biases were noted. A 0.2 m geotechnical solid around the underground workings was used as the depletion zone for reporting remaining resources

OK was used to estimate gold and silver within the underground model except for the Even Lower Mudstone and Footwall Andesite. 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 in the open pit model, 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, using a search distance of the variogram, with a minimum of three 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 Pass 3, using search distances of 2.5 times the variogram range, and a KV of <0.8 were classified in the Inferred category. For the LP domain, an average distance of 100 m was used as an additional constraint for the Inferred resources. 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.

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 rock types, 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.

1.13.1 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. Mineral Resources are reported inclusive of Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-2: Open Pit Constrained Mineral Resource Statement Reported at 0.7 g/t AuEq Cut-Off Grade by Domain

Classification	Tonnes 000	Grade		Contained Ounces			
		AuEq g/t	Au g/t	Ag g/t	AuEq Oz(000)	Au Oz (000)	Ag Oz (000)
Measured	21,784	4.8	3.5	92.4	3,355	2,481	64,679
Indicated	24,724	2.3	1.8	37.6	1,804	1,400	29,896
Total M + I	**46,508**	**3.5**	**2.6**	**63.2**	**5,159**	**3,881**	**94,575**
Inferred	3,420	1.5	1.3	20.2	170	140	2,222

Table 1-3: 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

Classification	Tonnes 000	Grade			Contained Ounces		
		AuEq g/t	Au g/t	Ag g/t	AuEq Oz (000)	Au Oz (000)	Ag Oz (000)
Measured	737	6.1	4.6	112.7	145	109	2,671
Indicated	550	5.1	4.4	62.6	91	77	1,107
Total M + I	**1,287**	**5.7**	**4.5**	**91.3**	**236**	**186**	**3,778**
Inferred	330	4.1	3.5	42.6	43	37	452

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 January 18, 2022.
- 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 resources are exclusive of the 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 and Zone 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 within the stope optimized shapes using a cut-off of 2.4 g/t AuEq for the long-hole mining scenario and 2.8 g/t AuEq for drift-and-fill mining scenario.
- Cut-off grades are based on a price of US$1,700 per ounce of gold, US$23 per ounce silver, 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 45 degrees
  - A reference mining cost of US$3.00 per tonne mined
  - A processing cost of 15.50 US per tonne processed
  - General and administrative costs of US$6.00 per tonne processed
  - Mining dilution of 5%
  - Mining recovery of 95%
  - Transportation and refining costs of US$25 per ounce AuEq
- Underground key assumptions for reasonable prospects for eventual economic extraction are as follows:
  - A reference mining cost of US$80 per tonne mined
  - A processing cost of US$25 per tonne milled
  - General and administrative costs of US$12 per tonne milled
  - All in costs of US$117 per tonne milled
  - Transportation and refining costs of US$25 per ounce 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.

1.14 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 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 for kinematic sectors, which were based on geotechnical assessment of available geotechnical and hydrogeological data from drilling, logging, mapping, sampling, and laboratory testing;
- NSR calculations for a gold concentrate assuming a 2% royalty and revenue from gold and silver metal. Prices of US$1550/oz gold and US$20/oz silver were used in NSR calculations.
- Pit shells generated using the Lerchs–Grossmann (L–G) algorithm in MinePlan software. Ultimate pit shells were generated using a revenue factor of 0.9 or metal price of $1,395/oz. These were used as the basis for the design;
- Pit designs were developed for the north and south pit areas. The initial north pit phases (Technical Sample, Quarry 1 and Quarry 2) were designed for the purpose of obtaining a technical sample and necessary NAG waste material to create supporting infrastructure. The north pit will consist of an additional three main phases, while the south pit will only contain a single small phase.

An NSR value per tonne of C$24.45/t was used to flag potential mill feed and waste blocks prior to dilution and represents the preliminary process and site G&A costs. This NSR value was also used to determine mill feed in the statement of open pit reserves;

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 0.19 g/t Au and 3.71 g/t Ag.

1.14.1 Mineral Reserve 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 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.

The total reserves for the Eskay Creek Project are shown in metric units in Table 1-4. Some variation may exist due to rounding.

Table 1-4: 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	17.3	3.64	99	4.92	2.02	55.1	2.73
Probable	12.6	2.10	50	2.75	0.85	20.5	1.12
Total	**29.9**	**2.99**	**79**	**4.00**	**2.87**	**75.5**	**3.85**

* Note: This mineral reserve estimate has an effective date of June 30, 2022 and is based on the mineral resource estimate dated January 18, 2022 for Skeena Resources by SRK Consulting (which has been updated since the PFS). 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$1,550/oz gold price and US$20.00/oz silver price. An NSR cut-off of C$24.45/t was used to define reserves based on preliminary processing costs of $18.22/t ore and G&A costs of C$6.23/t ore. The metallurgical recoveries varied according to gold head grade and concentrate grades. Gold and silver recoveries were approximately 83% overall during the LOM scheduling. Final operating costs within the pit design were C$3.72/t mined, with associated process costs of C$16.91/t ore and G&A costs of C$4.20/t ore.

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

1.15 Mining Methods

1.15.1 Geotechnical Considerations

The Project targets a deposit that will be mined via a 260 m deep North pit and 80 m deep South pit. A diversion tunnel is proposed to divert flows from the Tom MacKay Creek around the north pit boundary. BGC Engineering Inc. (BGC) undertook this work at the request of AGP Mining Consultants Inc. (AGP) to support this study of the Eskay Creek project.

Following completion of the 2021 drilling program, BGC conducted a compilation, review, and assessment of available geotechnical data and information to determine suitable pit slope design criteria by kinematic sector angles for FS-level mine planning tasks. BGC developed a geotechnical model that characterizes the rock mass conditions, structural geology, hydrogeology, and seismicity of the open pit and diversion tunnel areas. This model was used as a basis for the open pit and diversion tunnel geotechnical assessments.

Twenty-meter-high double benches are likely achievable in all sectors, with recommended catch bench widths ranging from 12. m to 37.5 m. The slope design criteria assume that controlled blasting will be implemented. Scaling bench faces and cleaning accumulated material from bench toes is recommended.

Based on the results of the bench scale and inter-ramp kinematic analyses, BGC prepared provisional recommended slope design criteria, which were then incorporated into the FS mine plan by AGP. BGC then carried out limit equilibrium inter-ramp and overall slope stability analyses on representative cross sections through the FS-level pit plan. Stability analyses indicate that the slopes of the FS pit meet the design acceptance criteria with horizontal depressurization 40 m behind the pit face in the east walls of the North pit, and 20 m behind the pit face in the north and south walls of the North pit. No depressurization was required in the South pit.

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.

1.15.2 Hydrogeological Considerations

Historic and recent groundwater investigations illustrate elevated hydraulic conductivity associated with the N-S trending faults in the proposed mining area. However, not all the fault systems are conductive; for example, the E-W trending Riedel shears are considered to have similar conductivity to the country rock or lower conductivity, potentially acting as barriers (aquitards) to flow. The former underground mine operators reported rapid response to precipitation events with increased mine inflows potentially resulting from the conductive faults, but potentially also from increased fracturing from mining activities, and inflows through unsealed exploration boreholes. Higher groundwater recharge in the former underground mine area is therefore expected compared to in undisturbed areas.

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 moderately conductive (calibrated K = 5E-07 m/s) compared to the footwall (rhyolite) rock (calibrated K = 5E-08 m/s) and will likely dewater more easily than the rhyolite, which reportedly has high fines content and drains poorly. 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. The planned ultimate pit bottom will be at 714 masl, and therefore only about 50 m of flooded working will require dewatering. However, dewatering the underground workings in advance of mining may promote overall pit wall depressurization.

The hydrological cycle implies a short period of groundwater recharge associated with spring melt and fall rain; a bimodal hydrograph with peaks in May / June and then in October / November. The average annual variation in groundwater levels is 3.5 m (range 0.5 m – 10 m). Groundwater levels in the pit area are generally deep: 30 m - 60 m and thought to be due to the active pumping that maintains the water level in the underground workings around 765 masl. Groundwater flux in the mining area is predominantly to the east, toward Ketchum Creek with only 10% of flow to Tom McKay Creek. On the western margin of the proposed waste rock storage area, groundwater depths are shallow (2-4 m) and the groundwater flow direction predominantly toward Tom McKay Creek. Groundwater depths north of Tom McKay Lake range from 4-9 m. There is hydraulic containment throughout most of the extent of the proposed tailings storage area, except in the south where modelling shows a westerly flow path to Harrymel Creek. The extent to which this flow path is cut-off by north-south fault is unknown and the subject of further investigation. Mine designs incorporate removal of conductive overburden materials (e.g., beneath the proposed TSF dams) and capture of shallow seepage from mine waste facilities in seepage collection ponds (e.g., in the waste rock storage area). Monitoring wells are being installed in groundwater flow paths between mining infrastructure and creeks to measure the potential effects to water quality.

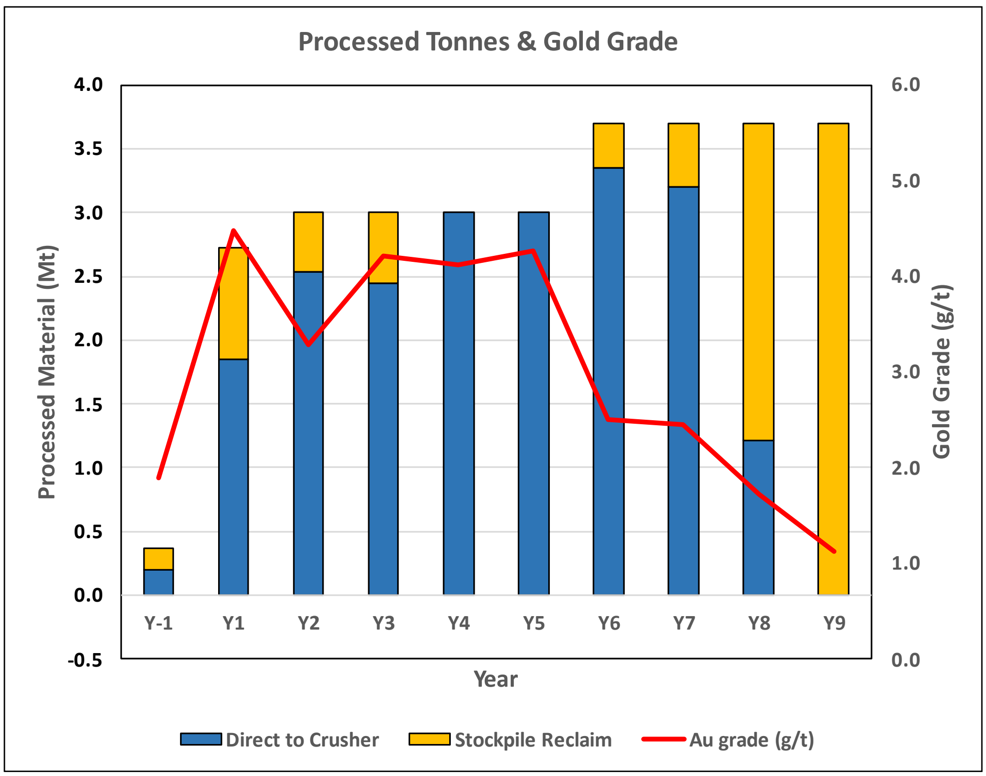
1.15.3 Mine Plan

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 FS. 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. Waste mining will occur on 10-m benches with catch benches spaced 20 m vertically. Berm widths will vary depending on the kinematic pit sector, orientation, and lithology type. The haul roads will be 30.2 m in width with a road grade of 10%.

The mine schedule plans to deliver 29.9 Mt of mill feed grading 2.99 g/t gold and 79 g/t silver over a mine life of eight years. Processing of low-grade ore from stockpiles will continue until year 9. Waste tonnage from the pits totalling 223 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is 7.5:1. The mine schedule assumes 3.0 Mt/a of feed will be sent to the process facility in years 1 to 5 using a suitable ramp-up in Year 1. The mill will operate at 3.7 Mt/a for Years 6 to 9. 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.

Figure 1-1: Planned Life of Mine Mill Feed Tonnes and Ounces



Note: Figure prepared by AGP, 2022.

The current mine life includes three years of pre-stripping and eight years of mining. Mill feed will be stockpiled during the pre-production years, with four stockpiles envisaged. A technical sample and two small quarries will be mined during pre-production so that process performance of the mill can be evaluated on a bulk sample.

A total stockpile capacity of approximately 6.0 Mt was reached in this schedule. 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.

Preproduction mining will be completed with small equipment up to 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 small loaders loading in tandem. 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.

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. The grade control holes will serve two purposes:

Definition of the mill feed grade and contacts; and location of previous underground infrastructure prior to blasthole rigs drilling.

Various rock types are present in the material mined within the final pits. The key difference since the PFS study was revised segregation of PAG and NAG waste rock. Based on recent test work, the only lithologies considered as NAG were hangingwall andesite and upper members of the 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 142 Mt and 82 Mt, respectively. The total amount of waste within the pits in mine plan is 223 Mt. This split in material will be determined by blast hole sampling and from the RC grade control drilling.

The Waste Rock Storage Facilities were designed in accordance with BC's "Interim Guidelines Mined Rock and Overburden Piles Investigation and Design Manual" (1991). 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 WRSFs 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.

1.16 Recovery Methods

The testwork provided was thoroughly analysed and several options of process routes were addressed in the initial stages of the feasibility study. Based on the analysis, the 2 stage milling and flotation (MF2) process route was maintained as the best suited for the testwork results and subsequent economic analysis for the material. The unit operations selected are typical for this industry.

The Project will be constructed in two distinct phases, as follows:

- Initial operation of 3.0 Mt/a for Years 1 to 5, which comprises:
    - 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 semi-autogenous grinding (SAG) mill, pebble crusher (installed for Year 4 operations), and ball mill in closed circuit with hydrocyclones;
- Rougher flotation with concentrate regrind and two stages of cleaning;
- Rougher tails slimes classification via two stages of hydrocyclones;
- Secondary grinding including ball mill and IsaMill and scavenger flotation, fed from the slimes circuit underflow;
- Fines flotation and two stages of cleaning, fed from the slimes circuit overflow;
- Concentrate thickening, filtration, drying and storage;

- Concentrate load-out by way of front-end loader filling concentrate transportation;

Final tailings pumping to the TMSF.

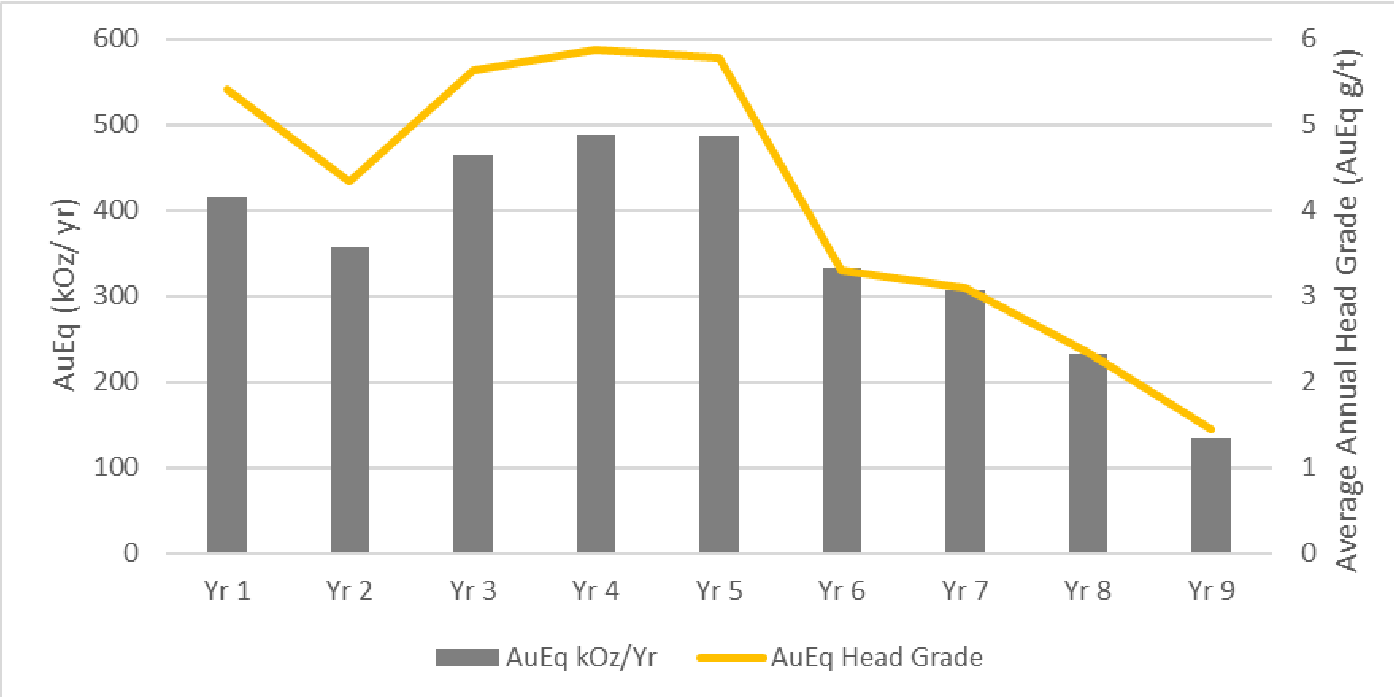
- Expansion to 3.7 Mt/a for the remaining mine life, which includes the initial equipment with the addition of the following installed for year 6 operation:
  - Secondary crushing circuit (cone)
  - A second ball and extra cyclones
  - Additional IsaMill

Key process design criteria are listed below:

- Initial operation nominal throughput of 8,220 t/d or 3.0 Mt/a
- Expansion nominal throughput of 10,140 t/d or 3.7 Mt/a
- average head grade of 2.99 g/t Au and 79 g/t Ag
- crushing plant availability of 70%
- operate two shifts per day, 365 d/a with plant availability of 92% for grinding, flotation, and filtration

Product will be gold concentrate to be sold to refineries. Annual Gold Equivalent production is shown in Figure 1-2.

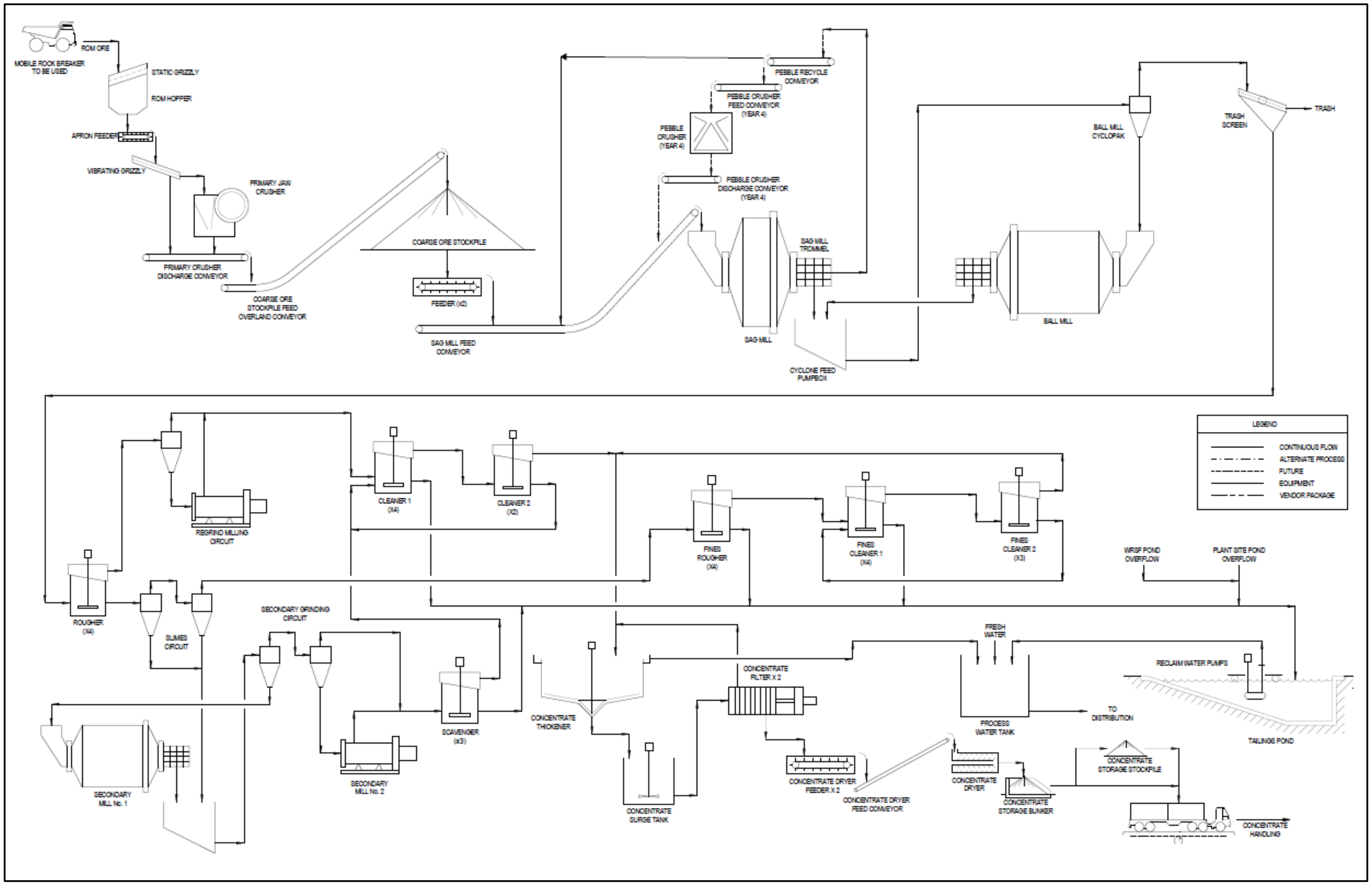
Figure 1-2: Eskay Creek Annual AuEq production and head grade



Note: Figure prepared by Ausenco, 2022.

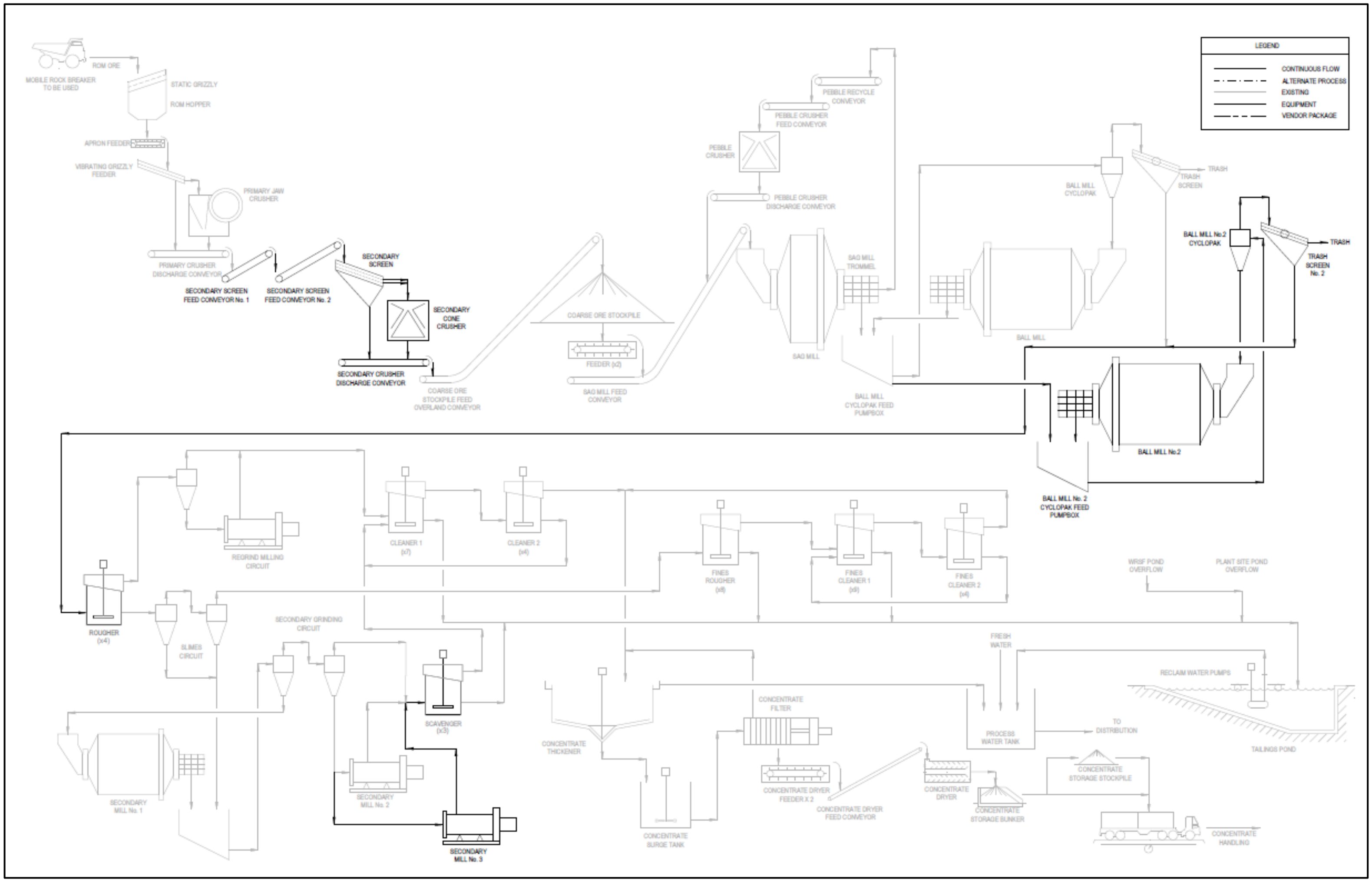
An overall process flow diagram showing the unit operations in the selected process flowsheet for the initial operation is presented in Figure 1-3 and for the expansion in Figure 1-4.

Figure 1-3: Simplified Process Flowsheet (1-5 Years)



Note: Figure prepared by Ausenco, 2022.

Figure 1-4: Simplified Process Flowsheet (Years 6+)



Note: Figure prepared by Ausenco, 2022.

1.17 Project Infrastructure

1.17.1 Overview

The overall site plan (see Figure 1-5) shows the major project facilities including the open pit mines, Tom MacKay storage facility (TMSF), waste rock storage facility (WRSF), water management ponds, process plant, mine services, historical site and main access road. Access to the facility is from the northern side of the property from the existing Eskay Creek Mine Road. Access to the process plant will be via the existing road to the historical Eskay Creek Site.

1.17.2 Access

Access to the Eskay Creek Project is via the existing 59 km all-season gravel road that connects to Highway 37. The access road is currently in good condition, where upgrades to two of the 8 bridges on this road will be required to accommodate equipment deliveries during construction and concentrate transportation during operation.

1.17.3 Power

The power supply for the Project will be provided from the 287 kV Volcano Creek interconnection point, where a new 287/69 kV substation will be installed and a 17 km, 69 kV overhead power line will be run to the mine.

The Eskay Creek Project has the following electrical load requirements:

Initial operation: Initial start-up requirement between Year 1 to 5 inclusive – 27.1 MW

Expansion: Full load requirement in year 6 to end of life – 31.2 MW

1.17.4 Tom MacKay Storage Facility (TMSF)

The existing TMSF was selected as the preferred tailings storage option since it is permitted as a tailings storage facility (TSF) and both tailings and PAG Waste Rock can be storage subaqueously to prevent these materials from generating acid. The TMSF has sufficient capacity to contain 109.4 Mt of tailings and PAG waste rock and will be constructed in three 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 2013) and Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (2016), 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 is "very high" based on HSRC Guidance Document, Section 3.4 (BC Ministry of Energy and Mine 2016) and CDA (2013) Dam Safety Guidelines.

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). The TMSF has sufficient capacity to store both tailings and PAG waste rock with four embankments. The dams will be constructed in 3 phases; Phase 1 (Year -1), Phase 2 (Year 1 and 2) and Phase 3 (Year 4 and 5). Northern three starter dams (Phase 1) will be constructed to an elevation of 1,092 masl. This includes a 1 m diameter penstock through the northeast dam (Dam 1) along the existing alignment of Tom MacKay creek. The phase 2 raise will be the expansion of the north dams to an elevation of 1,107 masl and a new embankment at the south end of the facility to prevent flow into Coulter Creek drainage. The final embankment raises (Phase 3) will be constructed to an elevation of 1,122 masl. In addition, the closure spillway will be installed to maintain 5 m of water cover over the PAG waste rock and tailings post closure in Year 7. TMSF along with the spillway designed to pass the Probable Maximum Flood (PMF). The northern embankments have a geomembrane liner system anchored to bedrock which will produce very little seepage due to the composite liner system. A base flow will discharge through the penstock into Tom MacKay Creek year-round. The southern embankment has a clay core and there will be minor seepage losses to the south through the clay core compared to the surface runoff on the south side of the embankment. The south side of the dam water will impound to an elevation of 1,107.70 masl before spilling into Coulter Creek watershed. Most of the flow into the Coulter Creek drainage will come from surface runoff and snow melt. In addition, floating turbidity fences will be placed around the active disposal areas to further aid in minimizing the migration of fine-grained suspends solids. In winter, a large enough area will be cleared of ice around disposal areas to allow the installation of the fences.

PAG waste rock will be deposited at the north end of the facility. PAG waste rock deposition will use a berm approach, depositing PAG waste across the facility from west to east. The berms 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, excavators, and a 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 to the north due to excavation operations. The final height of the berm will be 3 m below the water surface during operations and all materials will be 5 m below the water surface post closure.

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 and during winter holes will be drilled through the ice and the tailings line will be placed through the hole to the bottom of the TMSF. Due to the fine ore grind (P80 = 35 µm), the end of the pipeline will be positioned close to the bottom of facility (deposited tailings) along with a manifold with multiple port to reduce the velocity of the tailings slurry exiting the pipe along with an inline flocculant dosing station near the waters edge to maximize settling and minimize entrainment of fine particles to the surface of the TMSF. In addition, a floating turbidity fence will be placed around the barge to minimize migration of fine grain suspended solids and in winter a large enough area around the pipeline will be cleared of ice to install the fence. The minimum water depth over the tailings would be 3 m during operations and 5 m at closure to prevent both wind and ice remobilization of the tailings and prevent any PAG tailings from generating acid.

1.17.5 Water Supply

Fresh water makeup for the plant and potable supplies will be sourced from aquifers. Water pumped from the mine will meet the bulk of processing needs with any process water deficiency being recycled from Tom MacKay Storage Facility. Test boreholes have indicated good groundwater potential in bedrock associated with geological structures, and these should be targeted for establishing wellfields for the Project.

1.17.6 Water Management

The objective of surface water management is to protect groundwater and surface water resources. Feasibility Study infrastructure and upstream catchments for the Project were delineated based on topography data and footprints of facilities.

Contact and non-contact water are managed separately for the Project. Contact water is captured and transported in collection ditches and pipelines to sediment ponds, sumps, and contact water ponds. For roads, runoff will be captured in collection ditches and conveyed to sediment ponds, to remove greater than 10 microns particles prior to discharging into the environment. Contact water from the open pits, waste rock storage facility (WRSF), Ore Stockpile, Process Plant Pad will be capture in collection ditches and conveyed to pit sumps, Ponds 5 and 6. All runoff collected in these sumps and Pond 6 will be pumped to Pond 5. Then all water from Pond 5 will be pumped to the process plant and used in mining operations or pumped with the tailings to TMSF.

Currently, there are no diversion channels, collection ditches, or water treatment facility for the subaqueous deposition of the PAG waste rock and tailings in TMSF. Non-contact water is diverted around other mine infrastructure, where possible, through diversion channels, culverts, and creek crossings. A detailed description of surface water management structures is presented in Section 20.2.3.

The various contact flows for average and wet climatic conditions are provided in Section 18.6. Non-contact water will be conveyed around mine facilities in diversion channels where possible.

1.17.7 Snow Management

A snow management plan will be required to manage snow accumulation during the Project Operations since the Project site is in a snow-dominant region. The mine site is at an average elevation of 1,100 metres. The area experiences heavy rain and snow, with an average precipitation of 2,020 mm per year. The practices and proposed structures outlined in this plan have been developed to manage snow from Pit, Plant site, Waste Rock Storage Facilities (WRSF) and Haul Roads.

1.17.8 Accommodation

During construction period, a temporary 210-person rental camp for construction will be established and utilized together with the existing 227 beds at the historical camp. This rental camp will continue to operate during the first three years of operation, while a new 180 bed permanent operations camp will be constructed near the process plant area. This operation camp, together with 200 person modules that will be progressively relocated to this area from the historical camp will comprise the ultimate operational camp for the remaining life of mine, complete with all the required common facilities.

1.17.9 Buildings

The following enclosed areas and buildings are considered in the design, in order to support the facilities and operations of the project:

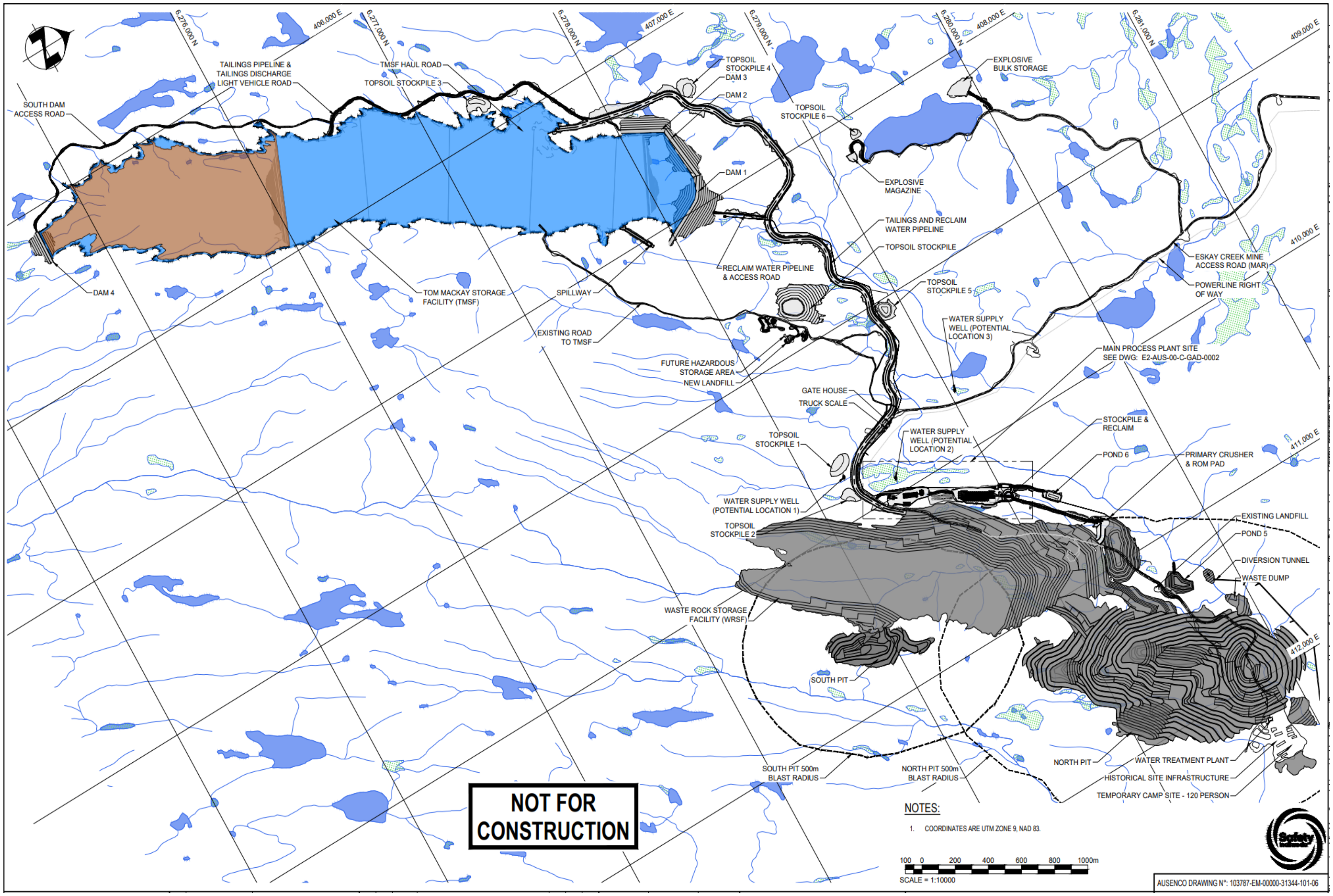
- Process Plant Building: This will be a 210m (long) x 36m (wide) pre-engineered building fully enclosed with metal cladding complete with HVAC.
- Crushing Plant Building: The building (29 m long by 9m wide) will be located over the primary crusher, control room and rock breaker equipment, adjacent to the ROM pad.
- Truck Workshop and Offices: This will be a 23 m (long) by 85 m (wide) pre-engineered building supported on a concrete foundation. The ground floor will be used for vehicle maintenance and washdown, with upper levels of the building dedicated to the changerooms and offices.
- Fuel storage station: The fuel station will consist of a 50 m (long) x 70 m (wide) open-air area including truck manoeuvring space. The area will be covered by a roof to protect against snow build-up.
- Plant Maintenance Shops & Warehouse: The plant maintenance shops and warehouse will located at the western end of the process plant building with a separated wall and will be 18 m wide by 36 m long.
- Main Administration Building & Process Plant Offices: 18 m (wide) x 18 m (long), double-storey building located adjacent to the process plant.

- Assay and Geochemical Laboratory: The assay and geochemical laboratory will be a 19.5 m (long) by 12.5 m (wide) building and will house equipment for guiding ongoing mining and process plant operations.
- Temporary Camp: a 210-bed camp, complete with required facilities (kitchen, gym, lunchrooms, etc,) which will be constructed near the Forest Kerr area.
- Permanent Operations Camp: a 380-bed facility, which is intended to utilize the 200-bed existing facility, relocated near the process plant area, and complemented by a new 180-bed camp, complete with all the required services and facilities.
- Other Facilities: which includes gate house, truck scale, onsite landfill facility, propane storage area, tire repair shop.

1.17.10 Concentrate Transportation

Concentrate will be loaded using front-end loaders into highway haul trucks (72,300 kg GVW) up to 49 t concentrate per truck (24.5 t per tandem dump trailer). Concentrate will be trucked using the main site access road and Highway 37 under a "bulk haul" permit from the Province of BC Ministry of Highways to move concentrate from the mine approximately 250 km to Stewart Bulk Terminals (SBT). SBT is a multi-commodity port facility with up to 16,000t storage for Skeena's gold concentrate in a dedicated storage building with existing conveying load out infrastructure. Concentrate will be loaded onto bulk carrier ships at SBT via its existing ship loading infrastructure.

Figure 1-5: Overall Site Plan



Note: Figure prepared by Ausenco, 2022.

1.18 Market Studies and Contracts

The Eskay Creek operation will produce a precious metal concentrate on site, which will then be shipped out of the province to processing facilities. There is currently no contract in place with any smelter or buyer for the concentrate.

Metal price selection of US$1,700/oz Au and US$19/oz Ag was based on a survey of recently published feasibility studies, long-term analyst consensus prices and the two-year trailing average of gold (US$1,826/oz) and silver (US$24/oz) prices as of September 6, 2022.

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, which was retained in the feasibility study. Concentrate quality parameters are based on the results of ICP analysis of gold–silver concentrates produced during the testwork program performed by BaseMet.

An independent market study was completed by Open Mineral AG to support the NSRs used in the 2022 FS and provide opinions on potential smelters, treatment charges and penalties, and net gold and silver payable. 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.

1.19 Environmental Studies, Permitting & Social or Community Impact

1.19.1 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 and management of water throughout the site to avoid potential long-term impacts to water quality and natural resources. NAG waste rock will be deposited in two locations: approximately 90% will be stored during mine operations in the Waste Rock Storage Facility (WDW, Section 16) that will be located to the west of the North Pit. Small quantities of NAG waste rock will be used as construction material for berms and small waste dumps adjacent to the North Pit along the Tom MacKay creek channel. Detailed closure planning and engineering will be undertaken once the conceptual closure plan is finalized after engagement with regulators and Indigenous Nations. Conceptually, it may involve relocating a substantial volume of NAG waste rock (several million tonnes) backhauled to the North Pit to cover PAG pit walls and benches to mitigate MLARD risks and this will be defined during detailed closure planning. 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, assessed and if required, managed to meet permit discharge limits prior to discharge. Process water will be discharged to the TMSF.

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

1.19.2 Closure and Reclamation Planning

The objective of the mine closure strategy for the mine will be to have a stable, revegetated site with 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 land uses. A Closure and Reclamation Plan will be developed during the permitting process to achieve post-mining land use objectives (e.g. wildlife habitat and traditional use opportunities), 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 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.

1.19.3 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 at a distance 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 use Tahltan cabins along the Eskay Mine Access 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 implementing an Engagement Plan for the Project as required by the provincial and federal EA processes in collaboration with TCG. 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 was submitted with the Initial Project Description in July 2021 to begin the Early Engagement Phase of the EA/IA process and continues to guide engagement efforts. 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 address federal and provincial regulations and Indigenous Nation preferences.

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 consulting and engaging 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.

The proposed Project is anticipated to undergo a concurrent Environmental Assessment /Impact Assessment (EA/IA), called a substituted process, under federal and provincial regulations and will also be reviewed concurrently by the Tahltan Nation for a consent decision. Since the Eskay Creek Mine has two existing Certificates from 2000 and 1994, 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 for the former underground mine was listed under Schedule 2 – Tailings Impoundment Areas, of the federal Fisheries Act.

For the proposed Project, Skeena will undertake a substituted regulatory assessment process to amend an existing EA Certificate or obtain a new EA Certificate for the open pit project. 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. After 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. An updated mine reclamation security bond will be established for the open pit project in conjunction with the updated mine plan and reclamation program under the Mines Act.

Skeena will apply for amended or new permits to support the technical bulk sample (not subject to a new EA/IA) prior to the EAC is issued for the open pit project. Separate amendments or new construction and operating permits for the open pit operation will be applied for after the EAC is issued.

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 government reviewers and Indigenous representatives. Skeena will consult with the working group on project-related developments during the EA/IA process. Skeena will consult with the public, Indigenous Nations 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).

1.20 Capital & Operating Costs

1.20.1 Capital Cost

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

The capital cost estimate summarized in Table 1-5 provides a summary of the Project capital cost estimate, with costs grouped into major scope areas as presented in Skeena's new release dated September 8, 2022 "Skeena Completes Robust Feasibility Study for Eskay Creek: After-Tax NPV (5%) of C$1.4B, 50% IRR and 1 Year Payback.

The costs are expressed in Q1 2022 Canadian dollars and include all costs related to the Eskay Creek Project (e.g., mining, site preparation, process plant, tailings facility, power infrastructure, camp, Owners' costs, spares, first fills, buildings, roadworks, and off-site infrastructure).

The project will be constructed in two distinct phases: Initial (3.0 Mt/a), and Expansion to 3.7 Mt/a. The estimate is based on an EPCM execution approach for the process/infrastructure areas, and a EPCM execution for the civil-earthworks camp, and power infrastructure packages, as outlined in Chapter 24.

- The following parameters and qualifications were considered:
- No allowance has been made for exchange rate fluctuations.
- There is no escalation added to the estimate.
- A growth allowance is included.
- For equipment sourced in US dollars, relevant exchange rates were used to convert to Canadian currency.
- Data for the estimates have been obtained from numerous sources, including:

- mine schedules
- feasibility-level engineering design
- topographical information obtained from the site survey
- geotechnical investigations
- Firm and budgetary equipment quotes from Canadian and International suppliers
- budgetary unit costs from numerous local BC contractors for civil, concrete, steel, electrical, piping and mechanical works
- data from similar recently completed studies and projects

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analysed. A percentage of contingency was allocated to each of these categories on a line item basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

As outlined in Table 1-5, the total capital cost is approximated $911M over the LOM and the costs are defined as follows:

- Initial capital costs: include the costs required to construct all the surface facilities, and open pit development to commence a 3.0 Mt/a operation. The initial capital cost is estimated to be C$592M.
- Expansion and sustaining costs: include the capital cost required to expand the throughput to 3.7Mt/a operation and required to sustain operations, with the most significant component being open pit mine development. The expansion and sustaining costs are $180M 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 C$138M.

Of the total initial capital costs, more than 90% of the project costs were derived from first principles bulk material take-offs and equipment sizing calculations, with supporting quotations for major equipment, and contractor supply/installation rates.

Table 1-5: Project Capital Cost Estimate

	Initial	Expansion & Sustaining	Closure	LOM Total
Mine				
Mine Development (C$M)	98	10	-	108
Mine Other (C$M)	19	9	-	28
Mine Equipment (C$M)	8	21	-	29
Sub-Total Mine (C$M)	**125**	**40**	**-**	**166**
Process Plant				

	Initial	Expansion & Sustaining	Closure	LOM Total
Processing (C$M)	178	32	-	210
Earthworks (C$M)	19	2	-	21
Sub-Total Processing (C$M)	**197**	**34**	**-**	**231**
Infrastructure				
Onsite Infrastructure (C$M)	69	65	-	134
Offsite Infrastructure (C$M)	50	23	-	73
Sub-Total Infrastructure (C$M)	**119**	**88**	**-**	**207**
Total Directs (C$M)	**442**	**162**	**-**	**604**
Indirects (C$M)	74	10	-	84
Total Directs + Indirects (C$M)	**516**	**171**	**-**	**687**
Owner's Costs (C$M)	30	-	-	30
Total excluding Contingency (C$M)	**546**	**171**	**-**	**717**
Project Contingency (C$M)	47	9	-	56
Sub-Total Including Contingency (C$M)	**592**	**180**	**-**	**773**
Closure (C$M)	-	-	138	138
Total (C$M)	**592**	**180**	**138**	**911**

* Numbers above are rounded to the nearest integer, therefore some sub-totals may not balance due to rounding.

1.20.2 Operating Cost Estimates

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

The operating cost estimate includes mining, processing, general and administration (G&A), and accommodations costs. The operating cost estimates for the life of mine are provided in Table 1-6.

Table 1-6: Operating Cost Estimate Summary (C$)

Tonnes Milled	Initial 3.0 Mt/a (typical)		Expansion 3.7 Mt/a (typical)		LOM *	
Operating Cost	C$M/a	C$/t milled	C$M/a	C$/t milled	C$M	C$/t milled
Mining	137	45.71	97	26.21	901	30.12
Process operations and maintenance	52	17.39	60	16.23	506	16.91
G&A	16	5.38	12	3.11	126	4.20
Total	**205**	**68.47**	**169**	**45.56**	**1,533**	**51.24**

3.0Mt/a costs represent a typical production year in the initial phase
3.7Mt/a costs represent a typical production year in the expansion phase. Mining declines and more material reclaimed from stockpiles after Y6 toward Y9.

1.20.2.1 Process Costs

The operating cost estimates are based on the following assumptions:

- Based on Q1 of 2022 Canadian dollars without allowances for inflation.
- For equipment sourced in US dollars, relevant exchange rates were used to convert to Canadian currency.
- Crushing availability of 70% and plant availability of 92%
- Propane Cost – C$0.60/L
- Gasoline Cost – C$1.44/L (3-year- trailing average)
- Diesel Cost – C$1.28/L (3-year trailing average)
- Power Cost –C$0.06/kWh
- Labour was assumed to be sourced locally within the region, within BC and Alberta.

1.21 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. 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:

- Pre-production and ramp-up period of three years;

- Mine life of 9 years;
- Base case gold price of US$1,700 /oz and silver price of US$19/oz were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.76 (US$/C$)
- Cost estimates in constant Q1 2022 C$ 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$983 M over the LOM.

A 2% NSR royalty has been assumed for the Project, resulting in approximately C$100M 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% (NPV 5%) is C$2,094 M, the internal rate of return IRR is 59.5%, and payback is 1 year. On an after-tax basis, the NPV 5% is C$1,412 M, the IRR is 50.2%, and the payback period is 1 year.

A summary of the Project economics is included in Table 1-7and shown graphically in Figure 1-6.

Table 1-7: Summary of Economic Results

Description	Units	Values
After-Tax NPV (5%)	(C$M)	$1,412
After-Tax IRR		50.2%
After-Tax Payback Period	(yrs)	1.0
After-Tax NPV / Initial Capex		2.4
Pre-Tax NPV (5%)	(C$M)	$2,094
Pre-Tax IRR		59.5%
Pre-Tax Payback Period	(yrs)	0.99
Pre-Tax NPV / Initial Capex		3.5
Average Annual After-Tax Free Cash Flow (Year 1-9)	(C$M)	$293
LOM After-Tax Free Cash Flow	(C$M)	$2,110

Figure 1-6: Projected LOM Cashflow

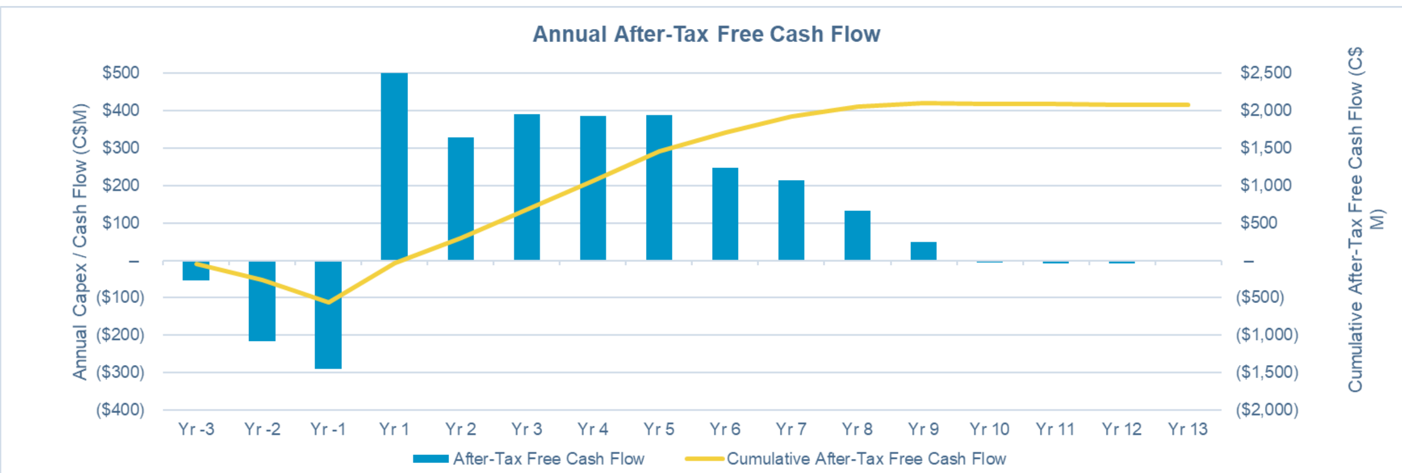


Figure prepared by Ausenco, 2022.

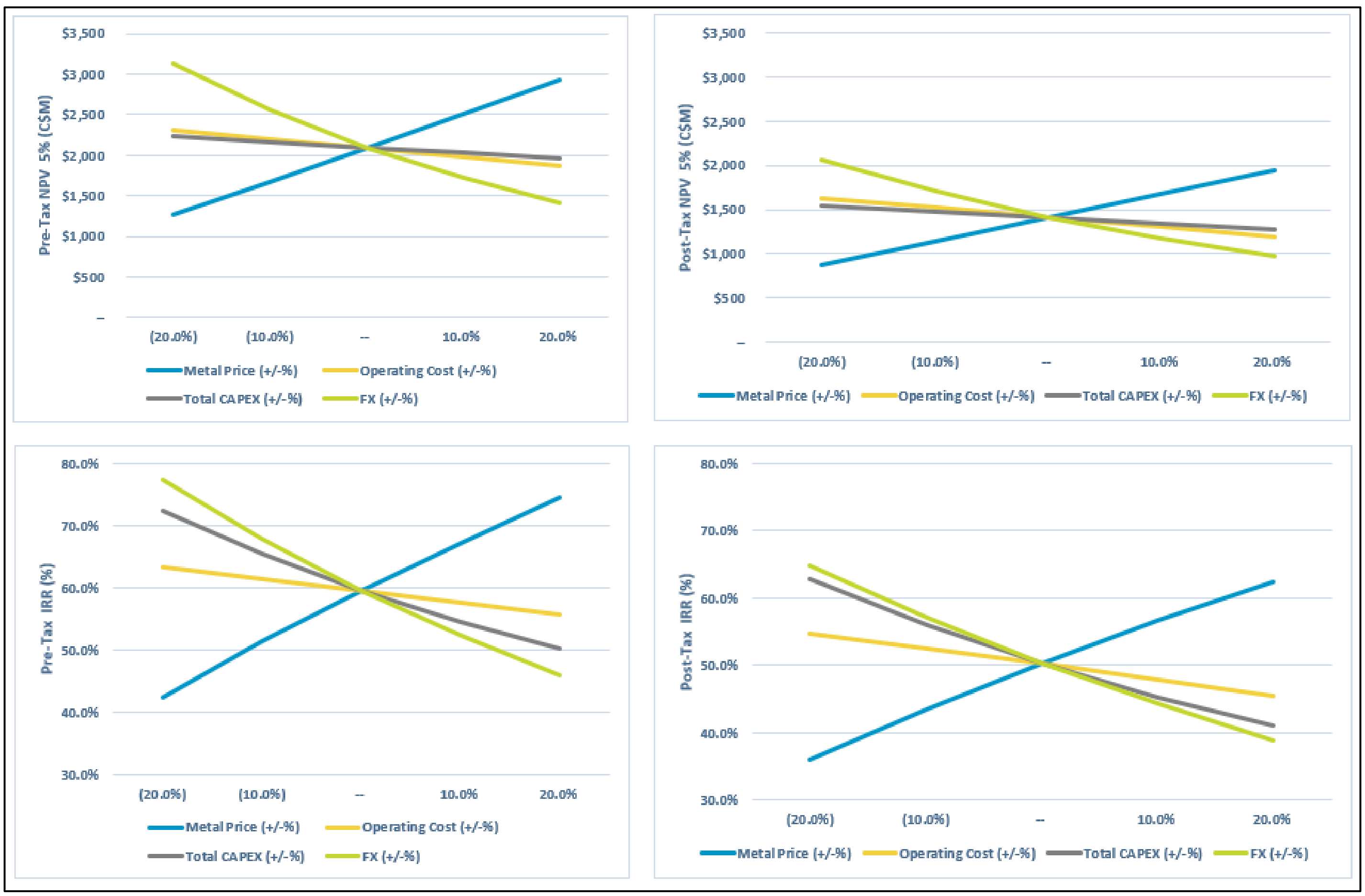
1.21.1 Sensitivity Analysis

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. Analysis revealed that the Project is most sensitive to changes in metal prices and exchange rates, 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. Table 1-8 summaries the sensitivity analysis results. Table 1-8 shows the pre-tax and post-tax sensitivity analysis findings.

Table 1-8: Sensitivity Analysis Summary

Sensitivity Summary	Even Lower Case	Lower Case	Base Case	Higher Case	Upside Case
Gold Price (US$/oz)	1,500	1,600	1,700	1,800	1,900
Silver Price (US$/oz)	15	17	19	21	23
After-Tax NPV(5%) (C$M)	1,044	1,228	1,412	1,596	1,780
After-Tax IRR (%)	41.0	45.7	50.2	54.6	58.7
After-Tax Payback (years)	1.29	1.14	1.01	0.93	0.83
After-Tax NPV / Initial Capex	1.8 x	2.1	2.4	2.7	3.0
Average Annual After-tax Free Cash Flow (year 1-10) (C$M)	237	265	293	321	350

Figure 1-7: NPV & IRR Sensitivity Results



Note: Figure prepared by Ausenco, 2022.

1.22 Interpretation and Conclusions

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.

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. The Mineral Reserve and Mineral Resource estimations for the Project both conform to industry-accepted practices and are reported using the 2014 CIM Definition Standards.

The proposed mine life includes three years of pre-stripping and 8 years of mining. Mill feed will be stockpiled during the pre-production years which will feed the mill after mining operations. A technical sample and two small quarries will be mined in pre-production so that process performance of the mill can be evaluated with a large representative feed sample of approximately 10 kt.

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.

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

The overall Project timeline will comprise three years of construction, during which time major activities will include bulk earthworks, development of the open pit mine and onsite and offsite infrastructure, and construction of the processing plant. Towards the end of construction, some pre-production will be fed to the processing plant. The processing plant will then operate for nine years, with a plant expansion to enable higher throughput taking place by the end of Year 5.

1.23 Recommendations

1.23.1 Overall

The financial analysis of this feasibility study demonstrates that the Eskay Creek Project has robust economics, and it is recommended to continue developing the project through engineering and de-risking, and into a construction decision.

Analysis of the results and findings from each major area of investigation completed as part of this FS suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. The following sections summarize the key recommendations arising from this FS. Each recommendation is not contingent to a subsequent one. Table 1-9 presents a summary of recommended tasks and budget and detailed in the subsections -that follow.

Table 1-9: Proposed Budget Summary

Description	Cost (C$)
Drilling	10,000,000
Metallurgical Testwork	600,000
Materials Handling Testwork	65,000
Mine Geotechnical	500,000
Mine Studies	400,000
Hydrological	75,000
Water Treatment	300,000
Meteorological Update	N/A
Infrastructure Geotechnical and Construction Borrow Material	1,300,000
Environmental, Permitting and Social	4,600,000
Water Sampling and PAG/NAG Evaluation	5,000,000
Total	**22,840,000**

1.23.2 Drilling

Skeena are currently drilling 60,000 m using skid-mounted and heli-portable drills during 2022 of which 42,000 m has been completed. This program is estimated with all-in drilling costs of $555/m, with a remaining spend of approximately $10.0 M. At program completion, the intent is to update the block model and resource estimate.

1.23.3 Sampling and QA/QC

- The QA/QC measures implemented in the initial 2018–2019 drill programs should be retained for future drill campaigns.
- Lithological, alteration, mineralization and structural data captured during future drilling 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.
- 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.

1.23.4 Mineral Processing & Metallurgical Testing

For the flowsheet selected during the 2022 FS, additional testwork is proposed to further refine metallurgical performance estimates and equipment sizing. Specifically, additional testwork should include:

- Variability testing on samples with selected feed characteristics, to improve the recovery and concentrate grade models and confirm metallurgical performance;

- Comminution testing on new samples to increase the database of results to optimize comminution power;
- Investigate opportunities to apply coarser grinds to samples that have higher work indices and likely higher SiO_2 contents;
- Investigate cleaner circuit flowsheet modifications to optimize regrind energy application, improve concentrate quality and improve dewatering performance;
- Optimisation of HW/Mudstone flowsheet conditions and confirm the impact of blending at 20 to 50% of plant feed;
- Generation of additional regrind mill feed samples for vendor testing (both IsaMill and HIGmill), also complete bench scale regrinding testing where applicable;
- Generation of additional final concentrate samples for filter/dryer vendor testing; and
- Material handling test work is recommended on crushed material and concentrate for design of bins, chutes, conveyors, and stockpile drawdown. This program is estimated at $65 k.

The last two items will require bulk samples and pilot plant runs to generate sufficient mass for testing.

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.

1.23.5 Mining Methods

1.23.5.1 Mine Geotechnical

Further data collection and interpretation tasks are recommended to fill in data gaps to support future stages of design. These include:

- Improvements to geotechnical core logging methodology, including the addition of Joint Roughness Coefficient (JRC), degree of alteration/weathering, fracture spacing, number of discontinuity sets, identification of faults/shears, logging of both “worst-case” and “representative” discontinuities (if not feasible to log all discontinuities), logging of joint roughness number (Jr) and joint alteration number (Ja) for every discontinuity, and the use of geotechnical intervals instead of runs for the main delineation of logging units.
- The collection of supplementary structural data in areas of the open pit and diversion tunnel where existing data is sparse or where additional data it is required to validate design inputs. Surface mapping is recommended to obtain information on discontinuity persistence and waviness across the open pit area and at the diversion tunnel portal locations. Additional characterization of the location, orientation and geotechnical characteristics of major structures (i.e., fault and shear zones) is also recommended.
- Supplementary laboratory strength data, particularly in the Hanging Wall Mudstone, Contact Mudstone, Footwall Sediments and Bowser Sediments units where existing laboratory data is limited. Additionally, discontinuities were not systematically tagged by structure type during the 2020 and 2021 drilling programs, so it was not possible to develop relationships between discontinuity type and shear strength. This is recommended for future studies.
- Calibration of rock fall analyses at portal locations based on ongoing observations of rock fall activity along the Tom Mackay Creek valley.

- An in-situ stress study in the diversion tunnel area. An analysis of borehole breakouts from televiewer surveys may provide information on in-situ stresses.

A budget of approximately $500 K is recommended.

1.23.5.2 Mine Studies

The following areas should be addressed during more detailed studies. These studies are collectively estimated at $400 k.

1.23.5.2.1 Grade Control

The 2022 FS 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.

1.23.5.2.2 Geology Model Improvement

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.

1.23.5.2.3 Dewatering Requirements

A proper understanding of pumping requirements and the hydrogeology is critical. Further work assessing this is recommended. Additional hydrogeological testing including packer testing, piezometer installations, pumping well construction and long-term aquifer testing is recommended. A numerical groundwater flow model should be calibrated and developed under transient conditions to inform subsequent geotechnical evaluations and depressurization assumptions.

1.23.5.2.4 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.

1.23.5.2.5 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.

1.23.6 Recovery Methods

The following activities are recommended to support the design of the processing plant beyond the feasibility study:

- Incorporate the aforementioned materials handling testwork (material flowability testwork results) and recommendations should be incorporated into the crushing and stockpile circuit detailed design.

- Incorporate the aforementioned metallurgical testwork to refine comminution, flotation and concentrate handling equipment sizing

1.23.7 Project Infrastructure

The following activities are recommended to support the detailed design of the Project infrastructure beyond the feasibility study:

- Further confirmatory geotechnical site investigations should be carried out at the preferred surface infrastructure site locations to characterise the foundation conditions associated with the proposed buildings, identify borrow material sources for construction activities, and provide information for support of the WRSF, plant site, ancillary facilities locations, and the TMSF designs. This program is estimated at $1.3 M.
- Further logistics planning and route surveys. This program is estimated at $100 k.
- The design of the 69 kV high-voltage powerline and substation should be further refined by BC Hydro and consultants.
- Additional wells will need to be installed and pumping tests carried out to establish sustainable yield and to support licensing. This program is estimated at $700 k.
- 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.

1.23.8 Environmental Studies, Permitting, & Social or Community Impact

These recommendations will focus on project environmental, permitting, and social de-risking activities, which will include:

- Continuation of a suite of monitoring and baseline environmental studies, some of which have been ongoing since 2020 for documentation of current conditions since 2020. Much of the baseline data collection since 2020 has established new or re-used historic baseline/permit compliance/monitoring sampling locations and future monitoring programs of a suite (e.g. climate, hydrology, surface and groundwater quality and quantity) will be useful for long-term monitoring and ongoing permit compliance as well as collaboration with Indigenous Nations.
- Proceeding through permitting and EA/IA and Indigenous processes and relationship building;
- Documenting the required data to support applications for operating permits and completion of such applications;
- Consultation, engagement 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.

Additionally, to characterize the waste rock material and minimize PAG, a lab could be established on the site to support PAG/NAG evaluation. This same lab could also support water quality testing and modelling to further validate the removal of the water treatment plant. A budget of approximately $5 M is recommended for this lab.

1.23.9 Meteorological Update

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 was installed at the Project in 2020 to provide a correlation between the Eskay Creek and KSM project data sets and continued monitoring will inform future project design and modelling as well as tracking of potential changes in site specific data trends.

2 INTRODUCTION

2.1 Terms of Reference & Purpose of this Report

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) along with SRK Consulting (Canada) Inc. (SRK), and AGP Mining Consultants Inc. (AGP) for Skeena Resources to summarise the results of the N.I. 43-101 Technical Report and Feasibility Study of the Eskay Creek Project (the Project) located in British Columbia (Figure 2-1). The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

BGC Engineering Inc. (BGC), and Environmental Resources Management (ERM) provided input to the report, and the individuals presented in Table 2-1, by virtue of their education, experience, and professional association, are considered Qualified Persons (QPs) as defined by N.I. 43-101. The QPs meet the requirement of independence defined in N.I. 43-101.

The Report supports disclosures by Skeena in a news release dated September 8, 2022, entitled, "Skeena Completes Robust Feasibility Study for Eskay Creek After-Tax NPV (5%) of C$1.4 B, 50% IRR and 1 Year Payback."

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).

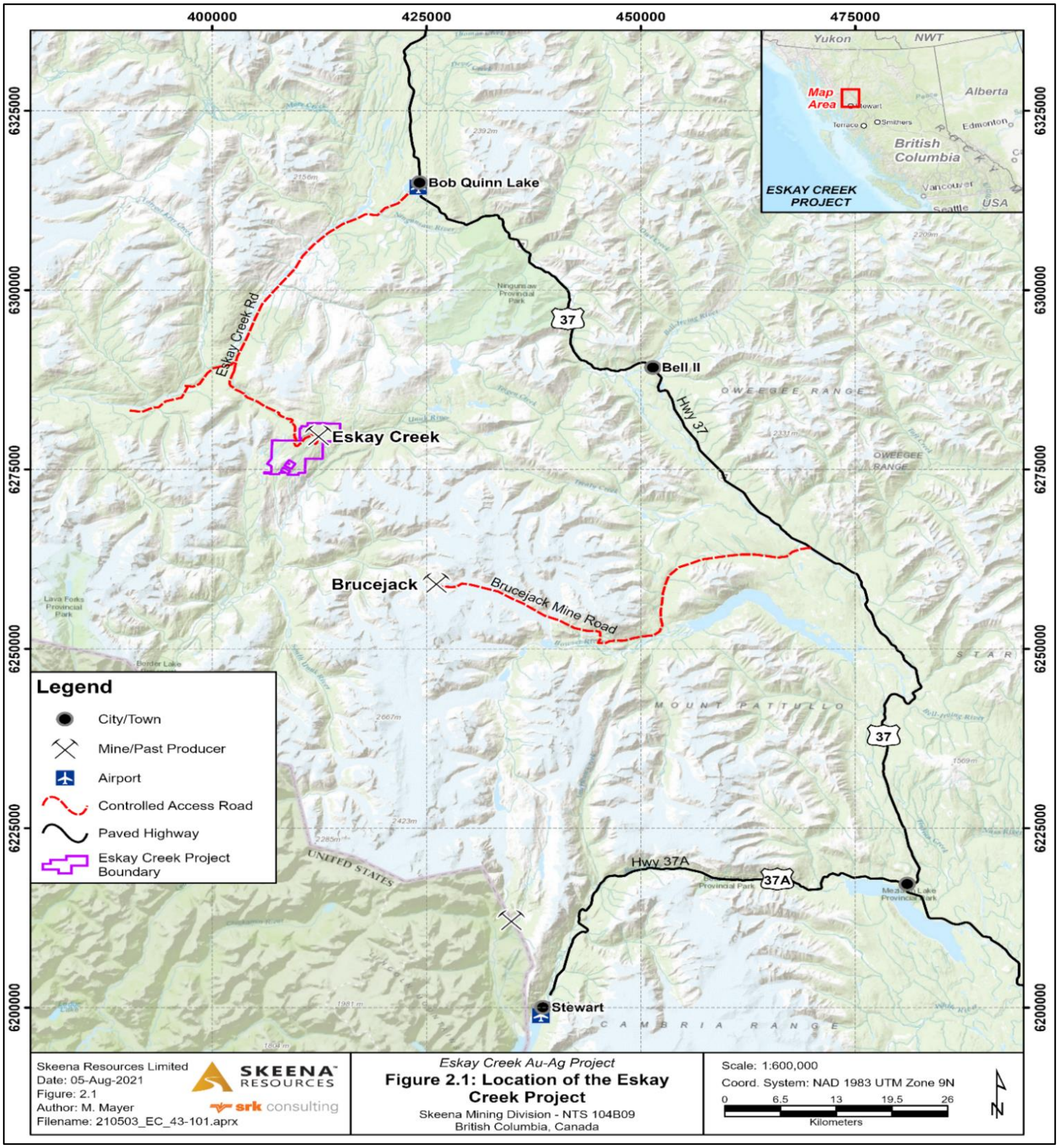
Skeena owns 100% of the Eskay Creek Gold-Silver Project, where it was purchased from Barrick Gold in October 2020.

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

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

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.

2.2 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 accordance with Form 43-101F1:

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of [Company]	Report Section
Kevin Murray	P.Eng.	Process Engineering Manager	Ausenco	Skeena	1.1-1.3, 1.16, 1.17.1-1.17.3, 1.17.8-1.17.10, 1.18, 1.20, 1.21, 1.22, 1.23.1, 1.23.6, 1.23.7, 2, 3, 17, 18.1-18.5, 18.14, 19, 21.1.1, 21.1.4-21.1.6, 21.1.7, 21.1.8-21.1.12, 21.2.1, 21.2.3, 21.2.4, 22, 24, 25.8, 25.9.1, 25.9.5, 25.11-25.13, 25.14.1.1, 25.14.1.5, 25.14.1.8, 25.14.2.5, 25.14.2.6, 25.14.2.8, 26.1, 26.5, 26.6, and 27
Mohammad Ali Hooshiar Fard	P.Eng.	Geotechnical Engineer	Ausenco	Skeena	1.17.4, 1.17.6, 16.4.2, 16.11.2, 16.11.3, 18.6-18.9, 18.11, 20.2.3, 21.1.3, 25.9.2, 25.10.2, and 25.14.1.6
Gerry Papini	P.Geo.	Hydrogeologist	Ausenco	Skeena	1.15.2, 1.17.5, 16.4.1, 16.4.3, 16.4.4, 18.12, 25.7.2, and 25.9.3
Davood Hasanloo	MASc., P.Eng.	Senior Water Resources Engineer	Ausenco	Skeena	1.17.7, 18.10, 18.13, and 25.9.4
Peter Mehrfert	P. Eng.	Principal Metallurgist	Ausenco	Skeena	1.12, 1.23.4, 13, 25.4, and 26.3
Sheila Ulansky	P. Geo.	Senior Resource Consultant	SRK	Skeena	1.5-1.11, 1.13, 1.23.2, 1.23.3, 4.1-4.3, 4.6, 4.10, 6-12, 14, 25.1, 25.2, 25.3, 25.5, 25.14.1.2, 25.14.1.3, 25.14.2.1-25.14.2.3, and 26.2
Rolf Schmitt	P. Geo.	Technical Director	ERM	Skeena	1.4, 1.19, 1.23.8, 1.23.9, 4.4, 4.5, 4.7-4.9, 5, 20.1-20.2.2, 20.3-20.6, 23, 25.10.1, 25.10.3-25.10.5, 25.14.1.7, 25.14.2.7, and 26.7
Willie Hamilton	P. Eng.	Principal Mine Engineer	AGP	Skeena	1.14, 1.15.3, 1.23.5.2, 15.1, 15.3-15.8, 16.1, 16.2, 16.5-16.10, 16.11.1, 16.12-16.16, 21.1.2, 21.2.2, 25.6, 25.7.3, 25.14.1.4, 25.14.2.4, and 26.4.2
Ian Stilwell	P. Eng.	Principal Geotechnical Engineer	BGC	Skeena	1.15.1, 1.23.5.1, 15.2, 16.3.1-16.3.7, 16.3.9, 25.7.1, and 26.4.1
Catherine Schmid	P. Eng.	Senior Geotechnical Engineer	BGC	Skeena	16.3.8

2.3 Site Visits and Scope of Personal Inspection

Ms. Ulansky, representing SRK, visited the Eskay Creek property from June 27–28, 2018. During that visit she viewed the general topography, independently located and surveyed 50 surface drill hole collars, and inspected the existing mine infrastructure. In addition, Ms. Ulansky completed a site visit from July 27–31, 2020, where she reviewed surface drill core to confirm the presence and nature of mineralization and appropriateness of the interpreted geological framework for resource modelling and estimation.

Mr. Papini representing Ausenco, visited the site on August 30–31, 2022. The objectives of the visit were to observe drainage conditions, locations of proposed infrastructure and locations for monitoring well installations. The visit was carried out with review consultants (Ground Water Solutions and Knight Piesold) and Tahltan Nation representative to discuss groundwater flow path, the conceptual groundwater model and monitoring requirements. The visit included a helicopter tour.

Mr. Hamilton, representing AGP, visited the Eskay Creek site from August 21–22, 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 27–28, 2020, and July 14–15, 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, Mr. Elfen reviewed the general topography and geotechnical surface conditions for the site wide infrastructure to ensure there were no significant geotechnical issues.

Mr. Stilwell, representing BGC, visited the Eskay Creek site from September 1-3, 2021. During his time on-site, he carried out a helicopter reconnaissance of the pit area, including bedrock exposures along the Tom MacKay Creek gulley. He also focused his time familiarizing himself with the various geotechnical units in representative sections of drill core and reviewing Ausenco's geotechnical and oriented core logging, sampling and point load index testing procedures. This included procedures being carried out at the drill rig and at a temporary on-site core storage facility setup by Ausenco. A closeout meet was held with Skeena and Ausenco staff on September 3, 2022.

Ms. Schmid, representing BGC, visited the Eskay Creek site from September 1-3, 2021. She accompanied Mr. Stilwell during his site visit. During her time on-site, she carried out a helicopter reconnaissance of the proposed tunnel alignment, including bedrock exposures along the Tom MacKay Creek gulley and proposed portal locations. She also focused her time familiarizing herself with the various geotechnical units in representative sections of drill core and reviewing Ausenco's geotechnical and oriented core logging, sampling and point load index testing procedures. This included procedures being carried out at the drill rig and at a temporary on-site core storage facility setup by Ausenco. A closeout meeting was held with Skeena and Ausenco staff on September 3, 2022.

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.

2.4 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: May 12, 2022;
- date of supply of most recent information on ongoing drill program: September 10, 2021;
- mineral resource estimate: January 18, 2022
- mineral reserve estimate: June 30, 2022
- date of FS financial analysis: September 6, 2022.

The overall effective date of this Report is the effective date of the financial analysis which is September 6, 2022.

2.5 Information Sources and References

The Report is primarily based on work done for Skeena during the feasibility study 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.

The authors have not attempted to verify the legal status of the property; however, the Government of British Columbia, Natural Resources' online mineral claims staking system, Mineral Rights Administration System (MIRIAD), reports that the Skeena Resources mineral claims are active and in good standing at the effective date of this report.

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

Mr. P. Mehrfert (QP) relied on the assistance of Dr. Adrian Dance (SRK Consulting) for Section 13 of this Technical Report. Dr. Dance wrote Sections 13.1 through 13.3 and contributed to other sections prior to Mr. Mehrfert's review. This information is used in Section 13 of the Report.

Reports and documents listed in Section 3 and Section 27 of this Report were used to support preparation of the Report.

2.6 Previous Technical Reports

Skeena has filed the following technical reports on the Project:

- Raponi, R., Elfen, S., Ulansky, S., Schmitt, R., Dance, A., Hamilton, W., Tosney, R., 2021: NI 43-101 Technical Report and Prefeasibility Study, Canada: report prepared by Ausenco Engineering Canada Inc. for Skeena, effective date 22 July 2021.
- 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 Engineering Canada Inc., 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.

2.7 Units of Measurement

Abbreviation	Description
3D	Three-Dimensional
°C	degrees Celsius
C$	Canadian dollars
US$	United States dollars
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

Abbreviation	Description
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
X	times

2.8 Abbreviations and Acronyms

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

Abbreviation	Name
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
COI	Constituent of interest
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
DGPS	Differential Global Positioning System
DTH	Down the hole Hammer
DWi	Mill Comminution
EA	Economic Assessment
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

Abbreviation	Name
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
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/Quality Control
QP	Qualified Person
R&M	Repair and maintenance

Abbreviation	Name
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
SPI	SAG Power Index
SRK	SRK Consulting Canada Inc.
SRM	Standard Reference Materials
SWP	Stewart World Port
TDS	Total Dissolved Solid
TMSF	Tom MacKay 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

While the authors have carefully reviewed, within the scope of their technical expertise, all the available information presented to them, they cannot guarantee its accuracy and completeness. The authors reserve the right, but will not be obligated to, revise the technical report and its conclusions if additional information becomes known to them after the effective date of this Report.

3.1 Taxation

The QPs have not independently reviewed the project taxation position. The QPs have fully relied upon, and disclaim responsibility for, taxation information derived from MNP who was retained for this information. Project taxation information has been provided through email communications titled "RE: Discuss with tax consultants on modelling assumption/details" on August 31, 2022.

This information is used in support of Section 22 of the Report.

3.2 Market Studies

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

Skeena engaged Open Mineral, expert in the marketing of complex gold concentrates. Open Mineral conducted studies and provided absolute opinions on potential smelters, treatment charges and penalties, and net gold and silver payable. Open Mineral's reports titled "Eskay Creek Au concentrate marketability" prepared on March 2022 and PowerPoint titled "Eskay Creek Au concentrate marketability August 2022", file name "EskayCreek_Au conc marketability_August update2022_v2" prepared August 2022 were made available to the QP. 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. K Murray considers it reasonable to rely on such information as the consultants are specialists in commodities trading. Detailed information outlining all payables, penalties, deductions, and charges, was provided to arrive at an 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 5,798.84 hectares (ha) and consists of the following (Figure 4-1):

- Forty-nine mineral claims totalling 3,968.58 ha (Table 4-1)
- Eight mineral leases totalling 1,830.26 ha (Table 4-2).

Forty-nine 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 Limited.

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 following:

- 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$17.5M.

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

Table 4-1: Mineral Claim Summary

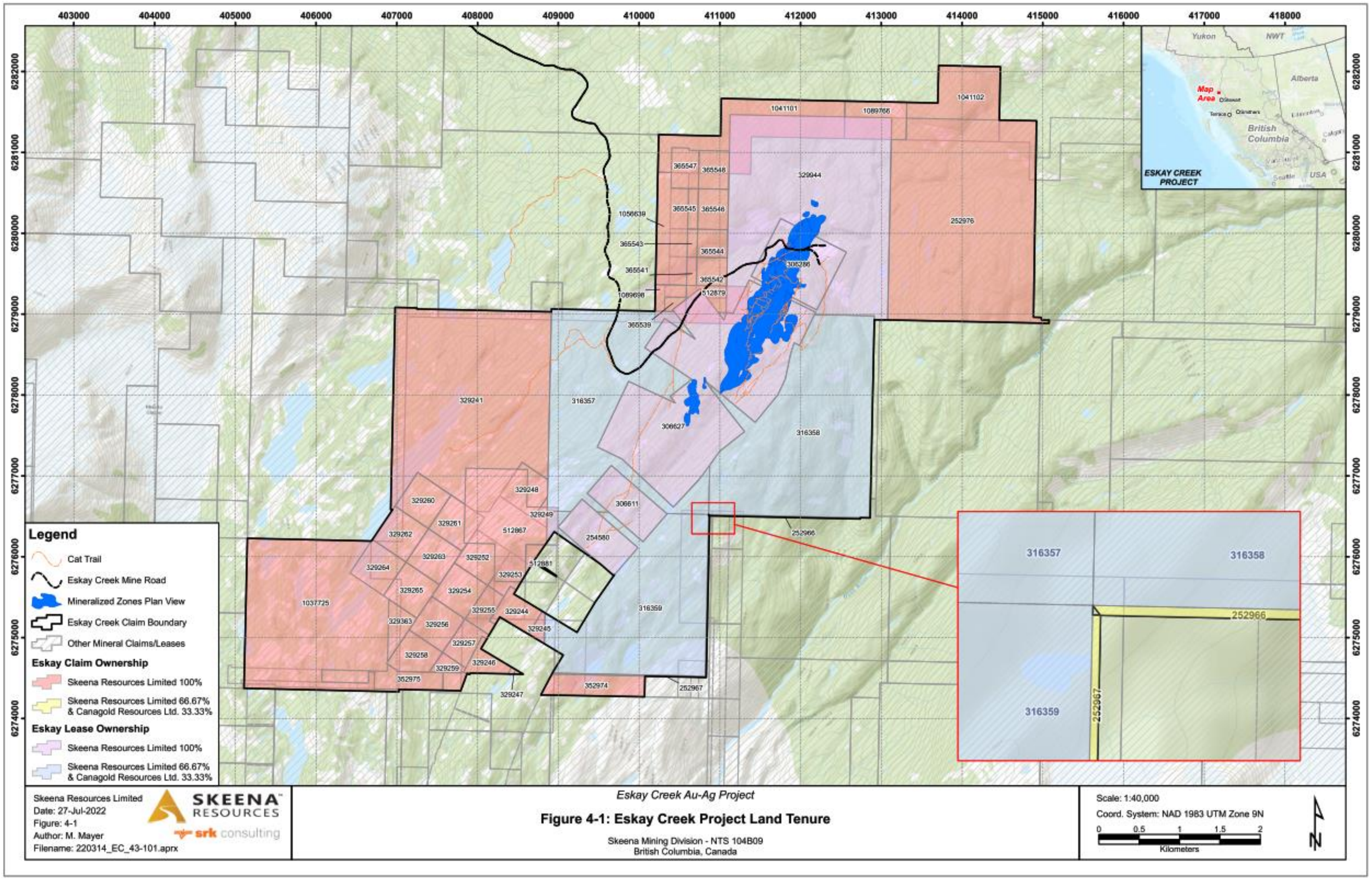
Tenure Number	Claim Name	Issue Date	Good to Date	Area (ha)	Owner Name	Number of Owners
252966	CAL #2	1989/08/05	2023/JAN/15	500	SKEENA	2
252967	CAL #3	1989/08/06	2023/JUN/22	400	SKEENA	2
252976	IKS 2	1989/08/02	2025/JUL/12	500	SKEENA	1
300298	P-1	1991/06/11	2023/MAY/20	25	SKEENA	1
300299	P-2	1991/06/11	2023/MAY/20	25	SKEENA	1
300300	P-3	1991/06/11	2023/MAY/20	25	SKEENA	1
300301	P-4	1991/06/11	2023/MAY/20	25	SKEENA	1
329241	MACK 23	1994/07/21	2025/JUN/25	500	SKEENA	1
329244	MACK 1	1994/07/21	2025/JUN/25	25	SKEENA	1
329245	MACK 2	1994/07/21	2025/JUN/25	25	SKEENA	1
329246	MACK 3	1994/07/21	2025/JUN/25	25	SKEENA	1
329247	MACK 4	1994/07/21	2025/JUN/25	25	SKEENA	1
329248	MACK 5	1994/07/21	2025/JUN/25	25	SKEENA	1
329249	MACK 6	1994/07/21	2025/JUN/25	25	SKEENA	1
329252	MACK 9	1994/07/21	2025/JUN/25	25	SKEENA	1
329253	MACK 10	1994/07/21	2025/JUN/25	25	SKEENA	1
329254	MACK 11	1994/07/21	2025/JUN/25	25	SKEENA	1
329255	MACK 12	1994/07/21	2025/JUN/25	25	SKEENA	1
329256	MACK 13	1994/07/21	2025/JUN/25	25	SKEENA	1
329257	MACK 14	1994/07/21	2025/JUN/25	25	SKEENA	1
329258	MACK 15	1994/07/21	2025/JUN/25	25	SKEENA	1
329259	MACK 16	1994/07/21	2025/JUN/25	25	SKEENA	1
329260	MACK 17	1994/07/21	2025/JUN/25	25	SKEENA	1
329261	MACK 18	1994/07/21	2025/JUN/25	25	SKEENA	1
329262	MACK 19	1994/07/21	2025/JUN/25	25	SKEENA	1
329263	MACK 20	1994/07/21	2025/JUN/25	25	SKEENA	1
329264	MACK 21	1994/07/21	2025/JUN/25	25	SKEENA	1
329265	MACK 22	1994/07/21	2025/JUN/25	25	SKEENA	1
329363	MACK 26 FR.	1994/08/03	2025/JUN/25	25	SKEENA	1
352974	STAR 21	1996/12/07	2023/JUN/22	250	SKEENA	1
352975	STAR 22	1996/12/07	2025/JUN/25	150	SKEENA	1
365539	KAY 1	1998/09/12	2025/OCT/06	25	SKEENA	1

Tenure Number	Claim Name	Issue Date	Good to Date	Area (ha)	Owner Name	Number of Owners
365541	KAY 3	1998/09/12	2025/OCT/06	25	SKEENA	1
365542	KAY 4	1998/09/12	2025/OCT/06	25	SKEENA	1
365543	KAY 5	1998/09/12	2025/OCT/06	25	SKEENA	1
365544	KAY 6	1998/09/12	2025/OCT/06	25	SKEENA	1
365545	KAY 7	1998/09/12	2025/OCT/06	25	SKEENA	1
365546	KAY 8	1998/09/12	2025/OCT/06	25	SKEENA	1
365547	KAY 9	1998/09/12	2025/OCT/06	25	SKEENA	1
365548	KAY 10	1998/09/12	2025/OCT/06	25	SKEENA	1
512867	<Null>	2005/05/17	2023/JUN/25	106.8	SKEENA	1
512879	<Null>	2005/05/18	2023/APR/06	35.58	SKEENA	1
512881	<Null>	2005/05/18	2023/JUN/25	17.8	SKEENA	1
1037725	ESKAY CREEK MAC 25	8/4/2015	2023/OCT/04	338.3283	SKEENA	1
1041101	ESKEY CREEK TREND	1/9/2016	2023/SEP/10	124.4705	SKEENA	1
1041102	ESKEY CREEK 1983 FILE	1/9/2016	2023/JUL/10	88.9027	SKEENA	1
1056639	MELISSA	2017/11/24	2023/OCT/06	53.35	SKEENA	1
1089698	ESKAY 3	2022-01-21	2023/JAN/21	17.79	SKEENA	1
1089766	ESKAY 1	2022-01-21	2023/JAN/21	35.56	SKEENA	1

Table 4-2: Mineral Tenure Summary

Tenure Number	Issue Date	Good to Date	Area (ha)	Owner Name	Percent Ownership	Number of Owners
316357	1994/04/30	2023/APR/30	276.7	SKEENA	66.67	2
316358	1994/04/30	2023/APR/30	367.7	SKEENA	66.67	2
306627	1992/06/01	2023/JUN/01	355	SKEENA	100	1
306286	1991/08/13	2023/AUG/13	73.56	SKEENA	100	1
306611	1992/06/01	2023/JUN/01	41.8	SKEENA	100	1
316359	1994/04/30	2023/APR/30	278.7	SKEENA	66.67	2
254580	1990/12/17	2023/DEC/17	41.8	SKEENA	100	1
329944	1994/12/06	2023/DEC/06	395	SKEENA	100	1

Figure 4-1: Mineral Tenure Location Plan



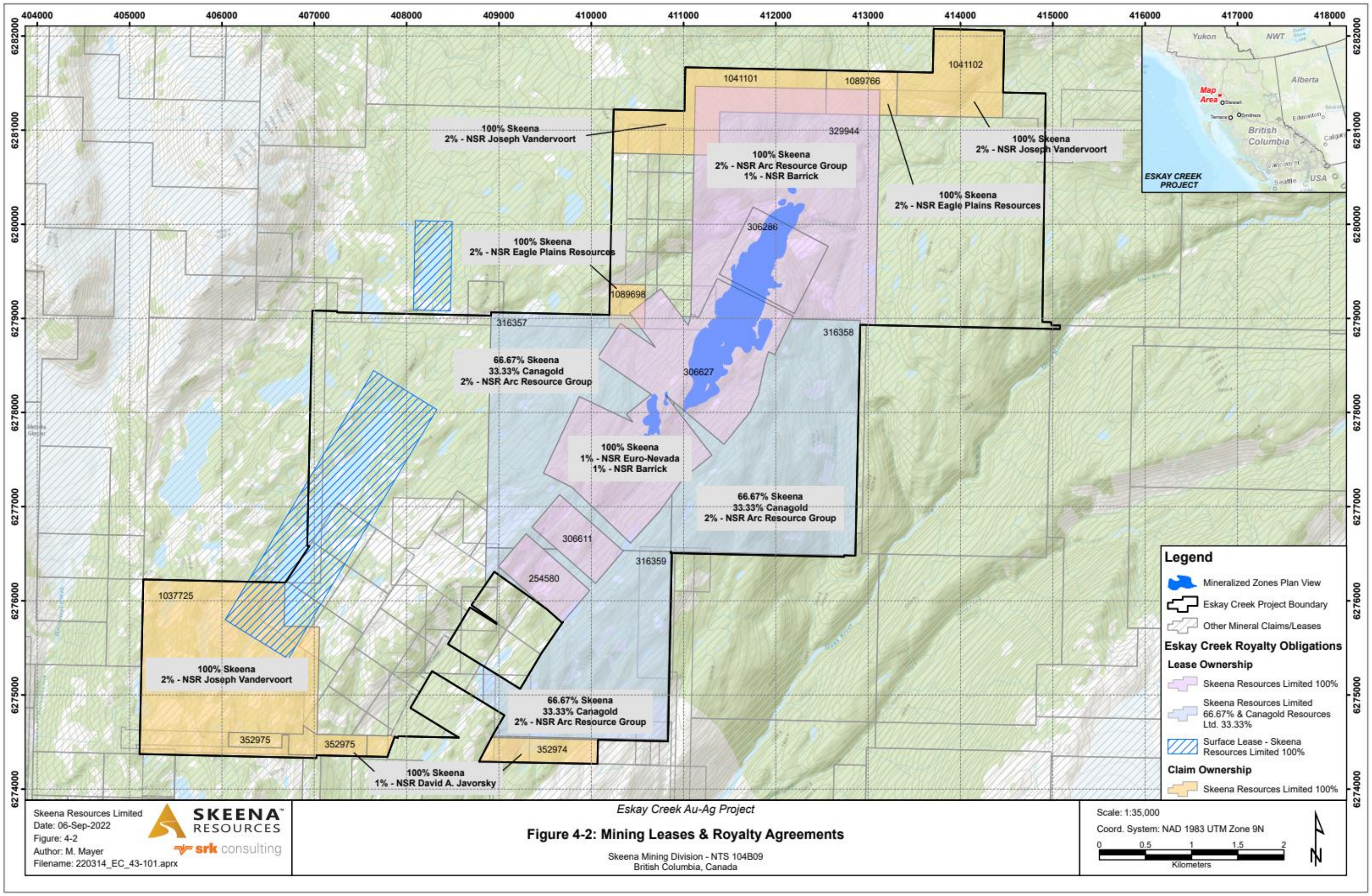
4.4 Surface Rights

Skeena holds the following surface rights interests:

- Surface lease number 634309 (December 24, 1994) between the Province of BC and Prime Resources Group Inc.; interest assigned to Skeena.
- Surface lease number 740715 (July 25, 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 (March 2, 1994) between the Province of BC and Prime Resources Group Inc.; interest assigned to Skeena on October 9, 2020
- Conditional Water Licence 114327 (effective April 20, 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's *Water Sustainability Act* for the proposed project. Specifically, the following *Water Sustainability Act* authorizations will include the following:

- 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.
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.
Joseph Vandervoort • Eskay Creek MAC 25 • Eskey Creek Trend • Eskey Creek 1983 File	2% NSR
Eagle Plains • Eskay 1 and 3	2% NSR
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.

Notes: **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. and Stikine Resources Ltd. (both now Barrick) and Adrian Resources Ltd. **3.** This agreement references the Royalty Deed dated August 1, 1990, between ARC Resource Group Ltd. and Adrian Resources Ltd. **4.** Option and Joint Venture Agreement dated November 4, 1988, between Canarc Resources Corp and Calpine Resources Incorporated (now Barrick). **5.** NSR Royalty Agreement w. Option to Purchase dated November 3, 2004, between Homestake Canada Inc. (now Barrick) and David A. Javorsky. **6.** Royalty Agreement dated October 2, 2020, between Skeena Resources Limited, and Barrick Gold Inc.

4.7 Permitting Considerations

Skeena has obtained provincial permits pursuant to the Mines Act, Water Sustainability Act, Environmental Management Act and other relevant regulations to authorize the activities to undertake the recommended work program. Permitting related to project development is discussed in further detail 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's consider that there are no significant factors or material risks that may affect access, mineral tenure, title or the right or ability to perform work on the Property. All mineral tenure, mining leases and crown land title is in good standing. Surface and aerial access to the project site is permitted and well-established. Permits to authorize work program activities are in place and applied for sufficiently in advance of work requirements.

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

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 (not connected to the road system) 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.

Travel to the property from local population centres will be primarily by Highway 16 (e.g. Terrace or Smithers) and via Highway 37 north to Bob Quinn and Eskay Mine Access road junction; however, there is a possibility to fly to Bob Quinn airport and shuttle by access road from there.

5.1 Climate

Based on data collected at the project area (elev. 1,033 masl), the total precipitation is estimated to be 1,319 mm (2021 January to October) and 1,795 mm (November 2020 to October 2021). The majority (60%) of annual precipitation falls as snow. Snowpack data indicates a peak in April of 2,111 cm. Cumulative snowfall data across four snow courses at the mine site collected between March to April of 2021 indicates a range of 132 – 488 cm of snow (elev. range 1,003 to 1,098 masl). The average daily temperature range is from -29.4°C (9 February 2021) to 25.4°C (28 June 2021), and the 12-month mean value was 0.5°C. Expected extreme temperatures range from -40°C to +30°C (SRK, 2019). An ongoing meteorology study continues to update this information and supports the other environmental baseline studies (Section 20).

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.

5.2 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 approximately 400 km south-east of the project and are assessable by provincial highways. Both communities are accessible by commercial airlines with regular flights to and from Vancouver.

Labour in support of mining activities can be locally and regionally 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.

5.3 Physiography

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (masl), 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 serrated 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). Additional information on soils, geohazards, and terrain stability are included in Section 20.

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. Additional information on hydrology is included in Section 20.

6 HISTORY

6.1 Ownership and Exploration History

The property has been explored by several companies since the early 1930's. Table 6-1 is a summary of the known ownership history, development and exploration work that has been undertaken on the Eskay Creek Project by various operators since 1932.

Table 6-1: Ownership and Exploration Summary on the Eskay Creek Project by Year

Year	Owner	Work Area	Description
1932	Unuk Gold/Unuk Valley Gold Syndicate	Unuk & Barbara Group claims (Core property)	Tom MacKay, A.H. Melville and Q.A Prout staked the Unuk and Barbara Group claims. The discovered a large, silicious, heavily pyritized zone carrying sphalerite, galena and some chalcopyrite with locally encouraging gold values.
1933	MacKay Syndicate	Unuk & Barbara Claims	Six open cuts were excavated and reported significant gold values.
1934	MacKay Syndicate/Unuk Valley Gold Syndicate	Unuk, Barbara & Verna D. Group Claims	Core drilling (261.21 m) from 11 drill holes was undertaken. Some prospecting was done around in the "dioritic" rocks of the Proust Dome (Unuk 13 claim) however generally low to sporadic gold values were obtained. Several quartz stringers with pyrite, spaherite and some galena were exposed in porphyritic lava at the north end of the property, with the best assay being 4.6 g/t Au and 21.2 g/t Ag in a grab sample.
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 Au and Ag showings. The showings were numbered, and these names are still in use today. Exploration initially focused on the south end of the property and gradually moved northward.
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. A second adit was driven 18.3 m at the #13 zone.
1940-1945			No activity due to World War II
1946	Canadian Exploration Ltd.	MacKay Adit	Optioned property. Conducted mapping and trenching. Underground development was extended in the MacKay Adit to 109.73 m a raise to surface at 46 m was put in.

Year	Owner	Work Area	Description
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 No.5, 21 and 22 Zones. Core drilling (22 holes) Near the MacKay adit, 13 closely spaced holes encountered some gold assays greater than 30 g/t in plagioclase phyric rock however the gold seemed to have an erratic distribution. Over 320m of trenches were excavated on the No. 21 zone. Minor veins filled with tetrahedrite and minor galena and sphalerite were noted in well fractured felsic rock. Seven drill holes intersected narrow veins that assayed thousands of grams of silver, but they were not abundant. At the No.22 Zone, 20 trenches totaling 250m and two diamond drill holes encountered mineralization similar to that in the #21 Zone. In the No.5 Zone, six trenches over 90m exposed relatively massive sphalerite, galena and pyrite mineralization.
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. Six underground drill holes (224.64 m) in the Emma adit were drilled and vein widths up to 4 m were encountered. The drift walls were sampled. Au, Ag, Pb and Zn minerals were found to occur mainly in the volcanic breccia. Locations and results of the mapping, rock, stream, sediment and soil sampling are unknown.
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.

Year	Owner	Work Area	Description
1967	Mount Washington Copper Co. / Stikine Silver	Kay 1–36 (Core Property)	Electromagnetic (EM 16) and magnetometer geophysical surveys; petrography. Locations and results of the geophysical surveys are unknown.
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. The 1.5 tonne bulk sample from the 22 Zone trenches yielded grams of gold, 7,435 grams of silver, 29 kg of lead and 42.7 kg of zinc with overall grades of 6.2 g/t Au, 4,957 g/t Ag, 1.9% Pb and 2.8% Zn.
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). The mapping project provided a basis of the BSC thesis by Donnelly. The model emphasis changed from precious to base metals and volcanic associated massive sulphide models. Drilling in the #5 Zone intersected two lenses approximately 40cm thick of banded massive sphalerite, galena and pyrite and was underlain by silicified rhyolite cut by stockwork of fine pyrite, sphalerite and galena veins. Significant assays included 8.1% Pb, 5.36% Zn and 1.9 opt Ag over 0.95m however the gold values were weakly anomalous. Results of the EM survey showed no major zones of good conductivity. The magnetic survey showed the area was underlain by rocks of very low magnetic susceptibility with the exception an anomalous area in the SE area of the grid which was interpreted to be an intrusive.
1977–1978			No activity.
1979	May-Ralph Resources Ltd.	22 Zone	A hand-cobbed bulk sample was collected from the #22 zone trenches to produce 1,236 grams of gold, 25,490 grams silver,412 kilograms of lead and 1,008 kilograms of zinc (note the tonnage was not reported).
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)

Year	Owner	Work Area	Description
			Several stream sediment anomalies with coincident lead, zinc and silver existed in a north-east trend in Argillite and Eskay Creek. These anomalies were substantiated by rock sampling.
1983–1984			No activity.
1985	Kerrisdale Resources Ltd.	#5 Zone; 21 Zone; 22 Zone	Mapping: rock and soil sampling. Core drilling (622.10 m). The core drilling in 5 holes above the #21 trenches identified a zone of spotty gold and silver values in the altered felsic volcanic rocks related to the 21A Zone. Soil and rock samples outline a coincident lead, silver and gold anomaly though to be the extension of the #5 zone.
1986	Consolidated Stikine Silver Ltd. (Consolidated Stikine)		No activity.
1987	Consolidated Stikine	#3 Bluff; 5 Zone; 21 Zone and 23 Zone	Stream sediment, soil and rock geochemistry sampling; split and assayed all Kerrisdale core. Low-grade gold values combined with lead-zinc values up to 5% from rock sampling on Red Bluff indicate potential for large tonnage, low-grade gold deposits.
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 intersected stibnite-realgar rich mineralization in the 21A Zone.
1989	Calpine/Consolidate d Stikine	21A Zone; 21B Zone; 22 Zone; GNC	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 (87,888 m). Legal surveys. The geophysical work and geochemical surveys outlined a chargeability and geochemical anomaly tested by hole CA89-109. It intersected 61m averaging 99 g/t Au and 29 g/t Ag in an area that became known at the 109 Zone (Britton et al., 1990). The extensive drill program delineated the 21A Zone and the 21B zone (called the South, Central and North zones at that time). Mapping and grab samples on the Porphyry and Tip Top showings show the GNC project exhibits potential to host a precious metals

Year	Owner	Work Area	Description
			deposit and is underlain by a repetition of the stratigraphic sequence which hosts the Eskay Creek Deposit. The airborne geophysical results showed the GNC project has similarities to the Eskay creek Property. Soil surveys and the IP and magnetometer surveys delineated further anomalies which were recommended to follow up. Dacite mineralization, containing disseminated arsenopyrite returned low gold assays.
1990	Calpine/Consolidated Stikine	21B Zone; 21C Zone; PMP; 109; 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, bulk sample. Underground development began in the 21B Zone. The drill program defined the extents of the 21B and 109 Zones and several satellite zones were discovered including the 21C and PMP Zone.
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. Location and results of the geophysical surveys are unknown.
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. Location and results of the geophysical surveys are unknown.
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 None of the drill holes from GNC intersected significant base or precious metal values. Downhole EM and surface geophysical techniques failed to detect a large conductive body at depth. No further work was recommended.
1994	Homestake	21B Zone; Adrian; Albino Lake	Mapping; rock sampling; borehole FEM geophysical surveys. Core drilling (4,080.95 m). Bonsai Property was optioned. Condemnation drilling with 5 holes at the Albino Lake waste rock disposal site confirmed there was no potential mineralization under or adjacent to the disposal site. Drilling of 5 holes in the Adrian to the north of the 21B Zone returned no significant precious or base metal assays. In addition, a high density of mafic dykes, structural complications and depth

Year	Owner	Work Area	Description
			to permissive stratigraphy downgraded these targets. Borehole geophysics failed to detect any off hole conductors to warrant further drill testing.
1995	Homestake	21B Zone; NEX Bonsai	Mapping, rock sampling, structural study by P. Lewis; Core drilling (3,468.1 m). 1:5000 mapping established the geometry and nature of the Bowser Lake Group/Hazelton Group contact along the Argillite and Coulter Creeks and west of Tom MacKay Lake. Four 1:1000 maps were produced over the Eskay Mine Property and established the structural geometry and lithofacies distribution of rocks in the project area and confirmed the geological setting of the area. Start of production on 21B Zone. The NEX zone was discovered as part of the program to test the north plunging stratigraphy of the 21B stratigraphy. Mapping and drilling on the Bonsai Property on the western margin of the Bowser Basin centred on a rhyolite sill with anomalous As, Sb and Hg values. The drilling failed to return any significant results.
1996	Homestake.	21C Zone; NEX; HW; Adrian; Bonsai	Mapping, rock sampling; trenching. Core drilling (21,280.80 m). Orthophoto Survey. Drilling tested portions of the HW Zone that were inaccessible from underground. The zone was traced over the same lateral extent as the NEX zone but higher up in stratigraphy in the hanging wall mudstones. Drilling of the south of 21B Zone to determine the need to extend underground working failed to intersect zones of mineralized mudstone, but a fairly continuous zone of mineralization in the underlying rhyolite was encountered. Bonsai property was returned to Teuton Resources Corp.

Year	Owner	Work Area	Description
1997	Homestake	21B Zone; 21C Zone; 21E Zone; Adrian; GNC; Mack; Star	Prospecting; silt sampling. Core drilling (16,220.47 m). The majority of the drilling was in the immediate vicinity of the of the underground workings and focused on identifying new zones of proximal mineralization or adding incremental increases to established zones. The 21C Zone was expanded to over 900m in strike length. The Deep Adrian was tested as a faulted offset to the Eskay Creek deposit west of MacKay Creek. Although the contact mudstone was intersected in all cases, there were no zones of significant mineralization or alteration, and no significant assay values were returned. Drilling in the East and West Limbs tested for 21B style mineralization in the contact mudstone, however the mudstone was unmineralized and produced no significant gold or silver values. No further surface drilling was recommended. Drilling in the 21E was successful in defining a small, fault-bounded block due east of the 21B and at the same stratigraphic horizon. The majority of lenses did not have mineralization values above cut-off grade of 15-20 g/t AuEq and were complicated by faulting and mafic dykes. Further work was recommended. Drilling at GNC and Mack claims west of Argillie creek were drilled to test the thickness of Bowser Group and identify any favourable Eskay Creek type horizons below. Although no significant results were encountered, anomalous base metal and pathfinder elements and strong chlorite in the footwall rhyolite were encountered and further work is recommended. Prospecting and mapping at Star Property identified a NE trending belt of basalt, rhyolite and dacite tuff. Rock samples returned low results.
1998	Homestake	21C; 21A; PMP; 5; 23; 22; 28; MacKay Adit; GNC; Mack; SIB Gaps; Star; Coulter	Mapping and prospecting; Whole Rock Geochemical analysis; Test gravity geophysical survey; Core drilling (21,909.63 m). Orthophoto survey. A surface sampling program was undertaken to revaluate zones within the footwall rhyolite to ascertain their potential as possible bulk mineable, low-grade bodies that could be used as mill feed near the MacKay Adit, #22 and #28 Zone . Anomalous precious metal values were predominantly associated with either quartz-sulphide stockwork veins or sericitic shears. None of the zones exhibited mineralization that was consistent enough either in grade or size to warrant further attention. Mapping at the SIB Gap, 4.5m southwest of Eskay Creek Mine confirmed continuation of the stratigraphic sequence however

Year	Owner	Work Area	Description
			based on the mapping and sampling, the surface potential was considered very low. Mapping in the GNC area show no significant changes to the Bowser Basin as a result of the mapping. Results of the test gravity show was that it would be difficult to extract definitive results to use the gravity method successfully in the Eskay Creek Area due to the rough topography and flat-lying nature of the mineralization which tends to give a more muted density contrast. Diamond drilling was successful in infilling the length of the known 21C Rhyolite zone on 25m centers. It was also successful in identifying new zones of high-grade HW and Mudstone over the central portion of the zone. Drilling in the 21A Zone encountered numerous wide, high-grade intercepts both in the rhyolite and mudstone. The intercepts of 4 holes were the thickest and highest-grade hits ever drilled in the zone causing renewed interest in this zone. The two short fences of holes to test the gap between the 21A and 21B zones were largely disappointing with only thin beds of mudstone with minor stibnite. Six holes testing the southern extension of the Pumphouse Zone was generally poor, with only minor zones or erratic base metal mineralization in the rhyoite and underlying dacite. No further work was recommended in this area. Drilling at GNC was successful in determining the Hazelton Group volcanic rocks beneath the Bower sediments but encountered little in the way of alteration or mineralization. The holes in the MACK claims failed to penetrate the rhyolite sequence due to the excessive thickness of the Bowser Group Lake sediments.
1999	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting; structural study; geophysical compilation. Core drilling (17,363.96 m). The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining. Drilling continued at the deep Adrian, East and West Limbs; however, did not encounter any significant mineralization.
2000	Homestake.	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting. Core Drilling (25,893.93 m). The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining.
2001	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb; Felsite	Mapping and prospecting. Core drilling (22,035.48 m).

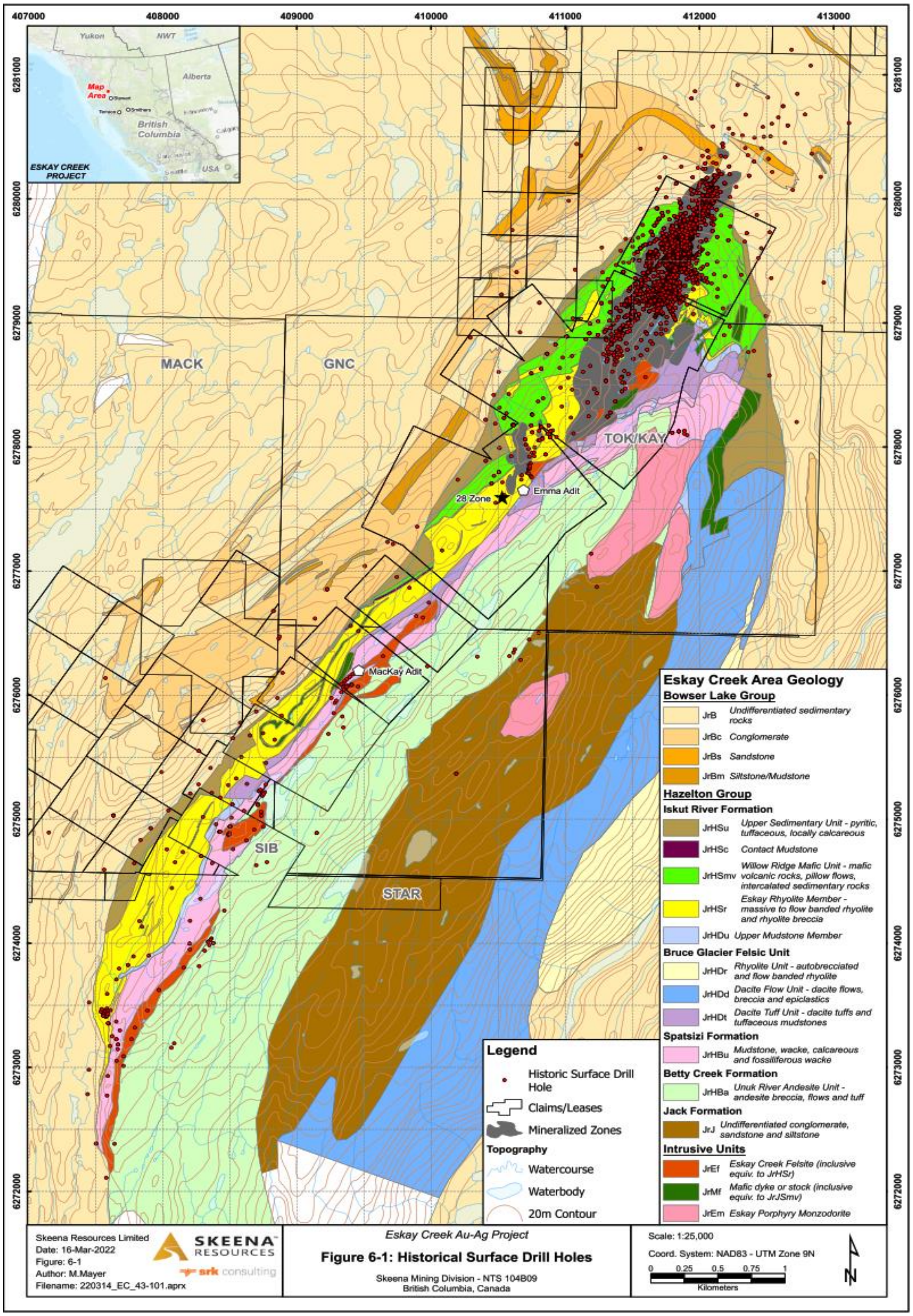
Year	Owner	Work Area	Description
		Bluffs; Sib Gaps; Pillow Basalt Ridge	The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining. The felsites in the souther of the property were drilled for possible low-grade mill feed. Although visually impressive, the program proved unsuccessful with no extensive intervals >1 g/t Au. Drilling of the Adrian intersected the mine stratigraphy at extreme depth. The contact mudstone was nearly non-existent, and no mineralization or alteration of interest was encountered.
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). The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining. No significant assays were returned from Mack or Deep Adrian. 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). The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining. Drilling in the Deep Adrian intersected mine stratigraphy at 992m depth. There were no significant results and no further work was recommended.
2004	Barrick	22 Zone; Deep Adrian; East and 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). The mine sequence was identified in the east and west limbs but no significant grades were identified. A silt and rock anomaly on Ridge block was tested, however drilling intersected only minor intervals on thin low-grade mineralization. In the 22 zone, extensive low-grade mineralization was identified with isolated high-grade pods located in the rhyolite. No significant results were returned from Deep Adrian.
2005	Barrick		Underground definition drilling (16,000 m). The drilling program continued to confirm the geometry, extents and grade of the zones at Eskay Creek for mining.
2006	Barrick		Underground drilling.

Year	Owner	Work Area	Description
2007	Barrick		Underground drilling.
2008	Barrick		Mine closed in April. Reclamation commences.
2009–2016	Barrick		Mine reclaimed. Continuous care and maintenance.
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 survey. Results are discussed in Section 10 of this report 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). Results are discussed in Section 10 of this report. 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 - 277 holes for 43,455.23 m. Resistivity and IP geophysics surveys over Eskay Creek Project. Results are discussed in Section 10 of this Report. Amended terms of the original option agreement; Skeena obtains 100% interest.
2021	Skeena	22; 21A; 21C; 21B; 21E: PMP; HW; NEX; LP; Tom MacKay; 23 Zone; East Dacite; Eskay Porphyry; Albino Lake	Surface core drilling: • Phase 2 - 28 holes for 2,873 m • Phase 3 - 75 holes for 10,727.2 m • Exploration - 67 holes for 12,536.9 m Surface rotary blast drilling: • Albino Lake - 20 holes for 405.7 m Soil and rock sampling, rift-basin reconstruction and targeting project – core re-logging phase. Results are discussed in Section 10 of this report. 2021 Pre-Feasibility Study (PFS) is released.

6.2 Historical Surface Drilling

Drilling was first conducted in the early 1930s by the MacKay Syndicate. Between 1934 and 2004, 1661 surface core drill holes totalling 377,983.26 m were drilled. Figure 6-1 is surface map showing the drill hole locations of all historical drilling.

Figure 6-1: Historical Surface Drilling



6.2.1 Surface Diamond Drill Methods

Limited details are available regarding drilling contractors and drilling procedures specific to each campaign prior to 1995. Table 6-2 summarizes the known drill contractors and methods.

Table 6-2: Drill Contractors and Methods

Year	Contractor	Rig Type	Core Size and Core Diameter
1996	Advanced Drilling of Vancouver	Boyles 56	
1995-1997	Hy-Tech Drilling of Smithers, B.C. (Hy-Tech)	JKS-300	BQTK (40.7 mm) NQTK (50.6 mm) NQ2 (50.6 mm)
1998	Hy-Tech	JKS-300 F-15	BQTK (40.7 mm)
2002	Hy-Tech	Tech-5000	NQ2 (50.6 mm)
2004	Hy-Tech	JKS-300 F-15	BQTK (40.7 mm)

6.2.2 Surface Diamond Drill Procedures

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 within 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 into an AutoCAD 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.

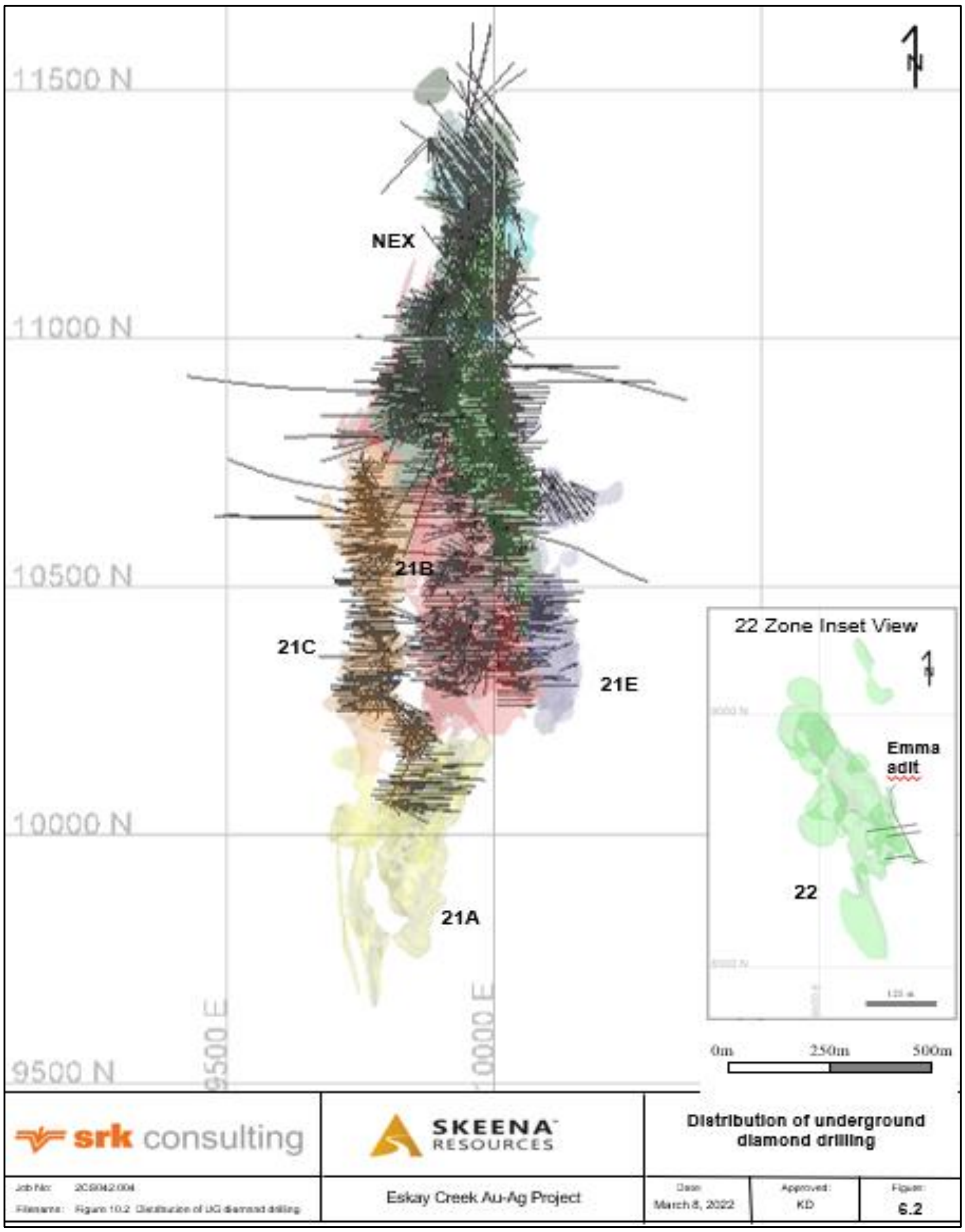
6.3 Underground Drilling

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

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.

A total of 6,149 underground drill holes were drilled totalling 317,242.31 m. Figure 6-2 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 6-2: Underground Drill Hole Location Plan



6.4 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.

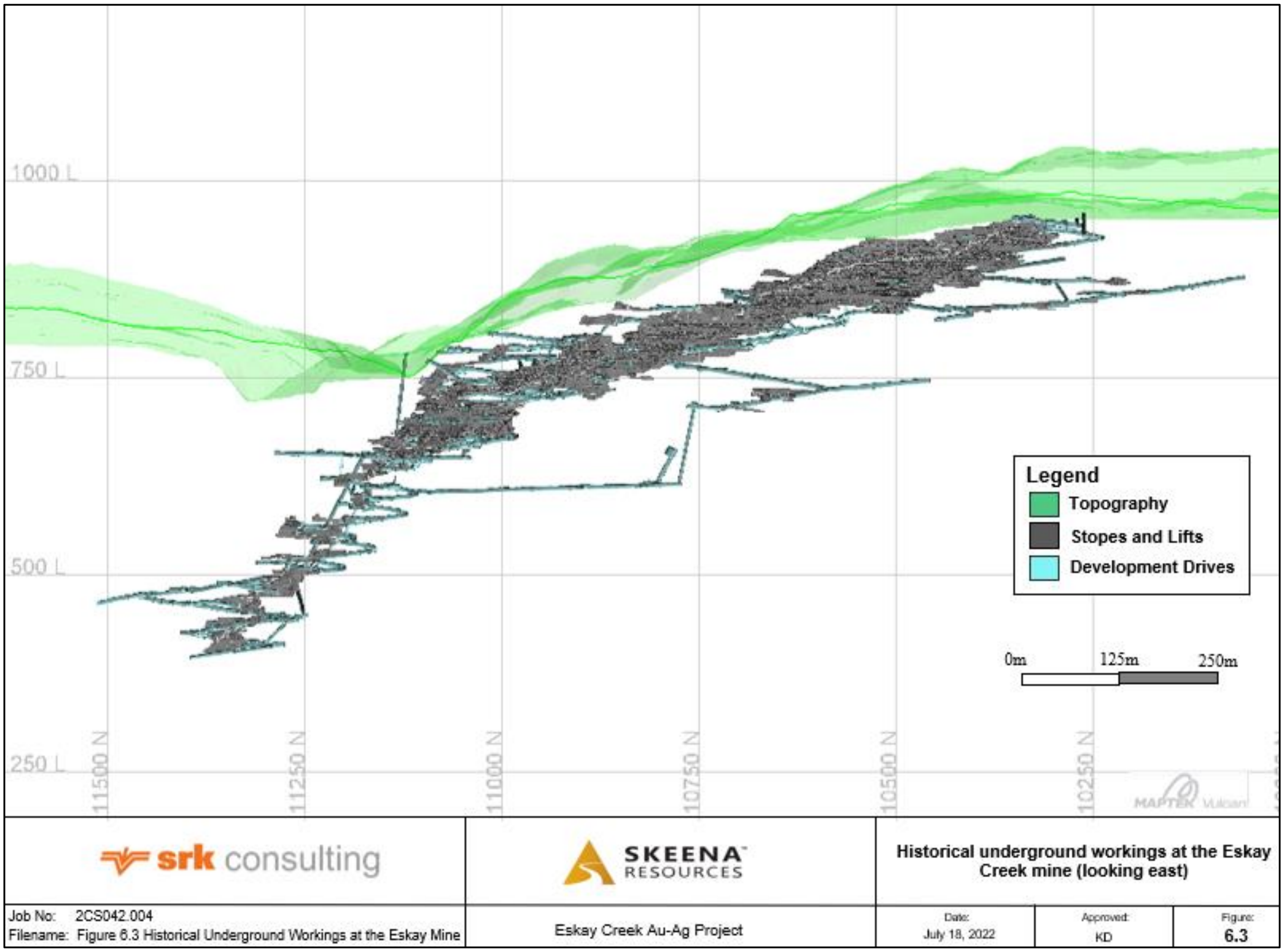
The Eskay Creek process plant began commercial production on 1 January 1998 at a 150 t/d rate. Production rates were incrementally increased from 1999 to 2004.

The Eskay Creek mine production is summarized in Table 6-3. Underground workings at the mine (stopes, lifts and development drives) are shown in Figure 6-3.

Table 6-3: 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 6-3: Historical Underground Workings at the Eskay Creek Mine (Looking East)

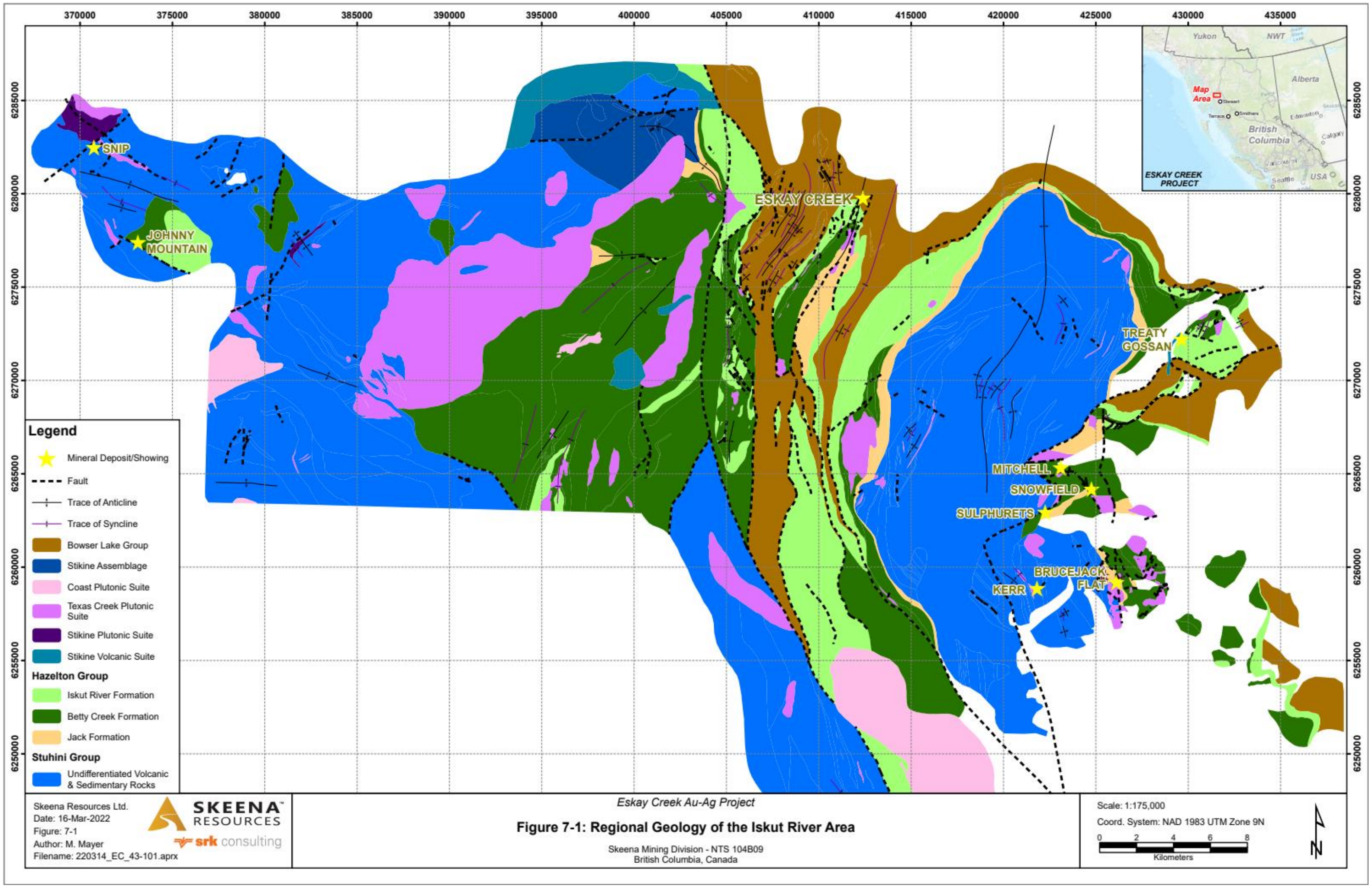


7 GEOLOGICAL SETTING AND MINERALIZATION

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



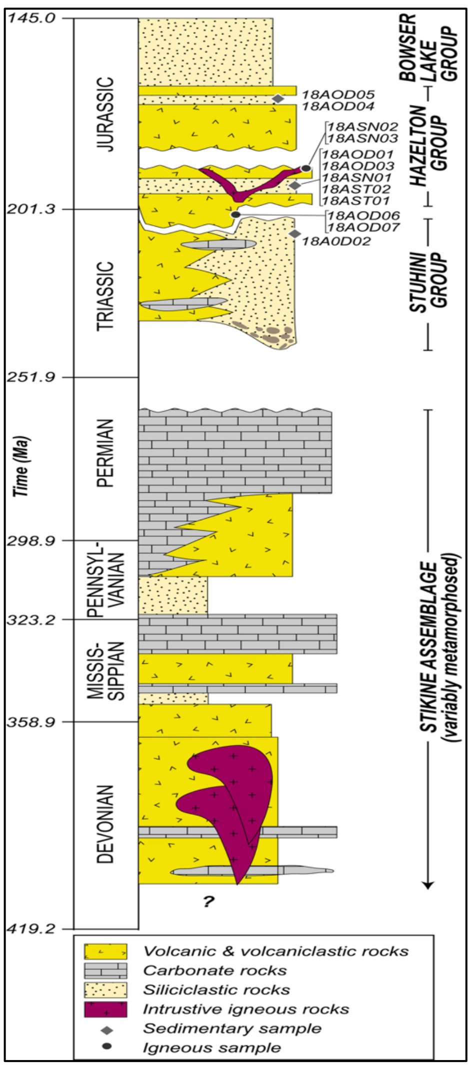
Note: Eskay Creek and Snip deposits are held by Skeena. Other mines and deposits shown are owned by third parties.

Table 7-1: Regional Stratigraphy of the Iskut River Region

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

Source: after Anderson, 1989 and Nelson et al., 2018.

Figure 7-2: Schematic Stratigraphic Column for Stikinia



Note: Figure prepared by George, S.W, 2021.

Table 7-2: Iskut River Region Plutonic Rock Suite

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/extrusive 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)

Note: table prepared by MDRU, 1992.

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.

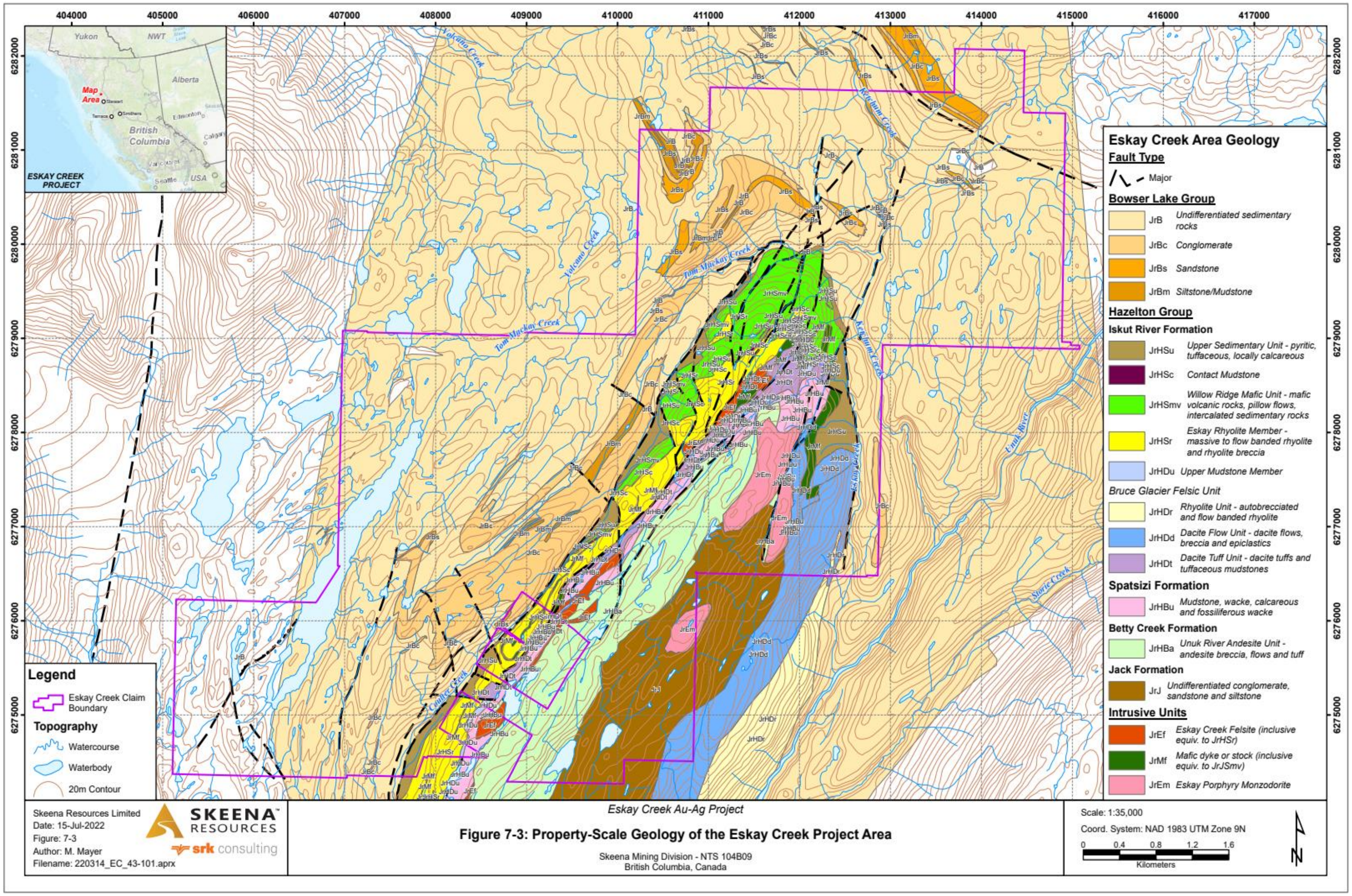
Given the important relationship of the Hazelton Group to mineral deposits throughout the area, there have been many local mapping campaigns through the years, completed by different workers and at different scales. The resulting stratigraphic framework, although detailed in parts, contained numerous 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. A regional geology map is shown in Figure 7-3.

Table 7-3: Stratigraphic framework for the Hazelton Group in the Eskay 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 kilometre thick successions of interlayered basalt, rhyolite, and sedimentary rocks	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	171-175 Ma

Formation	Lithologies	Sub-units	Age
	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.	Eskay Creek in the Contact Mudstone); similar thin, discontinuous lenses enclosed within volcanics occur elsewhere in the Iskut River Formation	
		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

Figure 7-3: Property-Scale Geology of the Eskay Creek Project Area



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 7-4).

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 7-5.

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 7-4: Local Geology and Mineralized Zones of the Eskay Creek Project

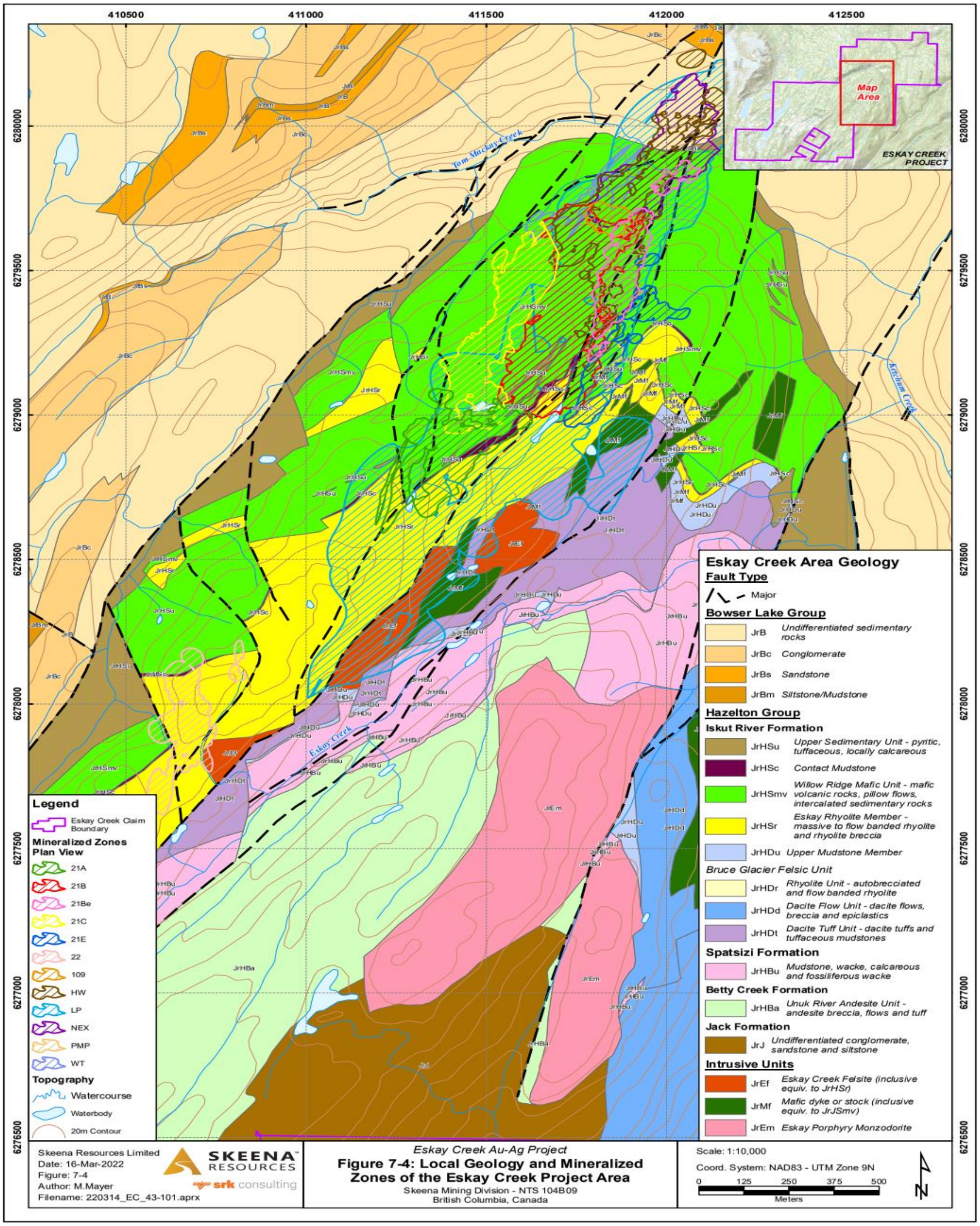
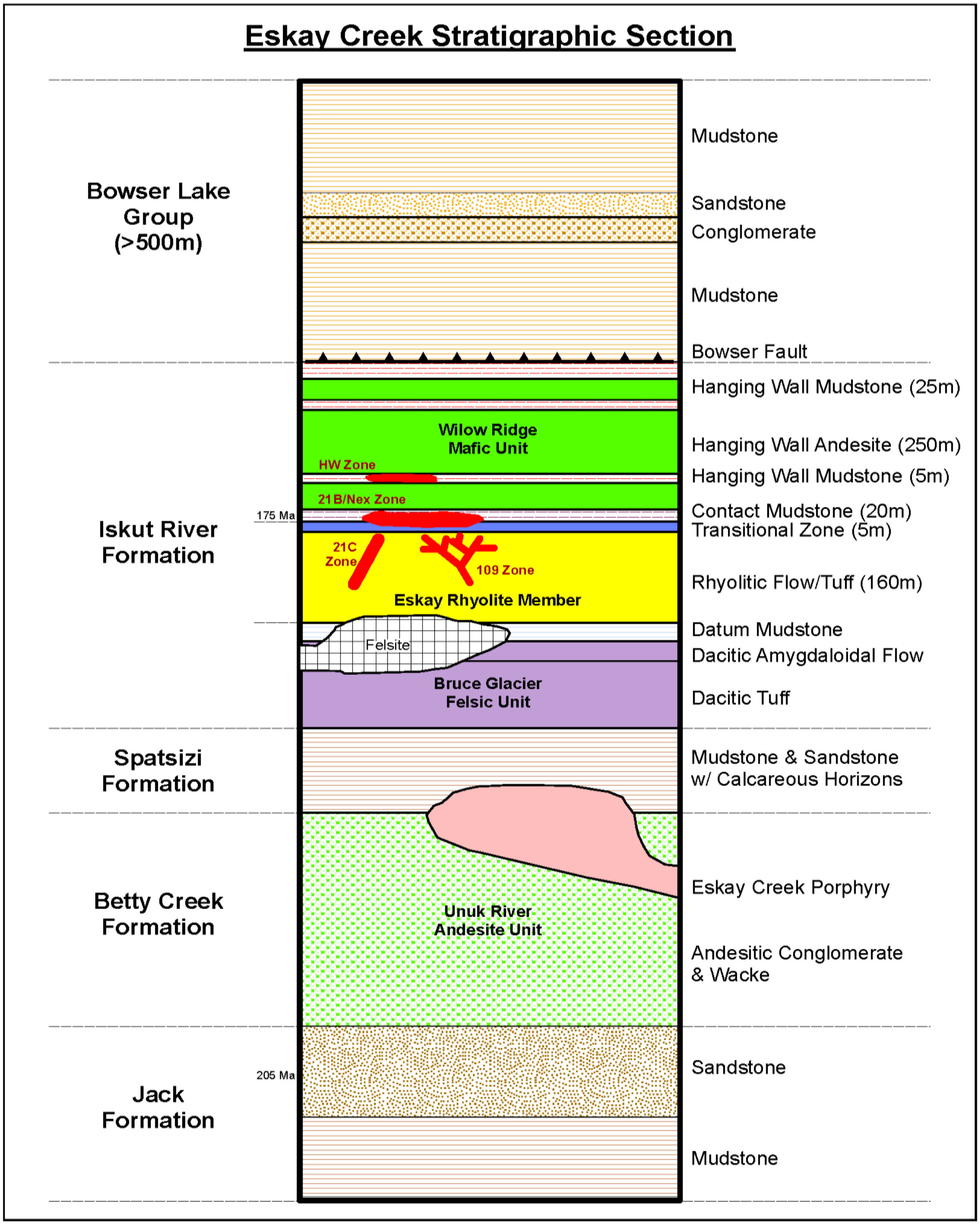


Figure 7-5: Eskay Creek Stratigraphic Section (modified after Gale et al., 2004)



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, which was mined out by previous operators, is 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 7-1). Mineralized zones are 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 Figure 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). Disseminated stibnite, arsenopyrite, tetrahedrite, and veinlets of pyrite, sphalerite, galena, tetrahedrite ± chalcopyrite.	At the base of the Contact Mudstone 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. Localized zones of cryptic, disseminated, precious metal-bearing mineralization.	Within the Contact Mudstone 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

Zone	Associated Elements	Characteristics	Stratigraphic Position
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

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 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 and is bound to the east by the Pumphouse fault. Stratiform-style, mudstone-hosted mineralization is approximately 300 m long by 200m wide and 10 m in thickness. 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.

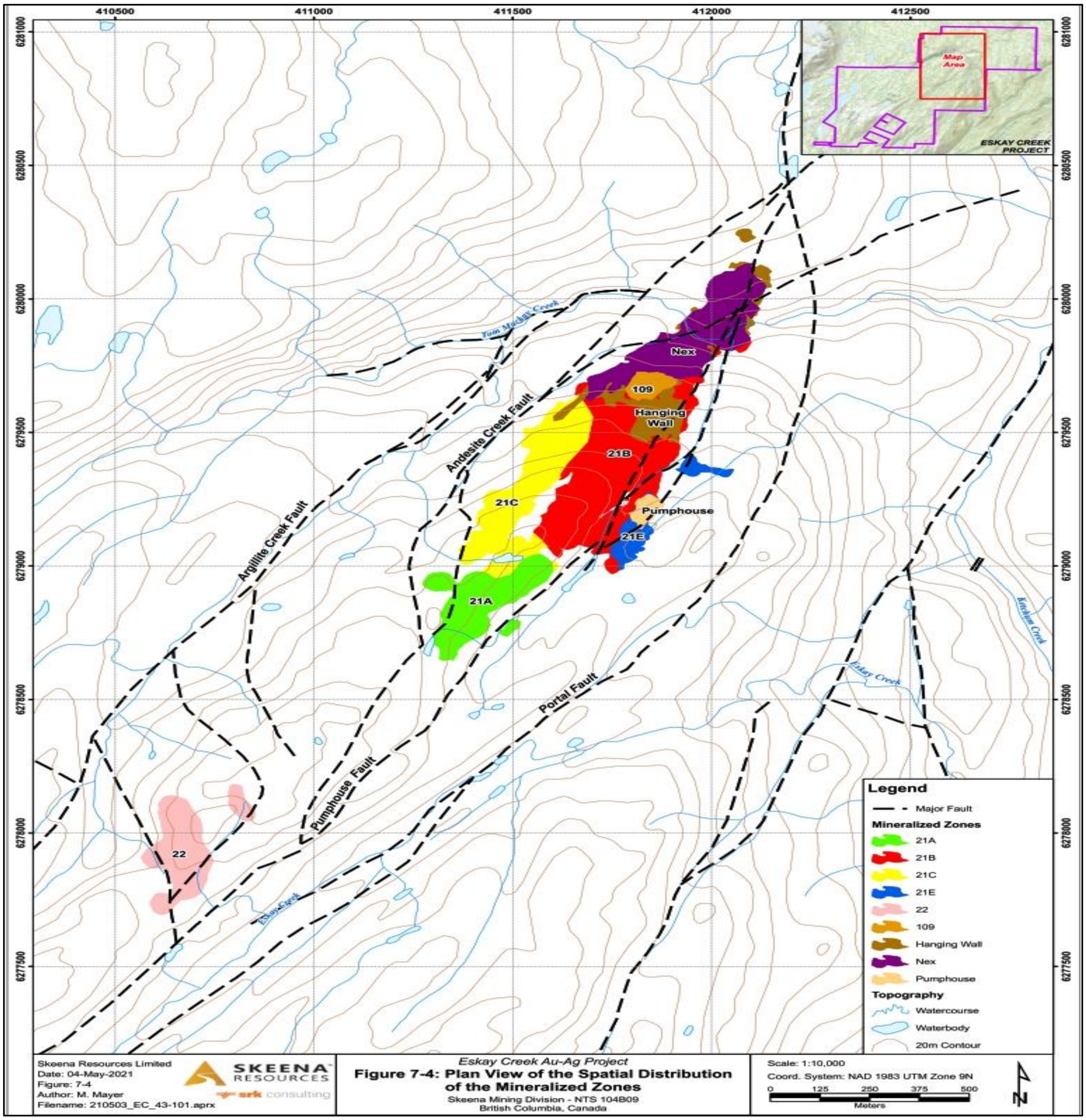
The mudstone is underlain by a discontinuous zone of intense Magnesium chlorite alteration and stockwork veining in the Eskay Rhyolite Member. Disseminated stibnite, arsenopyrite, and tetrahedrite also occur in the immediate footwall of the sulphide lens within the intensely sericitized rhyolite. Cinnabar 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.

7.3.2 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 mineralized zone, 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 7-6: Plan View of the Spatial Distribution of the Mineralization Zones



7.3.3 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 and is roughly 675 m long by 130 m wide. 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.

7.3.4 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. The zone is approximately 530 m long by 115 m wide with an average thickness of 10 m.

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.

7.3.5 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. The zone is roughly 1100 m long by 140 m wide. The thicknesses of the individual beds range from a few metres up to 20 m. 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.

7.3.6 North Extension Zone (NEX) Zone

The ~800 m long by 170m wide 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.

7.3.7 21E Zone

The 21E Zone sits on the eastern most block and is approximately 500 m long by 90 m wide. Locally, mudstone interbeds within the Hanging Wall Andesite host fine-grained to massive and locally clastic sulphides and sulphosalts. Individual beds range from a few metres up to 15 m thick. 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.

7.3.8 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. This zone spans the length of the deposit and is approximately 2600 m long by 500 m wide. 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.

7.4 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).

7.4.1 PMP Zone

The PMP Zone is a discordant zone of diffuse vein and disseminated sulphide mineralization 200 m long x 75 m wide that is 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.

7.4.2 109 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 in a zone that is 140 m long, 120 m wide and 30 to 80 m thick. 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.

7.4.3 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 in a zone roughly 350 m long by 80 m thick. Fine-grained arsenopyrite and stibnite are occasionally observed. Higher vein densities generally indicate better gold grades.

7.4.4 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 in a zone that is 450 m long by 100 m wide. Individual zones average 5 m in thickness but are locally up to 20 m thick.

7.5 Prospects/Exploration Targets

Exploration potential is discussed in Section 9.5.

7.6 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

8.1 Deposit Model

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.

Table 8-1 summarizes the key features of each deposit type.

Features that would classify Eskay Creek as a VMS deposit (Roth et al., 1999) include:

- It formed on the seafloor in an active volcanic environment with a rhyolite footwall and basalt hanging wall.
- There is a chlorite-sericite alteration in the footwall, and sulphide formation within a mudstone unit at the seafloor interface.

Unlike many VMS deposits, Eskay Creek has 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 sulfosalts, are more typical of an epithermal environment with low formation temperatures.

Features that would classify Eskay Creek as a subaqueous hot spring gold-silver deposit (Alldrick, 1995) include:

- broad hydrothermal systems marked by widespread sericite-pyrite alteration
- evidence of a volcanic crater or caldera setting
- accumulations of felsic volcanic strata.

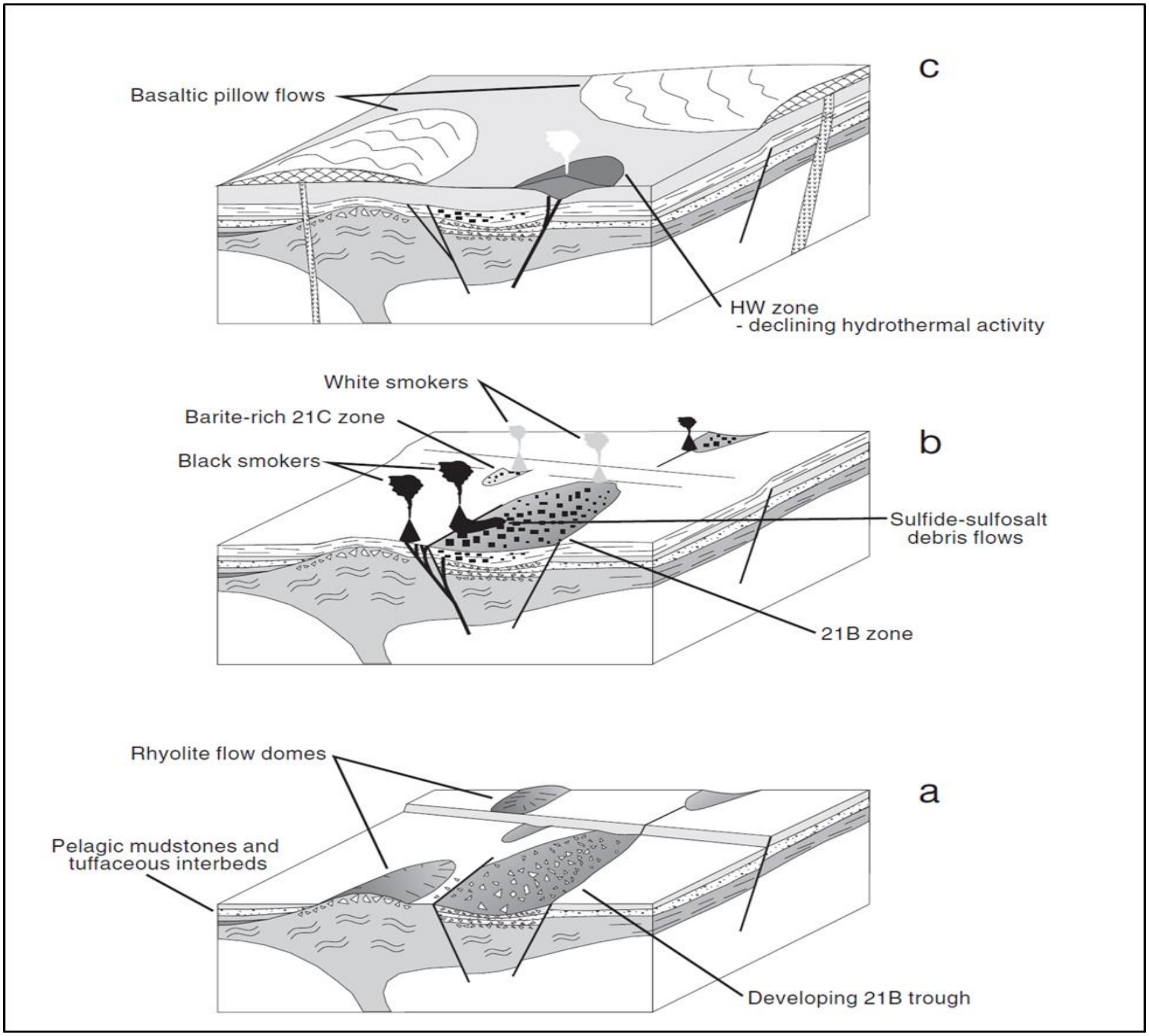
Roth et al., (1999) developed a deposit genesis model for the 21 Zones, that included the following phases:

- 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 Zone) 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).

Table 8-1: Deposit Type Features

	VMS	Hot Spring Au–Ag
Tectonic setting:	Oceanic extensional environments, such back-arc basins, oceanic ridges close to continental margins, or rift basins in the early stages of continental separation. Form at, or near, the seafloor through the focused discharge of hot, metal-rich hydrothermal fluids.	Active volcanic arcs (both oceanic island arcs and continental margin arcs) are likely setting.
Host/Associated rock types:	Terrigenous clastic rocks associated with marine volcanic rocks and sometimes carbonate rocks; these may overlie platformal carbonate or clastic rocks. Typically, a concordant sheet of massive sulfides up to a few metres thick and up to kilometres in strike length and down-dip; can be stacked lenses.	Mineralization hosted by intermediate to felsic flows and tuffs and minor intercalated sedimentary rocks. Pillow lavas, coarse epiclastic debris flows, and assorted subvolcanic feeder dikes are all part of the local stratigraphic package.
Deposit form:	Deposits typically comprise thin sheets of massive to well-layered pyrrhotite, chalcopyrite, sphalerite, pyrite and minor galena within interlayered, terrigenous clastic rocks and calcalkaline basaltic to andesitic tuffs and flows. There is typically a mound-shaped to tabular, stratabound body composed principally of massive (>40%) sulphide, quartz and subordinate phyllosilicates, and iron oxide minerals and altered silicate wall-rock. These stratabound bodies are typically underlain by discordant to semiconcordant stockwork veins and disseminated sulphides. The stockwork vein systems, or “pipes”, are enveloped in distinctive alteration halos, which may extend into the hanging-wall strata above the VMS deposit.	Highly variable. Footwall stockwork or stringer-style vein networks. Large, textureless massive sulphide pods, finely laminated stratiform sulphide layers and lenses, reworked clastic sulphide sedimentary beds, and epithermal-style breccia veins with large vugs, coarse sulphides and chalcedonic silica. All types may coexist in a single deposit.
Ore mineralogy (principal and subordinate):	Pyrite, pyrrhotite, chalcopyrite, sphalerite, cobaltite, magnetite, galena, bornite, tetrahedrite, cubanite, stannite, molybdenite, arsenopyrite, marcasite	Sphalerite, tetrahedrite, boulangerite, bournonite, native gold, native silver, amalgam, galena, chalcopyrite, enargite, pyrite, stibnite, realgar, arsenopyrite orpiment; metallic arsenic, Hg-wurtzite, cinnabar, aktashite, unnamed Ag-Pb-As-S minerals, jordanite, wurtzite, krennerite, coloradoite, marcasite, magnetite, scorodite, jarosite, limonite, anglesite, native sulphur.
Gangue mineralogy (principal and subordinate):	Quartz, calcite, ankerite, siderite, albite, tourmaline, graphite, biotite	Magnesian chlorite, muscovite (sericite), chalcedonic silica, amorphous silica, calcite, dolomite, pyrobitumen, gypsum, barite, potassium feldspar, alunite with minor carbon, graphite, halite and cristobalite.

Figure 8-1: Genetic Model



Note: Figure from Roth et al., 1999.

8.2 QP Comments on "Item 8: Deposit Types"

The QP is of the opinion that exploration programs that use either a VMS and/or a hot-spring deposit model in this project area are applicable for gold and silver mineralization targeting.

9 EXPLORATION

9.1 2018 – 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 – Mapping and Grab Sampling Program

9.2.1 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 22 grab samples were collected from outcrops on the Property for whole rock analysis and analysed by multi-acid multi-element inductively coupled plasma (ICP) and 44 structural measurements were taken (Figure 9-1). Samples that were the most altered or mineralized were collected.

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.

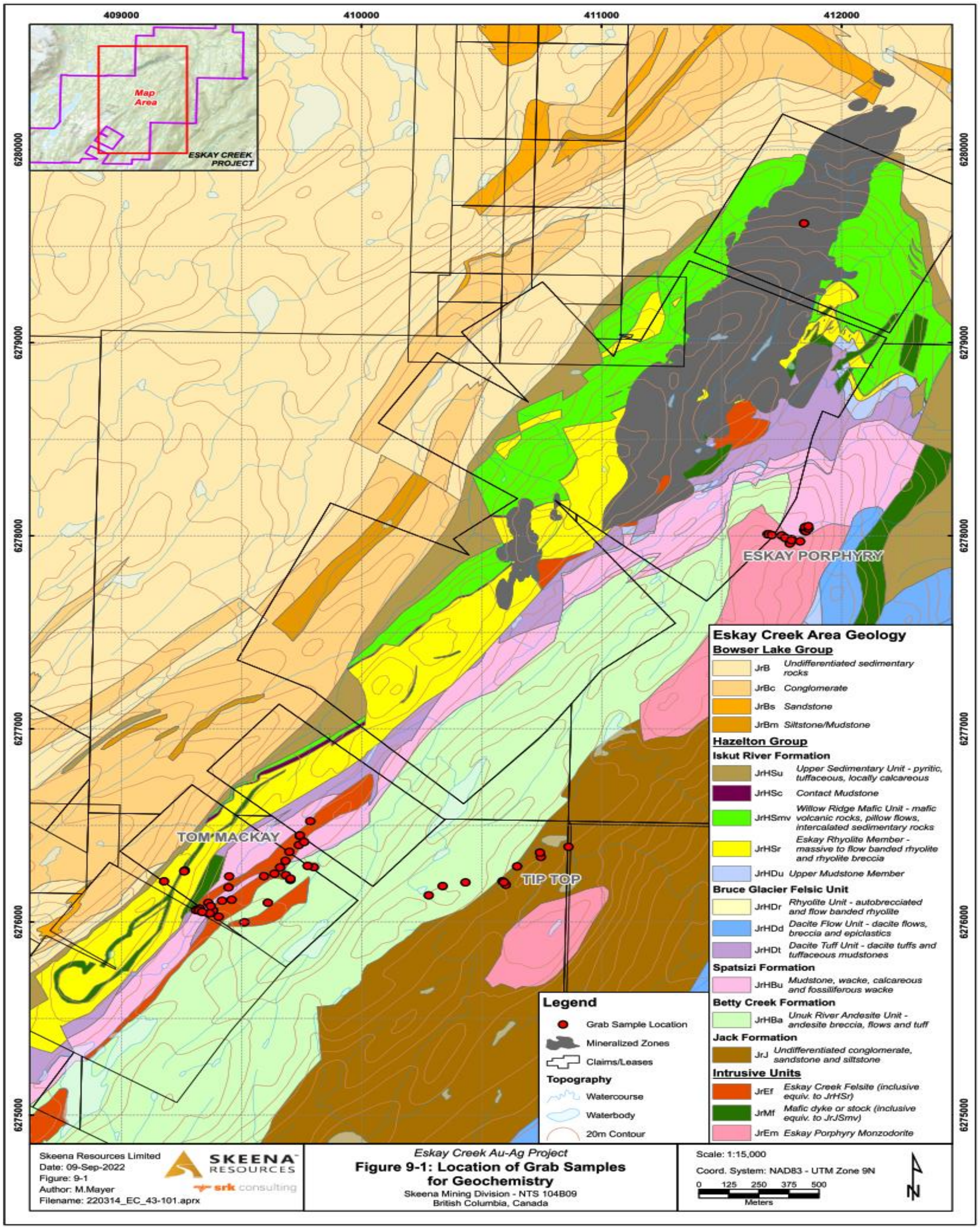
Sampling returned a number of anomalous gold and silver grades. Following the favourable geochemical assays, it was recommended to drill to the northeast of the Tom MacKay area.

9.2.2 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. 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 from both prospects. The location of the Eskay Porphyry and Tip top samples are shown in Figure 9-1 below.

Figure 9-1: Location of Grab Samples collected on the Property during 2019

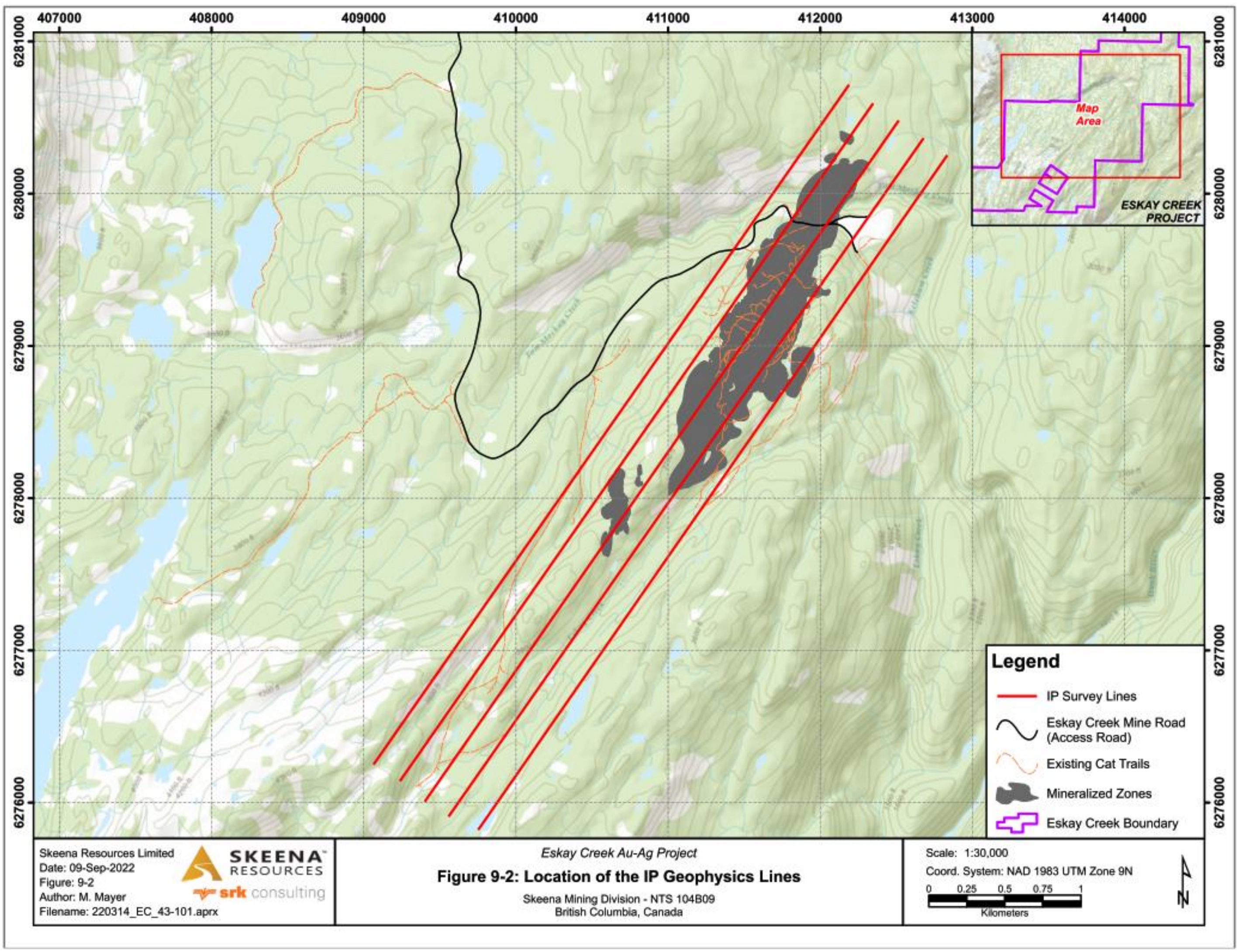


9.3 2020 - Geophysics

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 over the axis of the Eskay Creek anticline from the Bowser Basin south to the Tom MacKay Zones 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 (Figure 9-2).

Figure 9-2: Location of IP Geophysics Lines



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. 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.

Dias Airborne Limited of Saskatoon, SK, flew an airborne magnetic gradiometry survey over 5 days in 2020 using the QMAG full tensor magnetic gradiometer (FTMG) system. Forty-metre line spacing for a total of approximately 1060 line kilometres were completed, which included 965 km of survey lines and 95 km of tie lines.

The incorporation of the airborne magnetic datasets into the larger litho-structural model highlights the structural framework of the Eskay Creek Basin which can be applied to other regional targets associated with the Faults. This includes the along strike extension of the Pumphouse and Argillite Creek faults, as well as other parallel structures which may have potential.

9.4 2021 - Exploration

9.4.1 Eskay Rift-Basin Reconstruction and Targeting Project

From April 19 through May 3, 2021, relogging of diamond drill core was undertaken to establish an informal stratigraphy for strata that host the Eskay deposits. Relogging of drill core and resulting graphic logs were completed for 26 representative drill holes totalling approximately 7,439 m. Eighty-nine samples were collected for whole rock analysis to characterize lithofacies and alteration types.

9.4.1.1 Methods and Procedures

Information for each whole rock sample collected was recorded in GeoSpark (a MS Access based database application). It includes the Hole ID, the sample identification number, depth of sample, and the rock and mineralization details. A photo of each sample was taken along with the corresponding sample tag. The sample tag with a unique sample number was inserted in the sample bag and the sample location was marked with flagging tape in the drill core box.

9.4.1.2 Analysis

The whole rock samples that were collected during relogging have been prepared and analysed at ALS laboratory facilities located in Kamloops and Vancouver, BC, respectively. Samples were prepared using the PREP-31 method, which involves crushing the sample to 2 mm and then splitting off and pulverizing up to 250 grams to 75 microns. The resulting pulp was analysed by the CCP-PKG05 Complete Characterization package which includes the ME-MS61™ method. It involves dissolving 0.5 grams of material in a hot Aqua Regia solution and determining the concentration of 61 elements of the resulting analyte by the ICP-MS technique as well as measuring carbon and sulphur by combustion analysis.

9.4.1.3 Results

The production of stratigraphic logs, created during relogging across predefined cross sections through the Eskay Creek deposit and beyond, served as a basis to model various editions of basin evolution reconstruction. The trace and mobile elements of the whole rock analysis helped to confirm the stratigraphy on the property by understanding the true composition of certain rocks units that were otherwise obscured by intense alteration. It was also used to determine that the Contact Mudstones and HW mudstone are geochemically identical and therefore appear to represent a continuous deposition environment that was subsequently disjointed and separated by andesite flows and sills of the HW Andesite package. Although the data was reviewed to assist with alteration types, it was not particularly useful given the wide alteration footprint of the deposit.

9.4.2 Geochemical Soil Sampling Program

Inherited soils data collected by previous operators demonstrated strong correlations between Au-Ag mineralization exposed at surface and B-Horizon Au soil anomalies. Unfortunately, the historical soils coverage was discontinuous across the property, particularly along the Eastern Limb of the Eskay Anticline. In addition, the data collected by previous operators is poorly documented, generally lacks any quality assurance/quality control checks and is therefore of uncertain quality.

During the summer of 2021, Skeena collected 4,367 soil samples. The soil sampling program covered the majority of the lease boundaries, apart from areas defined as Bowser Basin geological units. The sampling entailed 116-line kilometres and was completed on a systemic 25-m x 100-m grid. Given the surficial footprint criteria for a near surface bulk tonnage target, these soil grid parameters permitted adequate coverage to detect an economic target.

9.4.2.1 Methods and Procedures

All soil sampling traverses were pre-planned, with pre-specified sampling intervals, typically 25 m spacing on northwest-southeast trending grid lines spaced 100 m apart. Field technicians navigated to the sample site using handheld GPS units. The soil sampler arrived at each sample site, identified the most appropriate location to collect the sample and laid out a sheet of plastic (12"x20" ore bag). The soil sample was collected using a hand auger at a depth of between 20 cm and 110 cm. Samplers strived to consistently collect B-Horizon sample material. Where necessary (e.g., rocky or frozen ground) a prospector's pick ('mattock') or planting spade was used to assist in obtaining the sample material. The soil was laid out on the sheet of plastic in the order it was recovered from the sample hole. The sampler placed the necessary amount of soil (400-500 grams) from the B-Horizon materials into a kraft sample bag. The bag was labelled with the unique sample identification number. The data for each sample was recorded using a database application on a GPS capable smartphone/tablet; collected information included location, soil colour, soil horizon, sample depth, and sample quality as well as any other relevant information. GPS coordinates were entered into a handheld GPS device as a secondary backup in case of smartphone/tablet failure. Standard reference materials were inserted into the sample stream at an approximate frequency of 1:20.

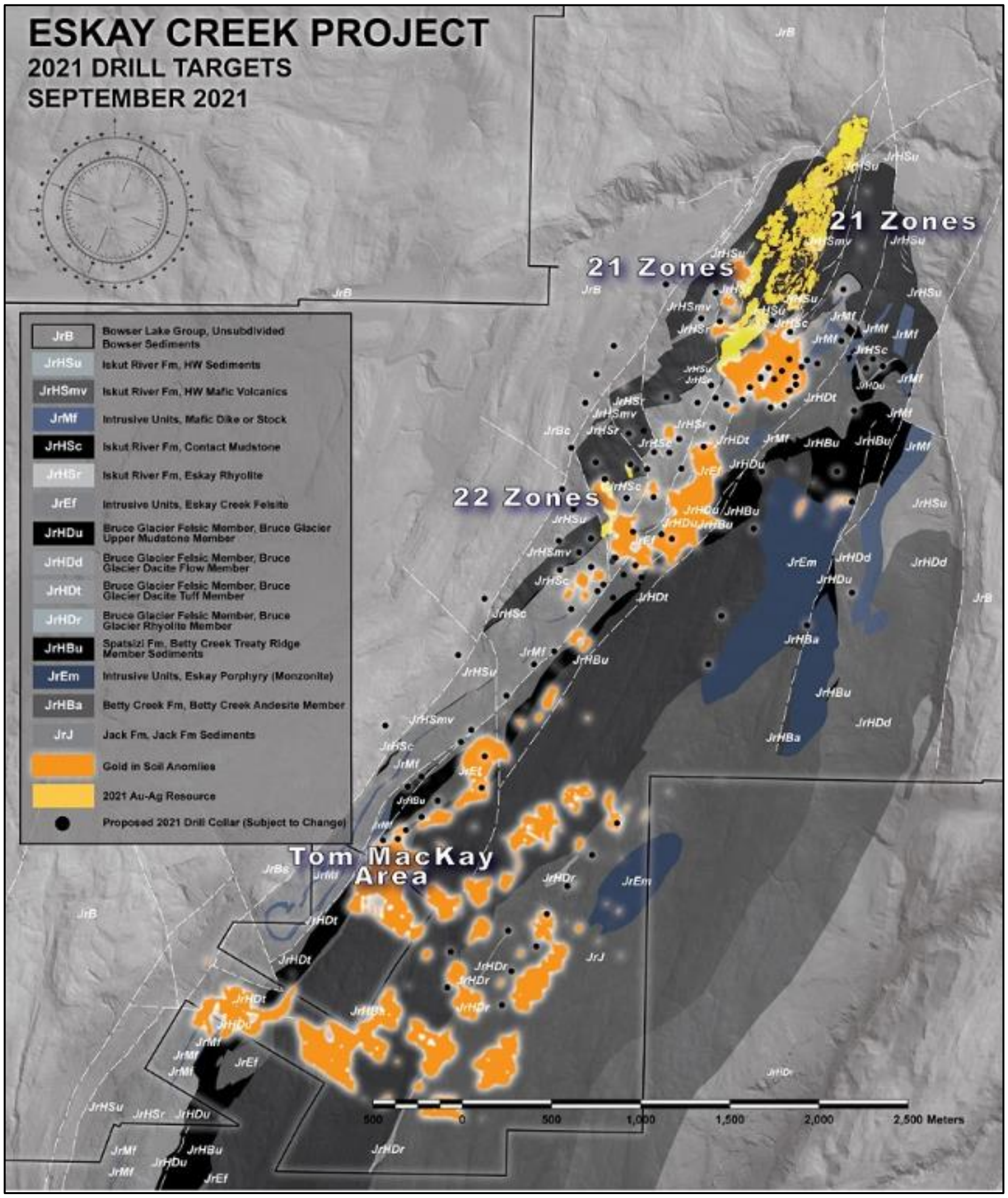
9.4.2.2 Analysis

All soil samples were prepared and analysed at ALS Laboratories in North Vancouver, BC. Samples were prepared using the PREP-41 method which involves drying the samples at 60 degrees Celsius and sieved such that up to 100 grams of material passes 180 microns (80 mesh). The samples were then analysed by the ICP-MS technique following an Aqua reqia digestion of the material for 41 elements.

9.4.2.3 Results

Anomalous values in Au and Ag, along with pathfinder elements such as Sb and As, highlight the remaining exploration potential along the Eskay trend, south of the deposit. Spatially, anomalous Pb and Zn values were also found to correlate relatively well with anomalous Au and Ag, while Cu anomalies had a weaker spatial correlation with the other precious and base metal anomalous values. Figure 9-3 shows the resulting gold in soil anomalies on the Property.

Figure 9-3: Gold in Soil Anomalies on the Property

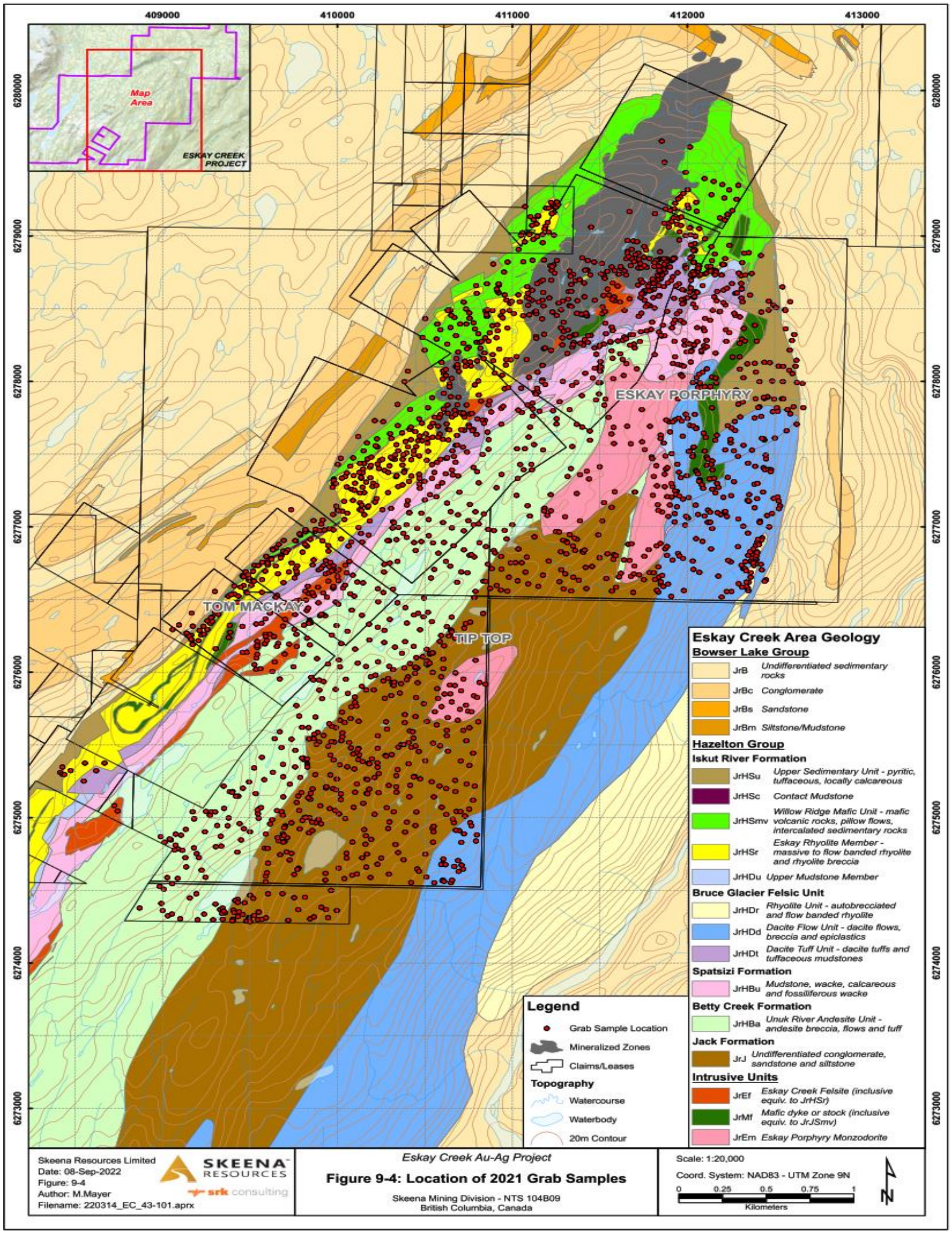


Note: Figure prepared by Skeena, 2021.

9.4.3 Regional Mapping and Grab Sampling

From June through August 2021, Skeena collected 2,296 rock samples throughout the property, apart from areas defined as the Bowser Basin geological unit, to assist in the characterization of the lithofacies and alteration types. In addition, geological field mapping and prospecting activities were completed over the entirety of the property with additional focus on geochemical anomalies reported in historical soil grids, grab rock samples and diamond drilling. The samples were collected to ensure coverage at outcrops that had no previous data recorded nearby. The most mineralized or altered parts of the outcrops were sampled. The location of all the grab samples collected during 2021 is shown in Figure 9-4.

Figure 9-4: Location of Grab sample collected during 2021



9.4.3.1 Methods and Procedures

Information for each station or sample was recorded in ArcGIS Field Maps (a database application) using a Samsung Active Tab 3 tablet. Data collected includes the coordinates, the sample identification number, structural measurements and the rock and mineralization details. A photo of the sample was also collected along with the corresponding sample tag. The sample tag with a unique sample number was inserted in the sample bag and the sample location was marked with flagging tape. Collected data was used to create, confirm, or update surface geologic maps and to enhance the data density on the project for later interpretation and drill hole targeting.

9.4.3.2 Analysis

All mapping and prospecting samples were prepared and analysed at ALS laboratory facilities located in Kamloops and Vancouver, BC respectively. Samples were prepared using the PREP-31 method which involves crushing the sample to 2 mm and then splitting off and pulverizing up to 250 grams to 75 microns. The resulting pulp was analysed by the ME-MS41 method, which involves dissolving 0.5 grams of material in a hot Aqua Regia solution and determining the concentration of 41 elements of the resulting analyte by the ICP-MS technique. Gold was analysed by Fire Assay which involves fusing 30 grams of the 75-micron material in a lead flux to form a doré bead. The bead is then dissolved in acid and the gold quantity in the sample is determined by Atomic Absorption Spectroscopy.

9.4.3.3 Results

Grab rock samples, as well as geochemical soil samples, highlighted a 2 km long section of the Pumphouse Fault corridor south of the 21A Zone along the Eskay trend with limited to no historical drill testing; new data showed this corridor to be prospective. Subsequent diamond drilling during 2021 in this area led to the discovery and expansion of the new 23 Zone. Geological mapping, along with trace and major element interpretation of the collected samples, has led to the reclassification of several lithologic units on the existing map as well as the redrawing of certain lithologic boundaries on the maps inherited from previous operators of the Eskay Creek Project.

9.5 Exploration Potential

There is remaining exploration potential in the Eskay Creek deposit. Several areas have been selected for drill targeting based on the geochemical soil sampling, and grab rock sampling campaigns along the Eskay Trend, which remain to be tested as shown on Figure 9-3.

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, 21A Zone, 23 Zone, 21C Zone and in the mudstones of 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 style mineralization.

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

9.6 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.

10 DRILLING

10.1 Introduction

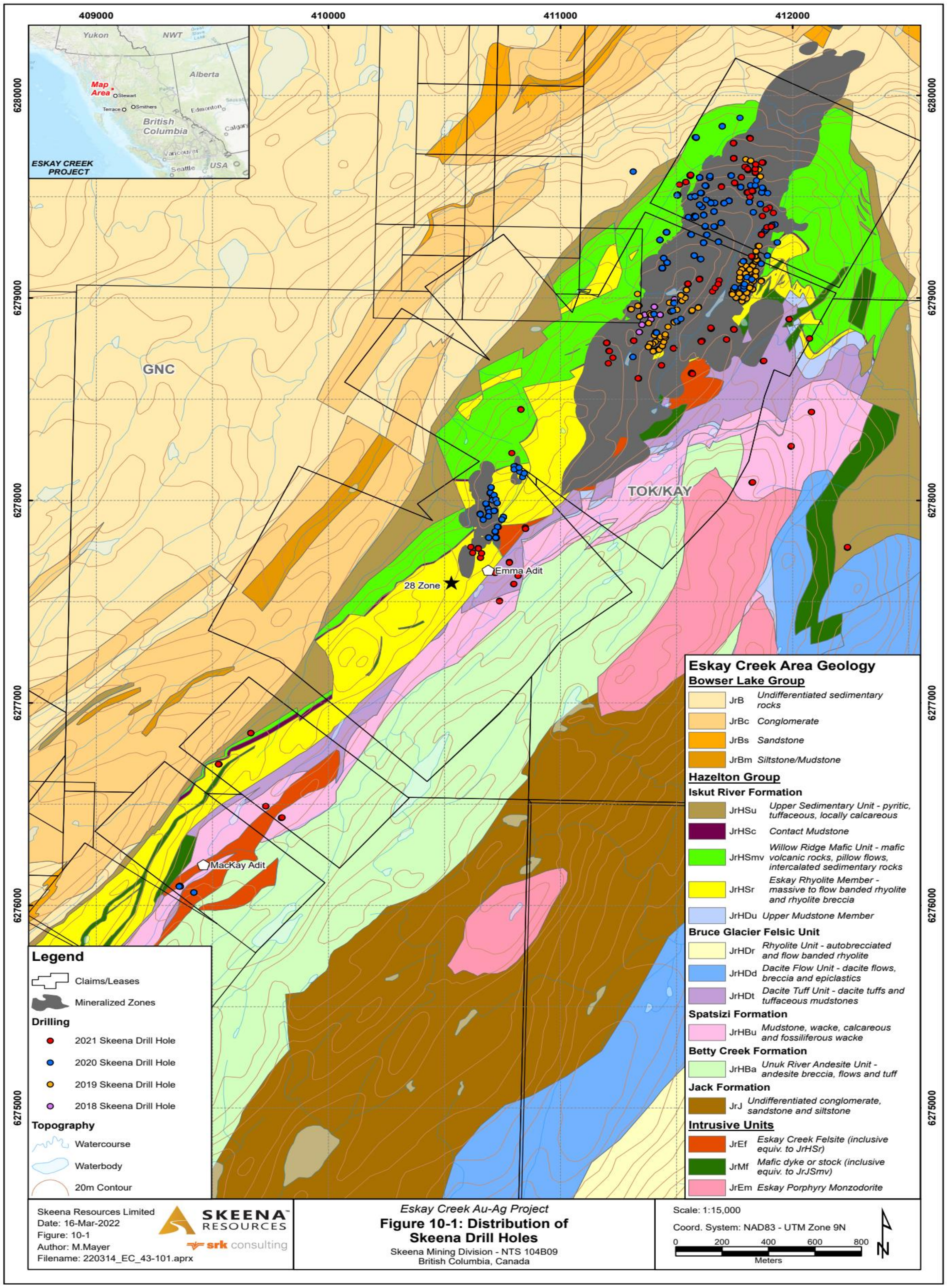
Surface drilling has been carried out by multiple operators, with the first drilling on the property by Unuk Gold in 1934. Data collected prior to Skeena's project interest is discussed in Section 6.

Since 2018 to the end of 2021 Skeena has drilled 913 surface drill holes totalling 128,362.89 m. Table 10-1 summarizes the surface drilling Skeena has completed on the Eskay Creek Project from 2018 to 2021. Figure 10-1 shows the location of the surface holes by year. Albino Lake holes are not shown.

Table 10-1: Drill Summary Table of Drilling Undertaken by Skeena

Period of Work	Area of Work	Number of Holes	DDH #'s	Metres Drilled
2018	21A / 21C / 22 Zones	46	SK-18-001 to SK-18-043; SK-18-048 to SK-18-051	7,737.45
2019	21A / 21B / 21E / HW Zones	203	SK-19-044 to SK-19-047; ~SK-29-052 to SK-19-247	14,091.87
2020	21A / 21B / 21C / 21E / HW / PMP / WT / MAC / 22 Zones	473	~ SK-20-248 to SK-20-788	79,992.79
2021	22 / 21A / 21C / 21B / 21E / PMP / HW / NEX / Albino Lake / Tom MacKay / 23 Zone / East Dacite / Eskay Porphyry	191	~ SK-21-789 to SK-21-997	26,610.78

Figure 10-1: Surface Drill Hole Location Plan of Skeena Drilling by Year



10.2 Logging Procedures

10.2.1 Diamond Drill Core

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 wax pencil. 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.

10.2.2 Rotary Air Blast Drilling

All technical tasks were completed by geologists and supervised geological technicians employed by Skeena. The Rotary Air Blast (RAB) drill works by channeling compressed air through 5-foot (1.52 m) single-wall drill rods to a pneumatic hammer attached to a semi-permeable bit, which acts as a jackhammer. The air forces rock chips and fines (the sample) through openings at the edge of the bit, where it then travels to surface along the sides of the rod string and is transferred from the borehole to a cyclone module by a sample hose. The sample drops out of the bottom into clean 5-gallon pails. Each sample comprises one 5-foot run. Lithology was recorded by the on-site drill rig geologist for the Phase 2 drilling program. The sample was then tipped out of the pails into white rice bags. Sample bags were labelled with a unique sample identification and sealed with a cable tie for shipment to the core logging facility, where it was left to dry. At the core logging facility, the white rice bags were opened and a sample for assay analysis was collected by scooping the homogenous material from the bottom to the top of each bag. These samples were placed in clear poly sample bags with an assay tag. The sample numbers were written on the bag, and the bag was then sealed for shipment to the lab. The remaining material was placed in newly labelled rice bags are retained at the core logging facility at North Spoils.

The Mineral Resource Estimate only considered diamond drill core assays; however, RAB assays were used for an in-house estimate of Albino Lake tailing's material. RAB assays are less accurate than diamond drill assays due to the increase in contamination, reduction in variability and loss in recovery with air blasted rock.

10.2.3 Skeena Drilling Methods and Procedures

10.2.3.1 2018 Diamond Drilling

From August to November 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 were 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 was initially stored at both the Eskay Creek Mine site carpentry shop and McLymont Creek staging area but is now stored at North Spoils.

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.

10.2.3.2 2019 Diamond Drilling

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. Drill core was stored at McLymont Creek Staging area.

10.2.3.3 2020 Diamond Drilling

From February to October 2020, with a hiatus from March to July due to Covid 19 restrictions, Skeena completed 197 diamond 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 to December 2020, Skeena drilled 277 holes for 43,455.23 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, HW and 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 helicopter-supported drilling, ITL used a DrillCo rig, which was used only for helicopter-supported 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 were 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 was initially stored at McLymont Creek Staging area but is now stored at North Spoils.

10.2.3.4 2021 Diamond Drilling

Four contractors were used throughout the year for the diamond drilling: Tahltech, Konaleen, Omineca Diamond Drilling Ltd (Omeneca), and Helm Diamond Drilling Ltd. (Helm).

Tahltech used both a Hydracore 2000 and Zinex A5 drill for the helicopter-supported and skid drilling, Konaleen used a Zinex A5 drill, Helm used both a Zinex A5 and Hydracore 2000 drill and Omineca used a Zinex drill. Helicopter drill moves and daily drill support was provided by Silver King using Eurocopter AS350 B2 helicopters. For heavier items, a B3 helicopter was used. The majority of the drill core was NQ in size, with the exception of 15 holes which were drilled by HQ, and 2 holes collared using HQ. HQ drilling was used for the metallurgy holes to acquire a larger sample size, and for holes located in fault zones.

From January 1 to January 11, 2021, the remaining 28 holes for 2,873 m of the Phase 2 program targeting the zones inside the 20 m buffer zone was completed.

From January 11 to February 6, 2021, Skeena drilled 7 holes for 712.5 m from their Phase 3 drill program targeting the PMP and HW Zones and 18 holes for 4,195 m of near mine exploration targets. Following a short hiatus, drilling resumed in early May to drill a further 50 holes for 5,819.4 m to complete the Phase 3 drilling in early August. The purpose of the Phase 3 drilling was to support upgrade of the mineral resource confidence categories in the HW, 22, 21B, 21A Fault Zones and to collect metallurgical samples for the NEX Zone. The near-mine exploration targets included the southern 21B Zone, the hanging-wall sediments along the Pumphouse Fault in the 21E Zone, the hanging-wall sediments adjacent to the A+ndesite Creek Fault in the 21C Zone, and the Eastern Dacites.

Exploration drilling began in late August and continued to mid-November drilling 67 holes for 12,536.9 m. The regional exploration targets tested the Tom MacKay area, Eskay Porphyry, Pumphouse corridor, northern 21A Fault Zone, 22 Zone extension, 23 Zone and the Eastern Limb targets.

Drill core was logged and sampled at core logging facilities located at the McLymont Creek staging area from January to October. In October, core was logged and sampled at the North Spoils. All drill core was moved from the McLymont Creek staging area and is now stored at the North Spoils.

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.

10.2.3.5 2021 Rotary Air Blast Drilling

The Albino Lake drilling was completed by Tahltech, using a Rotary Air Blast Fraste Multidrill ML (MDML). The purpose of the drilling was to test the Au-Ag grade potential in the Albino Waste Facility (AWF) on 50 m centers. Phase 1 drilling commenced on the frozen ice surface on March 13th, however due to deterioration in ice conditions the program was suspended on March 22nd after drilling a total of 8 holes for 193.8 m. On October 1st the Phase 2 drilling began utilizing a floating barge on the water surface. Due to the onset of winter, the program was suspended on October 12th after drilling

12 holes for 211.84 m. Drill hole collars were initially located using handheld GPS units and were surveyed at the end of the drill program using a Trimble DGPS. All drill holes were short, less than 25 m, and vertical. Therefore, no downhole surveys were taken.

10.3 Recovery

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

10.4 Sample Lengths/True Thickness

The sample lengths were determined by the geologist during logging. 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.

The average sample lengths for the RAB holes were 1.52 m. All holes were vertical and true widths are considered to be 100% of the interval length.

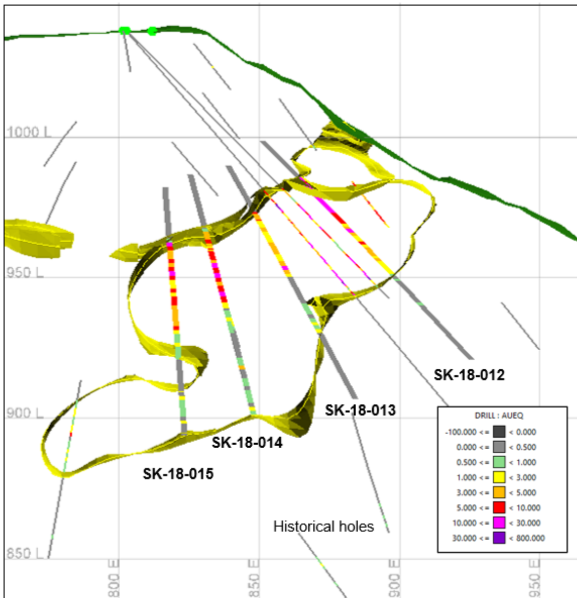
10.5 Results of 2018 Drilling

Drilling during 2018 was designed to infill areas with low drill density in the 22, 21A and 21C Zones to sufficient drill spacing to allow for future economic analyses and to collect fresh material for an upcoming metallurgical characterization and testing program as no historical drill core remains for any zones at Eskay Creek.

Drilling in the 21A and 21C Zones spatially correlated and confirmed the continuity of the mineralization in the historical drilling. The 21A Zone demonstrated excellent geological and grade continuity as shown in Figure 10-2.

The planned drilling in the 21A and 21C Zones was completed in its entirety; however only 30% of the planned infill drilling was completed in the 22 Zone due to the onset of winter weather conditions. The drilling program was designed to infill and upgrade the Inferred resources in the 22 Zone by increasing drill density to 20 m intercept spacing. Only six of the planned 20 holes were drilled. A representative section with significant intervals is shown in Figure 10-3 below.

Figure 10-2: 2018 Drilling in the 21A Zone



Significant Intervals

Hole	From	To	Length	Au	Ag	AuEq
SK-18-012	**73.5**	**118.5**	**45**	**9.07**	**76**	**10.09**
incl.	73.5	75	1.5	12.35	20	12.62
and	75	75.69	0.69	10.8	9	10.92
and	76.5	77.5	1	20.6	14	20.79
and	77.5	78.5	1	14.8	33	15.24
and	78.5	80	1.5	20.8	71	21.75
and	80	81	1	30.2	108	31.64
and	81	82.5	1.5	10.95	254	14.34
and	82.5	84	1.5	11.95	563	19.46
and	84	85.5	1.5	23.4	247	26.69
and	85.5	87	1.5	18.8	83	19.91
and	87	88.5	1.5	14.8	36	15.28
and	88.5	89.05	0.55	17.3	17	17.53
and	89.05	89.6	0.55	23.1	20	23.37
and	89.6	91	1.4	13.7	43	14.27
and	99	100	1	13.55	3	13.58
and	100	101.27	1.27	10.65	7	10.74
SK-18-013	**74**	**100.5**	**26.5**	**2.89**	**63**	**3.73**
SK-18-014	**74**	**102.3**	**28.3**	**5.39**	**139**	**7.24**
incl.	86.3	87.5	1.2	8.2	518	15.11
and	87.5	89	1.5	8.42	14	8.61
and	89	89.6	0.6	8.01	741	17.89
and	89.6	91	1.4	15.15	215	18.02
and	95	96	1	5.89	239	9.08
and	96	96.5	0.5	9.07	232	12.16
and	96.5	98	1.5	9.24	232	12.33
SK-18-015	**75.6**	**108.5**	**32.9**	**3.74**	**95**	**5**
incl.	75.6	77	1.4	11.15	15	11.35
and	86	87.5	1.5	5.5	387	10.66
and	87.5	89	1.5	3.02	423	8.66

Note: All values are un-cut and drill intercepts are core length

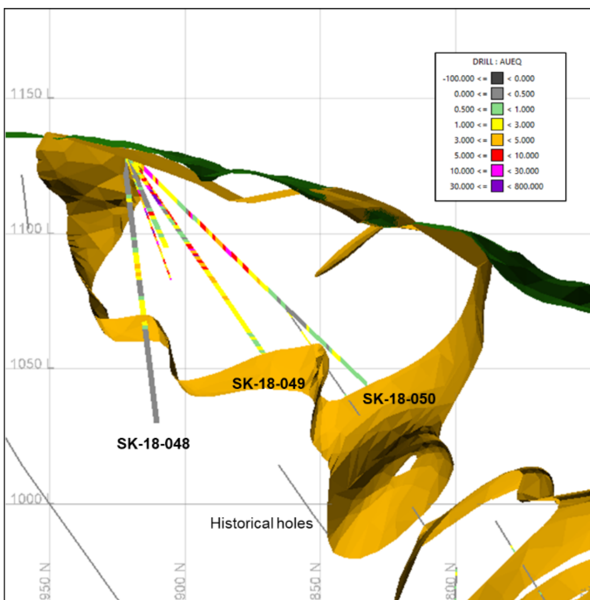
srk consulting | SKEENA RESOURCES | **2018 Drilling in the 21A Zone Section 10005 N +/- 10m**

Job No: 2CS042.004
Filename: Figure 10.2 2018 Drilling in the 21A Zone

Eskay Creek Au-Ag Project

Date: February 12, 2022 | Approved: KD | Figure: **10.2**

Figure 10-3: 2018 Drilling in the 22 Zone



Significant Intervals

Hole	From	To	Length	Au	Ag	AuEq
SK-18-048	**39**	**68**	**29**	**1.23**	**12**	**1.39**
SK-18-049	**2.85**	**95**	**92.15**	**4.07**	**122**	**5.71**
incl.	16.5	18	1.5	10.95	36	11.43
and	18	18.7	0.7	20.8	113	22.31
and	18.7	19.5	0.8	32.1	257	35.53
and	19.5	21	1.5	3.05	154	5.1
and	21	22.5	1.5	28	177	30.36
and	22.5	24	1.5	78.1	1840	102.63
and	24	25.1	1.1	33.6	633	42.04
and	25.1	26	0.9	8.3	193	10.87
and	45	46	1	6.94	465	13.14
and	48	48.65	0.65	9.86	2220	39.46
SK-18-050	**3.9**	**77**	**73.1**	**2.61**	**138**	**4.44**
incl.	11	12.5	1.5	10.4	19	10.65
and	12.5	14	1.5	13.5	95	14.77
and	14	15.5	1.5	21.9	81	22.98
and	18	18.6	0.6	17.3	209	20.09
and	34	35	1	2.46	494	9.05
and	35	36	1	4.6	795	15.2
and	53.5	55	1.5	2.53	831	13.61
and	55	56.5	1.5	4.33	745	14.26

Notes: All values are un-cut and drill intercepts are core length

srk consulting | SKEENA RESOURCES | **2018 Drilling in the 22 Zone Section 9545 E +/- 20m**

Job No: 2CS042.004
Filename: Figure 10.3: 2018 Drilling in the 22 Zone

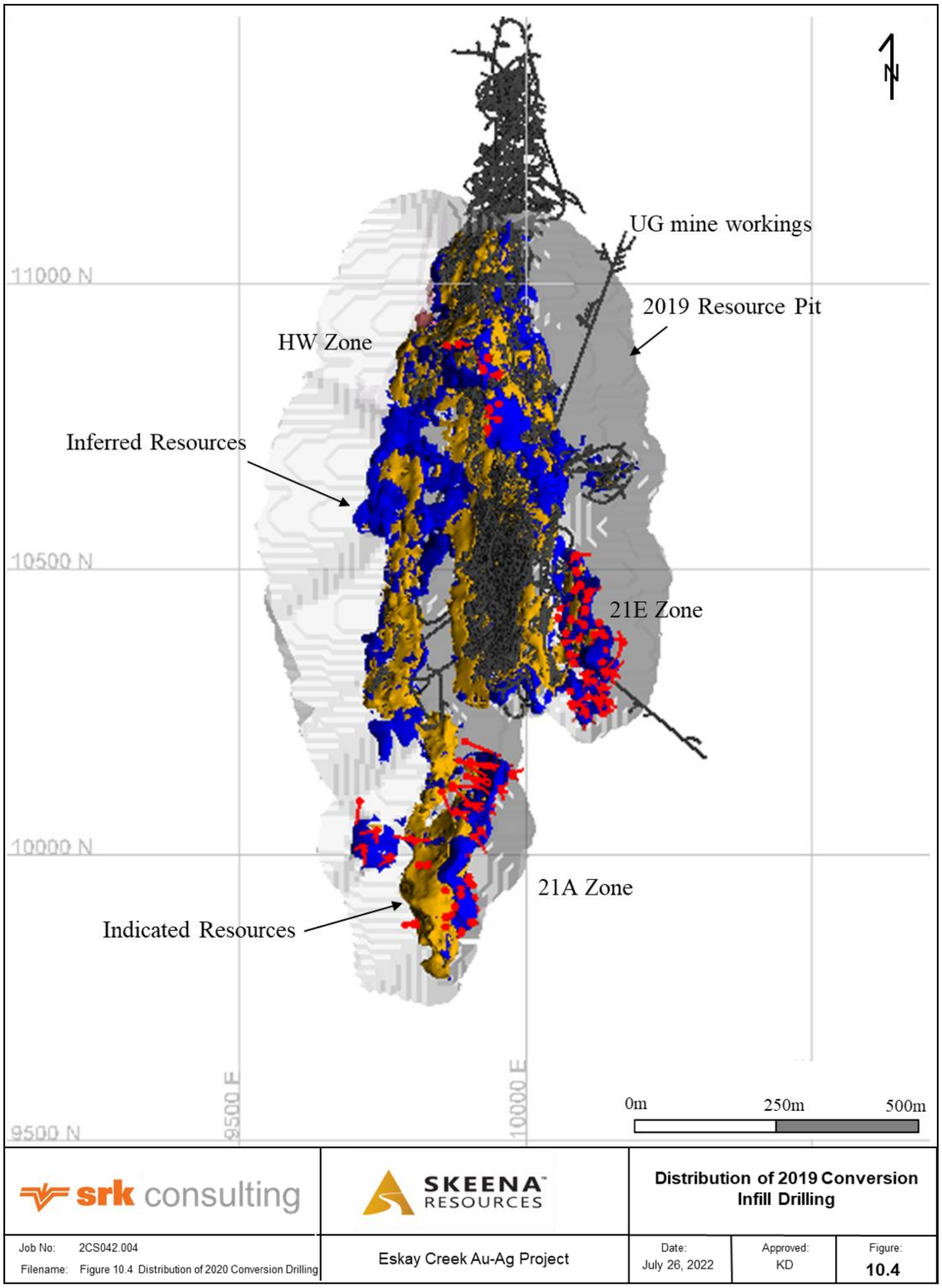
Eskay Creek Au-Ag Project

Date: July 12, 2022 | Approved: KD | Figure: **10.3**

10.6 Results of the 2019 Drilling

Following the release of the updated Mineral Resource Estimate model in February 2019, Skeena embarked on a major infill drilling program in to reduce the drill hole spacing to support potential upgrades of blocks classified as Inferred to higher confidence categories in the 21A, 21E and HW Zones, and test for potential mineralization extents (Figure 10-4).

Figure 10-4: Distribution of 2019 Conversion Infill Drilling. Drill Holes are shown in Red

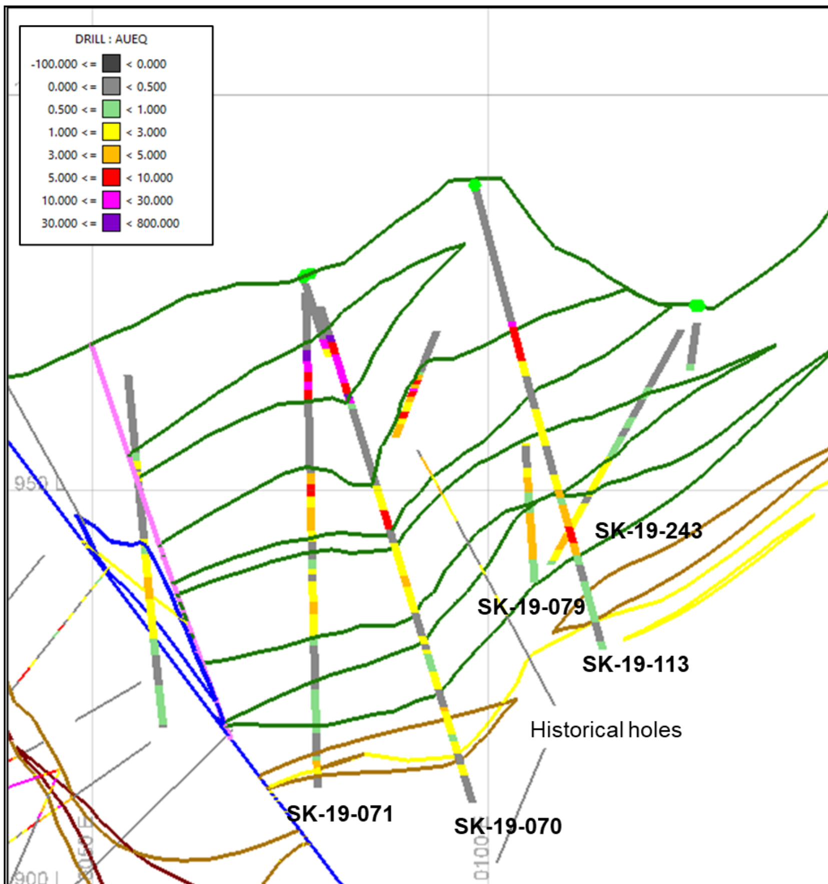


The program was successful in reducing the drill spacing and converting most of the Inferred resources to either Indicated or Measured for blocks in the 21A, 21E and HW. Figure 10-5 is a representative section showing mineralization in the hanging-wall sediments of the 21E Zone.

The western lobe of the 21A Zone was interpreted to be exclusively hosted in the tabular Contact Mudstone. The drill campaign showed the presence of a previously-unknown hydrothermal vent associated with gold–silver mineralization in this location.

Gold–silver mineralization, hosted in the Lower Mudstone, was encountered 100 m stratigraphically below the Contact Mudstone in the 21A Zone. This horizon had not been previously expected to host higher-grade mineralized zones (Figure 10-6) and was further tested in 2020 drilling.

Figure 10-5: 2019 Infill Drilling in the 21E Zone



Significant Intervals

Hole	From	To	Length	Au	Ag	AuEq
SK-19-070	**8.2**	**18**	**9.8**	**9.14**	**348**	**13.78**
incl.	8.2	9	0.8	20	1945	45.93
and	10.66	11.5	0.84	10.6	25	10.93
and	11.5	12.4	0.9	11.8	22	12.09
and	12.4	13.1	0.7	1.97	714	11.49
and	13.1	13.9	0.8	8.11	248	11.42
and	13.9	14.55	0.65	12.5	25	12.83
and	16	16.5	0.5	21.4	706	30.81
and	16.5	17.34	0.84	8.48	422	14.11
SK-19-070	28.08	66.5	38.42	1.3	23	1.61
SK-19-071	**9.47**	**16**	**6.53**	**9.58**	**569**	**17.18**
incl.	9.47	10	0.53	52	67	52.89
and	10	11	1	2.91	2120	31.18
and	11	11.8	0.8	4.08	484	10.53
and	11.8	12.4	0.6	14.3	38	14.81
and	12.4	13	0.6	8.01	15	8.21
and	13	13.6	0.6	9.68	23	9.99
and	13.6	14.55	0.95	7.31	647	15.94
and	14.55	16	1.45	1.87	355	6.6
SK-19-071	25.11	47	21.89	1.91	63	2.74
SK-19-113	**18**	**25.13**	**7.13**	**3.03**	**179**	**5.42**
incl.	18	18.67	0.67	7.74	175	10.07
SK-19-113	29.35	34.35	5	0.84	49	1.49
SK-19-113	38.13	50	11.87	2.84	26	3.19
SK-19-079	**24.31**	**52**	**27.69**	**1.46**	**41**	**2.01**
SK-19-243	**16.42**	**37.5**	**21.08**	**1.33**	**33**	**1.77**

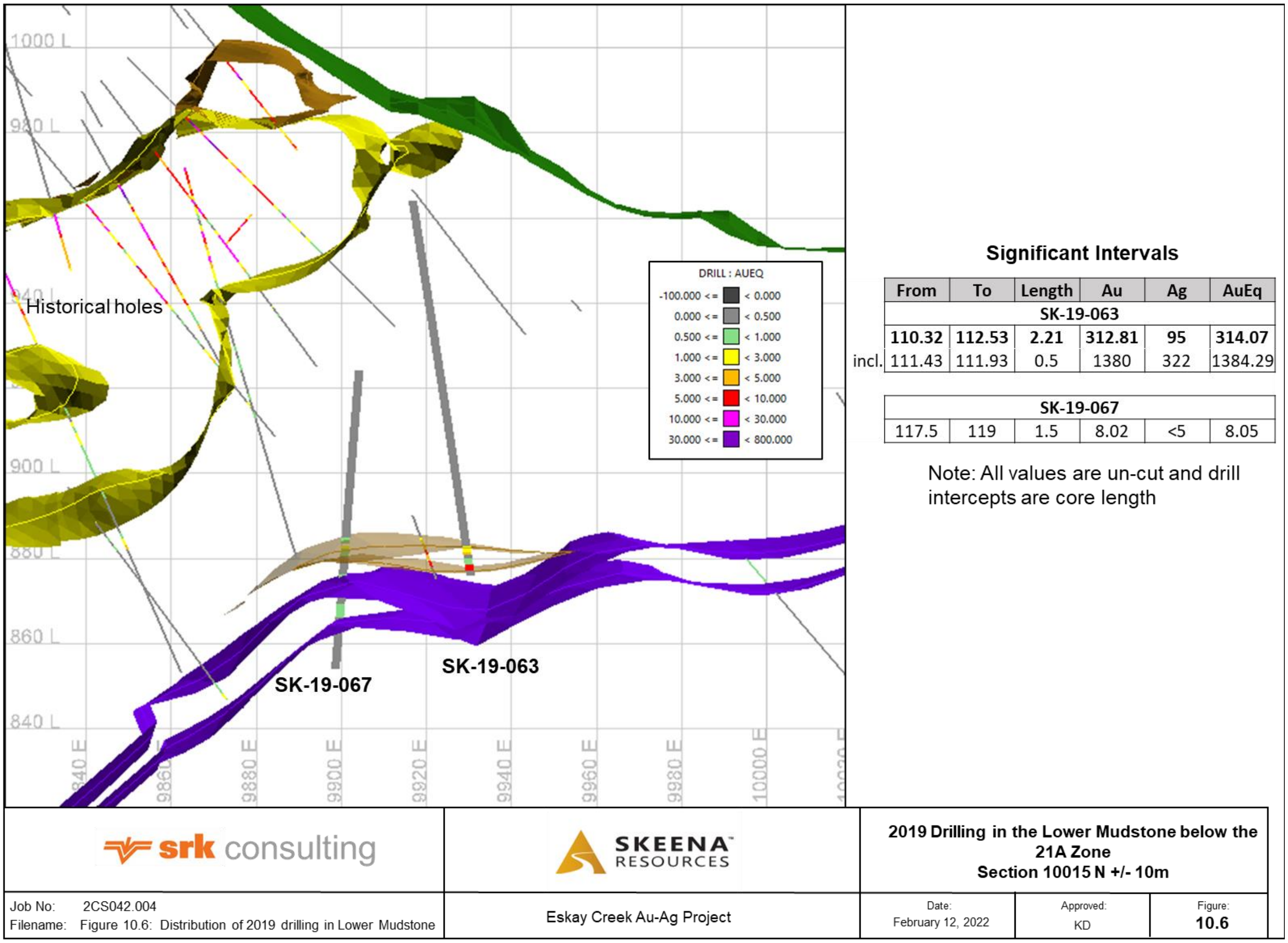
srk consulting | SKEENA RESOURCES | **2019 Drilling in the 21E Zone Section 10405 N +/- 10m**

Job No: 2CS042.004
Filename: Figure 10.5: 2018 Drilling in the 22 Zone

Eskay Creek Au-Ag Project

Date: February 12, 2022 | Approved: KD | Figure: **10.5**

Figure 10-6: 2019 Drilling in the Lower Mudstone below the 21A Zone

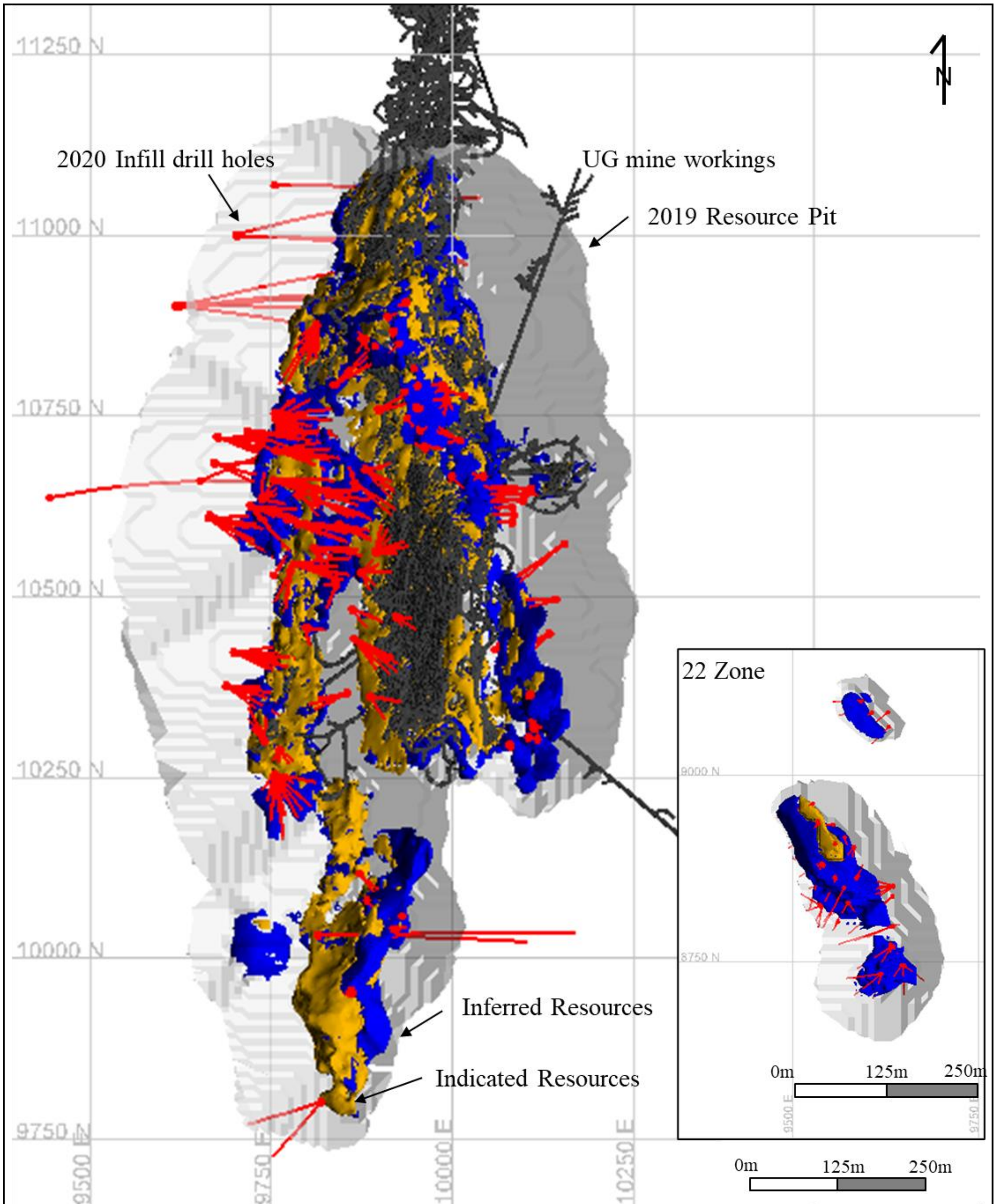


10.7 Results of the 2020 Phase 1 and Phase 2 Drilling

Skeena continued to embark on a major infill/conversion drilling program during 2020 to support an upgrade of the Inferred mineral resource confidence categories in the 22, 21A, 21C, 21B, 21E, HW and PMP Zones (Figure 10-7), as well as to continue 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. Phase 2 of the program allowed infill drilling within the 20m buffer zone imposed by Barrick around the underground workings.

The infill drilling program was successful in upgrading the confidence in the resource categories in most of the zones. Representative sections through the deposit are shown in Figure 10-8 and Figure 10-9. below in the 21C and HW Zones respectively.

Figure 10-7: 2020 Infill Drilling



srk consulting	SKEENA RESOURCES	Distribution of 2020 Conversion Infill Drilling with 22 Zone inset		
Job No: 2CS042.004 Filename: Figure 10.7 Distribution of 2020 Conversion Drilling	Eskay Creek Au-Ag Project	Date: July 26, 2022	Approved: KD	Figure: 10.7

Figure 10-8: 2020 Infill Drilling in the 21C Zone

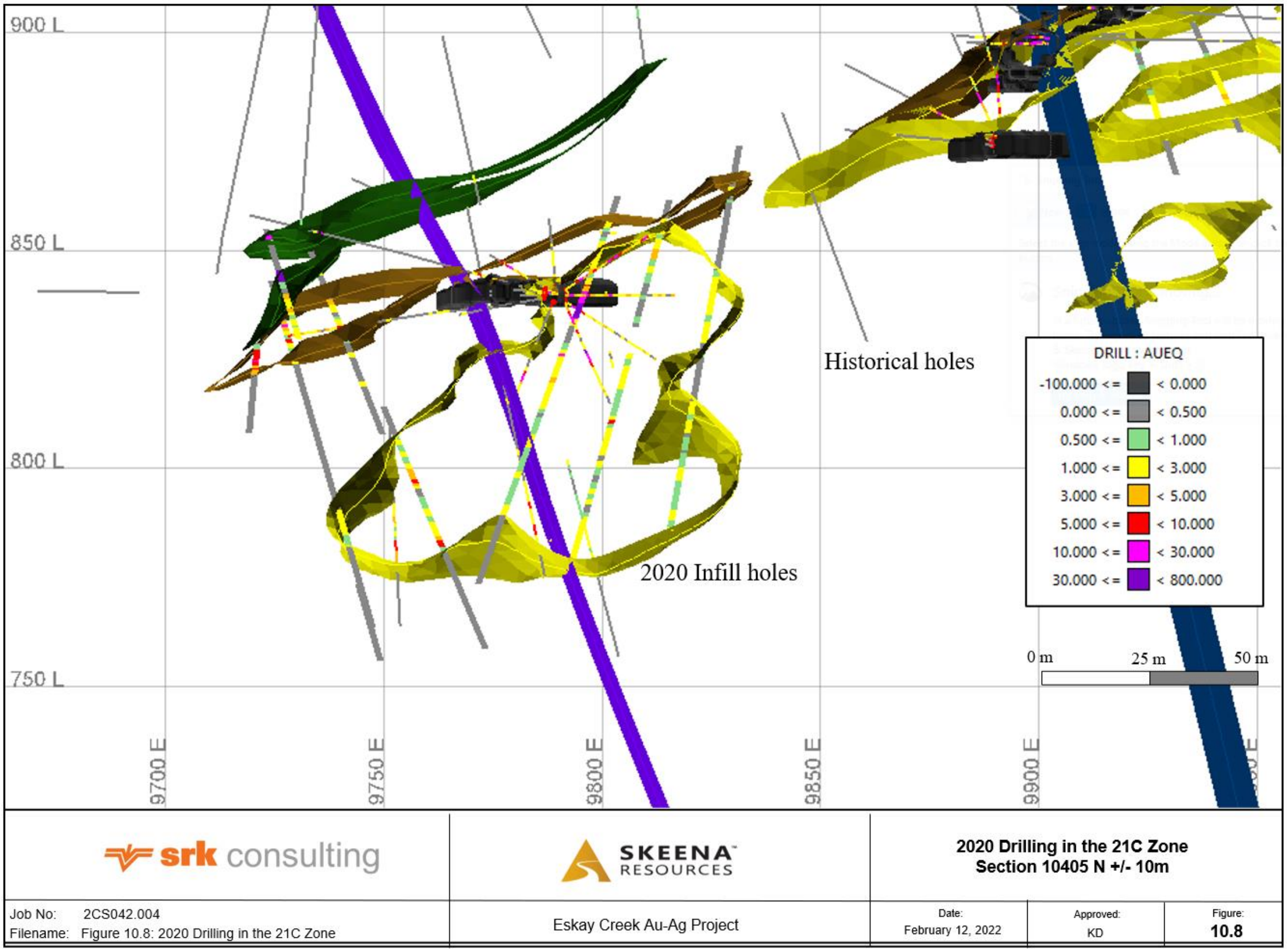
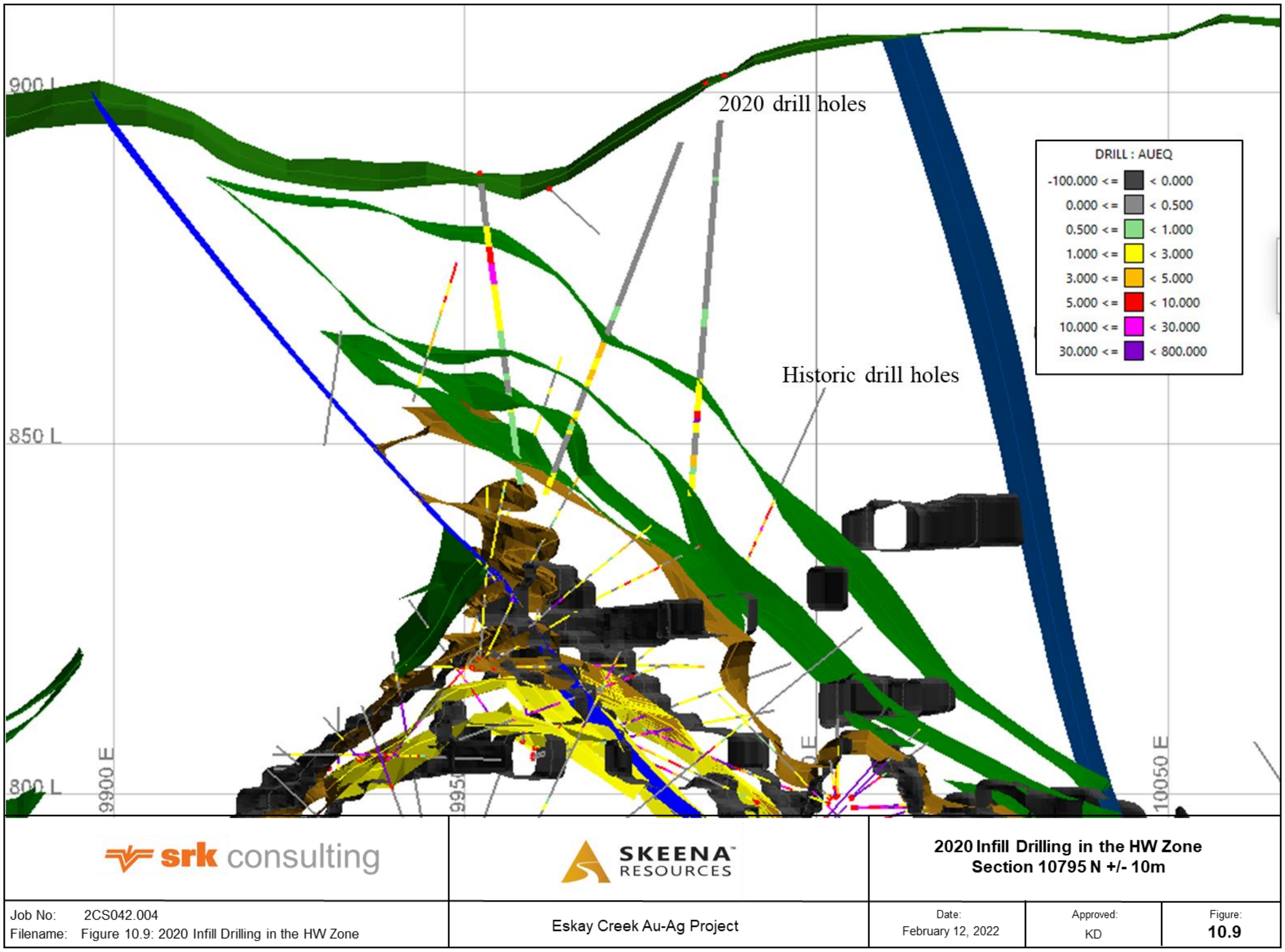


Figure 10-9: 2020 Infill Drilling in the HW Zone



10.8 Results of the Phase 3 2021 drilling

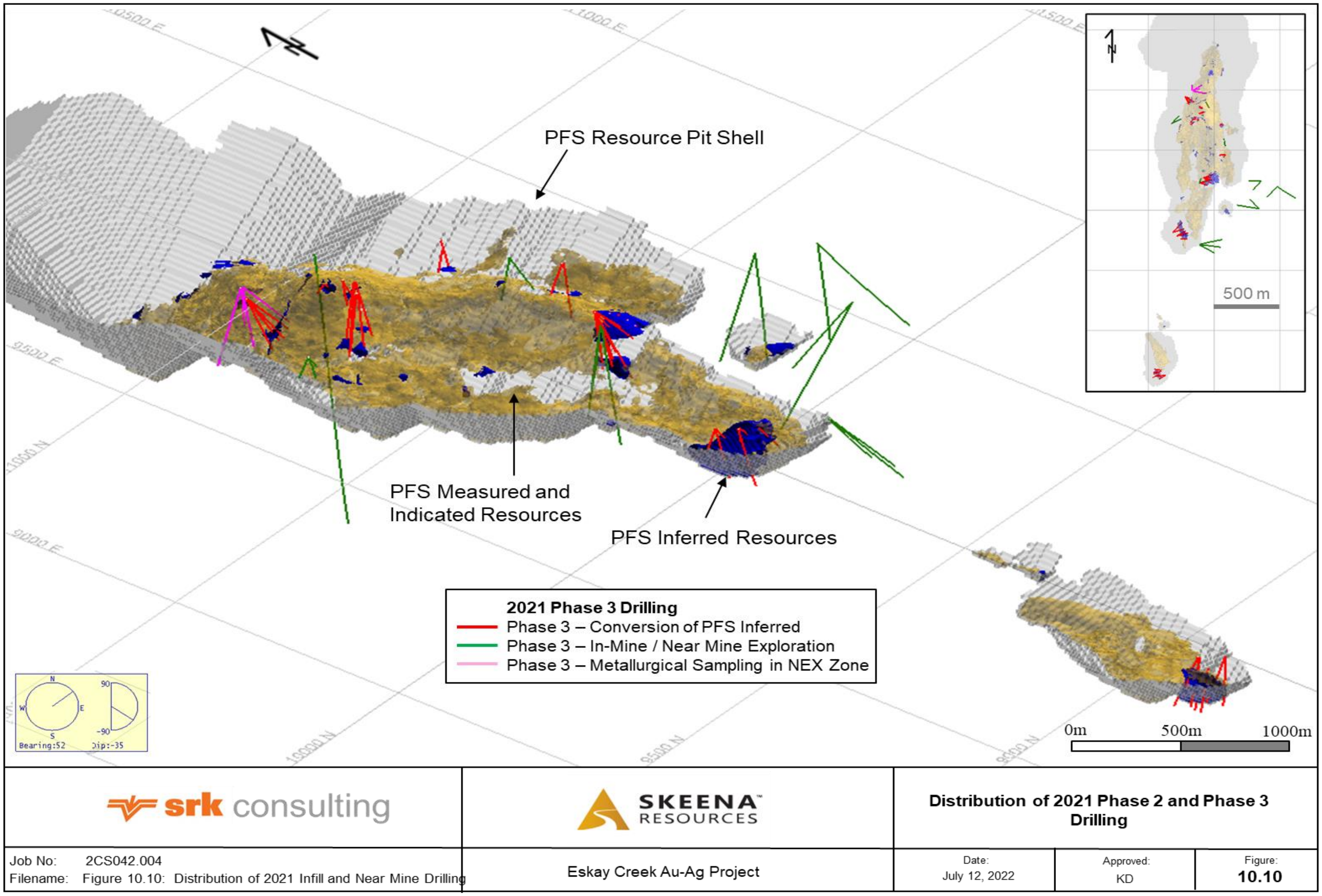
Following the release of the Preliminary Feasibility Study in 2021, the drilling for Phase 3 targeted the Inferred Resource with the aim of drill program to support upgrade of the mineral resource confidence categories in the 22, 21A Fault, HW, and PMP Zones. In addition, metallurgical samples for the NEX zone were collected, and near-mine drilling was undertaken to expand mineralization. Figure 10-3 shows the location of the conversion, metallurgy and infill drilling holes. These holes were drilled prior to the database cut-off of September 10, 2021 and were incorporated into the updated 2022 Mineral Resource Estimate.

The program was successful in converting most of the Inferred resources to either Indicated or Measured in the 22, 21B, 21A Fault, and HW Zones.

Mineralization in the 21C hanging-wall sediments, discovered in 2020 during the Phase 2 infill program in proximity to a subvertical, reactivated synvolcanic structural corridor, has been expanded 50 m along strike to the north and was upgraded to Indicated resources. This newly developing zone of mineralization remains open for expansion. The hanging wall sediments in the 21E Zone in proximity to the Pumphouse Fault were also expanded and upgraded to Inferred resources.

Near-mine drilling of the Eastern Dacites resulted in expansion of the mineralization solids and conversion to Inferred resources.

Figure 10-10: Distribution of 2021 Phase 3 Drilling



10.9 Drilling Completed since the Database Close out

The close out of the database used for the 2022 Mineral Resource Estimate was September 10, 2021, following the completion of Phase 3 drilling.

Since optioning Eskay Creek in 2017 and ultimately purchasing from Barrick in 2020, drilling has focused almost wholly on de-risking the project through category conversion (infill) drilling. The primary targets for the regional exploration program are bodies of near surface, bulk tonnage Au-Ag mineralization that occur along the main Eskay Trend from the 21A Zone south towards the 22 Zone and Tom MacKay Zones (Figure 10-11). Targets located south of the 21A were searching for synvolcanic feeders as well as mineralization hosted by the Lower Mudstone and Even Lower Mudstone units. Targets towards the east include the Eskay Porphyry, which has received only limited historical drilling and was never analysed for gold, as well as favourably altered Lower Mudstone and Dacite units located on the east limb of the Eskay Anticline. These lower stratigraphy target areas experienced only limited historical exploration activity through the early 1990's.

Highlights of the 2021 Exploration drilling include:

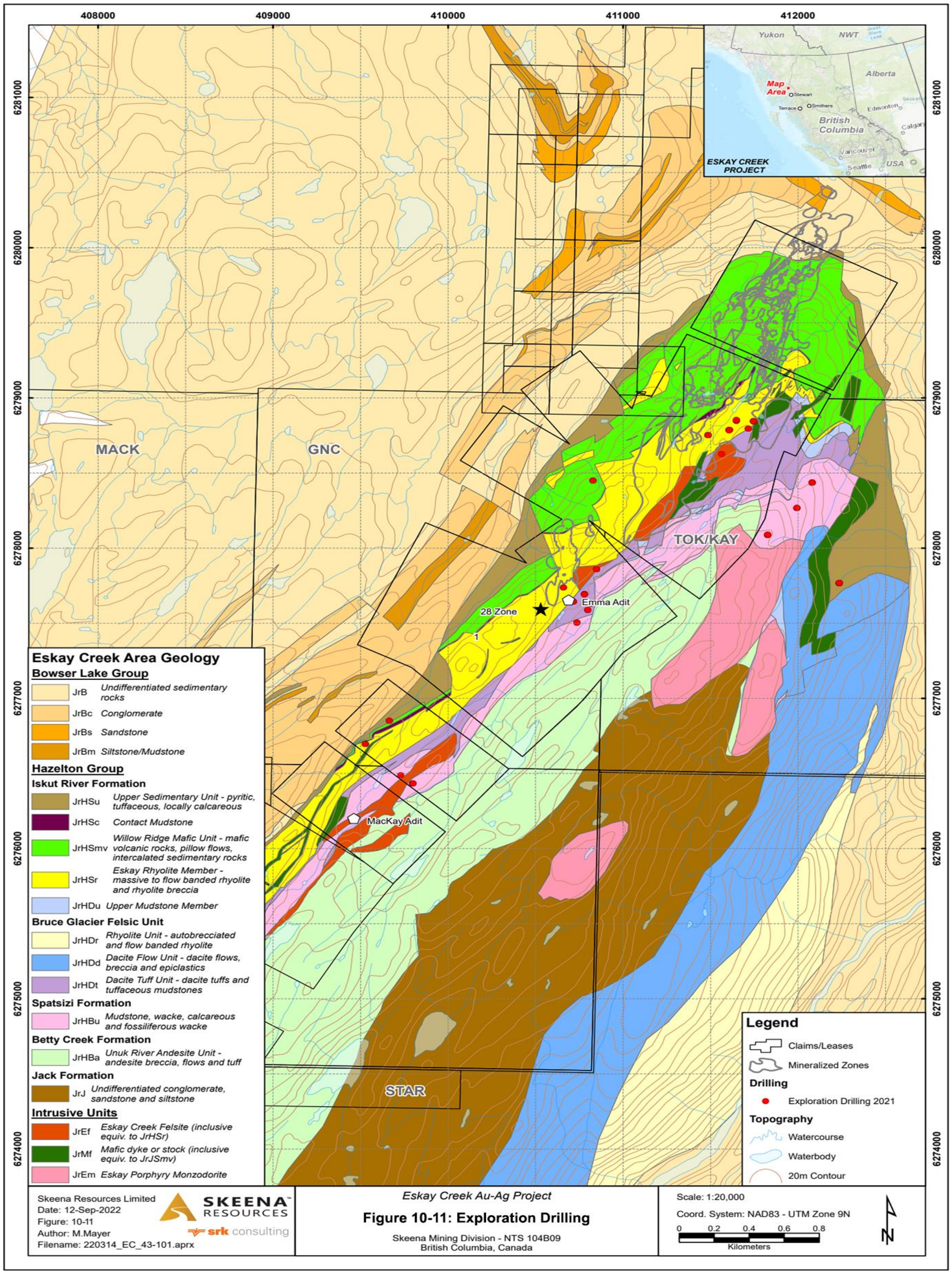
- The discovery and expansion of the 23 Zone, a zone of near surface Au-Ag mineralization beginning only 15 m vertically below surface and situated 200 m east of the high-grade 21A Zone. The area was historically drilled from surface by previous operators on widely spaced drill centers. Selective drill hole sampling during this era meant that the discovery of the larger and more robust mineralized widths observed as a result of the 2021 drilling program, were missed historically. Mineralization in the 23 Zone is almost exclusively hosted within dacitic volcanic rocks and to a lesser degree, the overlying Lower Mudstone unit. The Au-Ag tenor is consistent with that observed in footwall mineralization elsewhere on the property (21A, 21B, 21C, 22 Zones). Concentrations of the epithermal suite of elements (As, Hg, Sb) are negligible, as is the case with this style of footwall mineralization across Eskay Creek. Drilling to date indicates a shallow, westerly dipping geometry and the 23 Zone remains open for expansion in all directions. Proximity to surface, in addition to the grades and widths of this evolving zone are a perfect complement to the existing mine plan and potential throughput expansion.

- The presence of Au-Ag mineralization in the Even Lower Mudstone (ELM) approximately 200 metres southeast of the 22 Zone.

- Following up on resource category conversion drilling performed in early 2021, drilling intersected high-grade gold mineralization 60 m west of Skeena's current 21A Zone pit-constrained resources. Hosted entirely within the rhyolite sequence, this new mineralization is only 30 m vertically below surface. This discovery remains open for expansion 120 m to the north and already occurs within the limits of the contemplated open pit from Skeena's 2021 Prefeasibility Study.

- Gold in soil anomalies supported by geological considerations were targeted by nine drill holes on the eastern limb of the property. Despite intersecting highly altered host rocks, no significant grades were returned in these specific locations.

Figure 10-11: Location of the Exploration Drill Holes Drilled During 2021



10.10 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, Skeena performed a rigorous analysis of the historical data prior to adopting into their database.

11.1 2018 – Sept 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 quality assay 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 to 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. As of March 2022, the ME_MS81 method took precedence for Ba, Ga, La, U and Th as four-acid digestion will not fully dissolve some minerals such as barite.

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 over limits 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.

Antimony over limits were re-analysed by atomic absorption of a 0.2–0.4 g sample digested in a hydrochloric acid-potassium chlorate (Sb-AA08). The lower detection limit for this method was 0.1% and the upper detection limit was 100%.

11.1.2 QA/QC Verifications 2018 – Sept 2021

Skeena implemented formal QA/QC programs for all phases of drilling between 2018 and September 2021. In total, five drilling phases were completed, including 2018, 2019, 2020 Phase 1, 2020/2021 Phase 2, and 2021 Phase 3. For the purposes of reporting, QA/QC is discussed by year and in some cases, drilling phases overlap years. The close out date of the latest database is September 10, 2021, and QA/QC validations are only relevant up to and including the 2021 Phase 3 drilling program.

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 11-1 depicts 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	2018–2021	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	2018–2021	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	2019–2021	5.21	5.837	4.583	297	321	273
OREAS 622	2019–2021	1.85	2.048	1.652	102	111.9	92.1
CDN-ME-1902	2020–2021	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 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 (). The combined quality control samples equate to 51% of the total assays submitted in 2018.

Table 11-2: QC Samples 2018 Drilling Program

QC Sample	Type	Subtotal	Total	% of Total
Total Blanks			112	7%
CRMs	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 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 2019 Drilling Program

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%**

Five CRMs were used during the 2021 Phase 2 and Phase 3 drilling programs, three of which originate from CDN, and two from OREAS (Skeena, 2020a; Skeena, 2020b). A total of 270 control blanks, 355 reference samples, 44 preparation duplicates, and 272 pulp duplicates were inserted and analysed in 2021 (Table 11-5). The percentage of combined quality control samples equates to 12% of the total assay samples submitted in 2021.

Table 11-5: QC Samples Combined Phase 2 & 3 Drilling Programs 2021

QC Sample	Type	Subtotal	Total	% of Total
Total Blanks			270	29%
CRMs	CDN-GS-25	14		
	CDN-ME-1312	115		
	OREAS 603b	29		
	OREAS 622	113		
	CDN-ME-1902	84		
Total CRMs			**355**	**38%**
Duplicates (internal ALS)	Prep	44		
	Pulp	272		
Total Duplicates			**316**	**34%**
Total QC			**941**	**100%**

11.2 Specific Gravity Analysis

Specific gravity (SG) measurements were routinely collected from drill core during Skeena's 2018, 2019, 2020 Phase 1 and Phase 2, and 2021 Phase 2 and Phase 3 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 5,432 SG measurements were taken and categorized according to dominant lithology type and mineralization zone. Table 11-6 shows the nine dominant lithology types and twenty-eight ore zones versus their average SG values.

Table 11-6: Specific Gravity vs. Lithology per Estimation Zone

Zone	Rock Type	No. of Samples	Specific Gravity
1*	Bowser Group	1	2.64
2*	Hanging Wall Sediments	528	2.72
3*	Hanging Wall Andesite	1995	2.83
4*	Contact Mudstone	74	2.67
5*	Rhyolite	955	2.66
6*	Lower Mudstone	30	2.79
7*	Dacite	162	2.79
8*	Even Lower Mudstone	83	2.79
9*	Footwall Andesite	81	2.75
90	Lower Package - Rhyolite	2	2.69
91	Lower Package - Lower Mudstone	4	2.75
92	Lower Package - Dacite	39	2.83
93	Lower Package - Even Lower Mudstone	10	2.79
94	Lower Package - Footwall Andesite	8	2.78
95	Pumphouse - Rhyolite	9	2.67
99	109 - Rhyolite	5	2.67
101	101 - Rhyolite	196	2.63

Zone	Rock Type	No. of Samples	Specific Gravity
201	21A - Rhyolite	253	2.67
202	21A - Mudstone	25	2.81
301	21C - Rhyolite	198	2.67
302	21C - Mudstone	65	2.74
303	21C - HWA	26	2.87
401	21B - Rhyolite	197	2.67
402	21B - Mudstone	61	2.75
501	21Be - Rhyolite	24	2.68
502	21Be - Mudstone	2	2.73
504	Rhyolite/ Mudstone	0	2.75
601	21E - Rhyolite	68	2.66
602	21E - Mudstone	8	2.73
603	21E - HW	50	2.74
604	21E - North	2	2.79
703	HW	89	2.76
801	NEX - Rhyolite	6	2.67
802	NEX - Mudstone	8	2.73
811	WT - Rhyolite	9	2.67
3021	21C MS with Barite	24	3.89
7021	HW with Barite	25	2.98
Average		**5,432**	**2.75**

* Areas outside but proximal to the main ore zones (90 to 7021) were also ascribed an SG value.

Specific gravity was coded into the resource model using rock type divisions per estimation zone. Two additional units were evaluated separately due to their barite-rich contents: the 21C mudstone and HW units were coded with higher overall specific gravity values. In addition, a default value of 2.75 was applied to blocks for which lithology had not been coded.

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 Sept 10, 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, and 2021 Phase 2 and Phase 3 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 2022 Mineral Resource estimate was submitted to SRK on September 10, 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, 2020/2021 Phase 2, and 2021 Phase 3 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 826 drill holes drilled during the 2018, 2019, 2020 Phase 1 and Phase 2, and 2021 Phase 2 and 3 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, 2020 Phase 1, and Phase 2 drilling programs, whereby the database was compared directly with the provided assay certificates. Certificates for the 2019, 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

Ten percent (10%) of all samples from the 2021 Phase 2 and Phase 3 drilling programs were checked manually, and no errors were found.

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 and 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 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, 2020/2021 Phase 2, and 2021 Phase 3 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–Sept 2021	ALS Global	Reference material, blanks, in-house prep, and pulp repeats	Certificates available

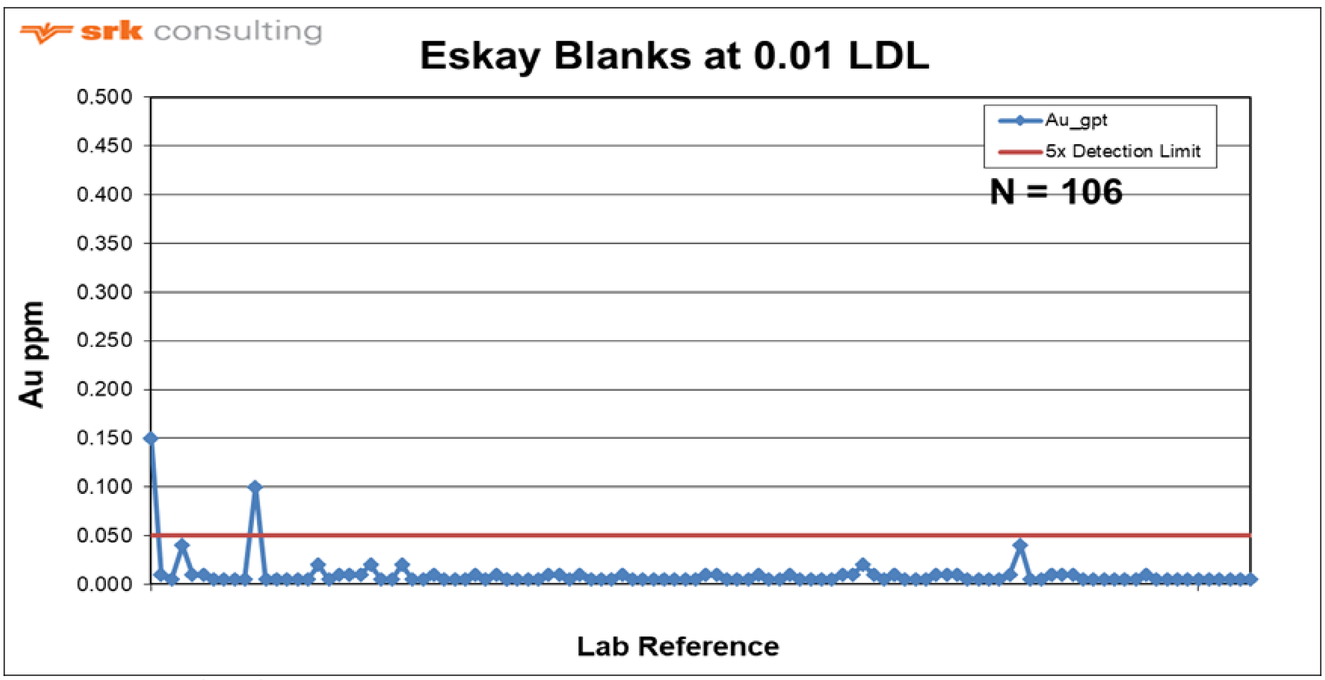
12.1.4 2018 – Early 2021 QA/ QC

Official QA/QC programs were undertaken in 2018, 2019, 2020 Phase 1, and 2020/2021 Phase 2, and 2021 Phase 3, 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.

12.1.4.1 2018 QA/ QC

An analysis of 106 blank gold samples confirmed that the least amount of contamination was transferred from sample to sample (Figure 12-1). 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 drilling program.

Figure 12-1: 2018 Drilling Campaign Blank Results



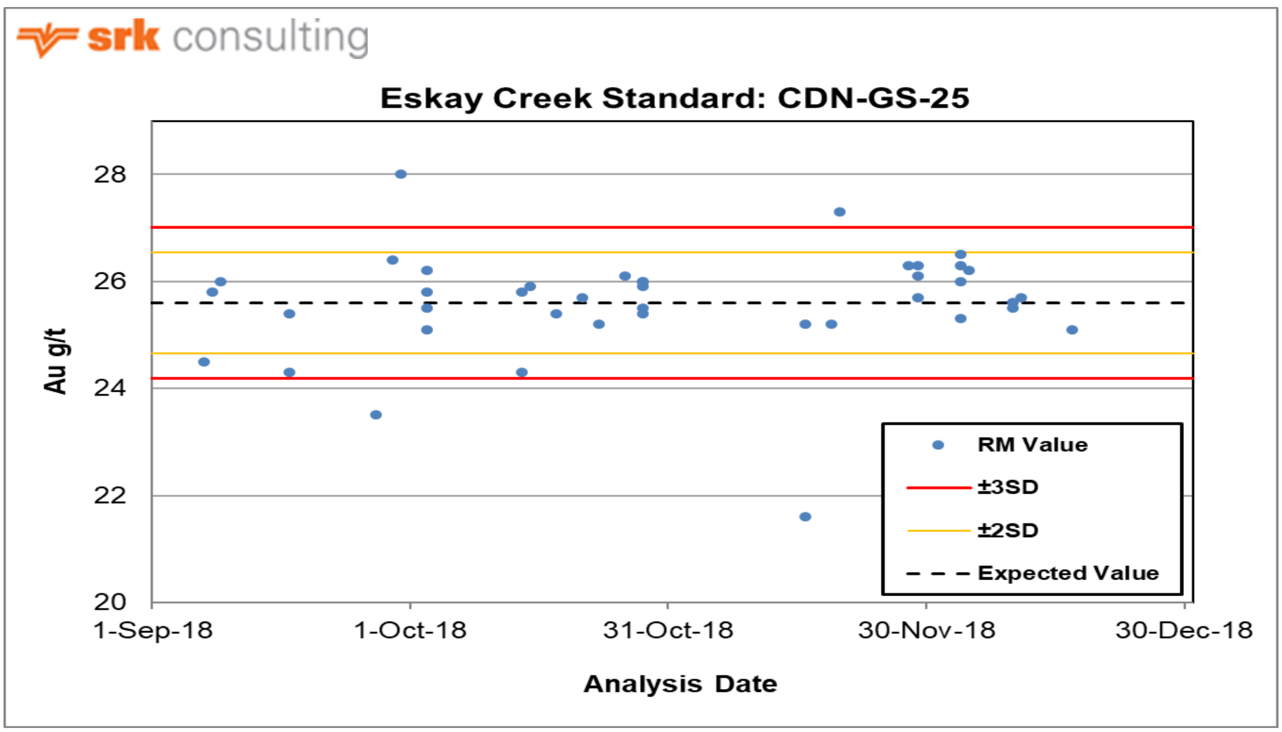
Note: Figure prepared March 23, 2021

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 12-2, Figure 12-3, Figure 12-4, and Figure 12-5). 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 12-2). 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 12-3). 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 12-4). 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.

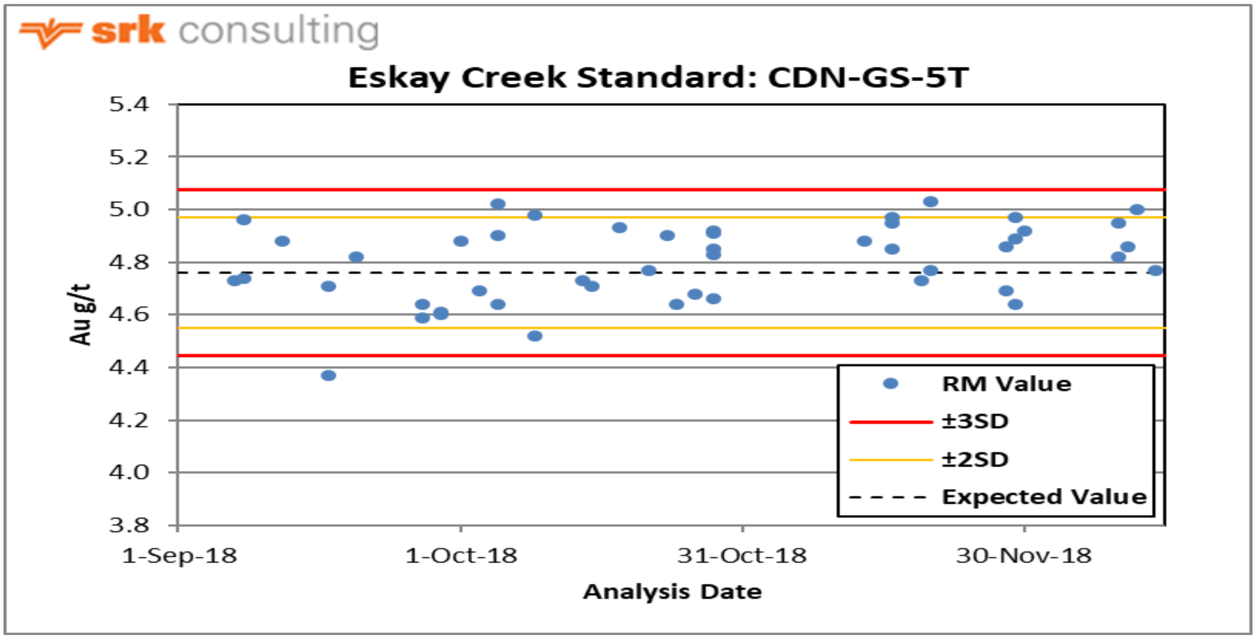
CRM CDN-ME-1601 resulted in several sample mislabels, which were duly corrected (Figure 12-5). Four samples occurred above the three standard deviation limit and above 10% of the expected value. These four samples occurred within low-grade assays, and it was not considered necessary or material to retest the surrounding assays.

Figure 12-2: Standard CDN-GS_25 from the 2018 Drilling Campaign



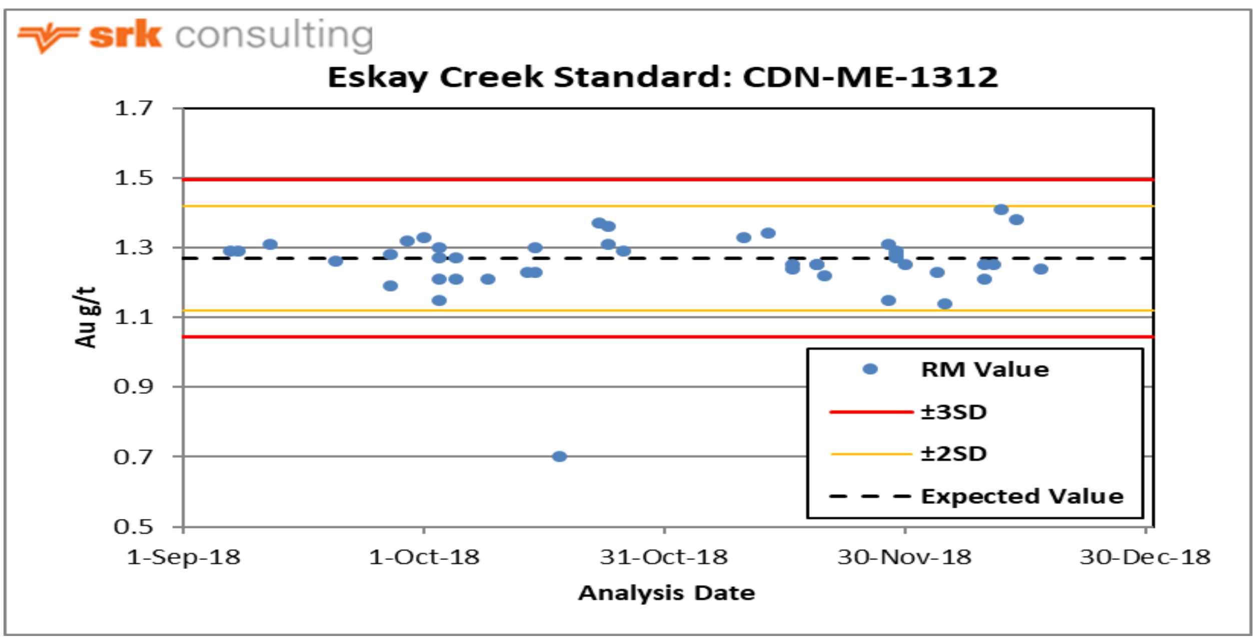
Note: Figure prepared March 23, 2021

Figure 12-3: Standard CDN-GS-5T from the 2018 Drilling Campaign



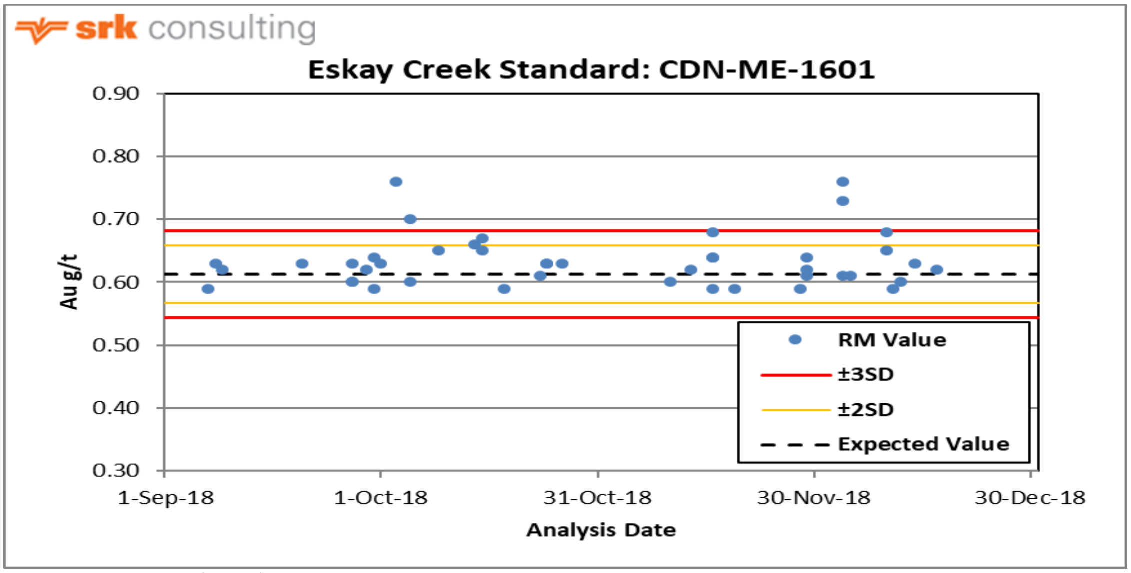
Note: Figure prepared March 23, 2021

Figure 12-4: Standard CDN-ME 1312 from the 2018 Drilling Campaign



Note: Figure prepared March 23, 2021

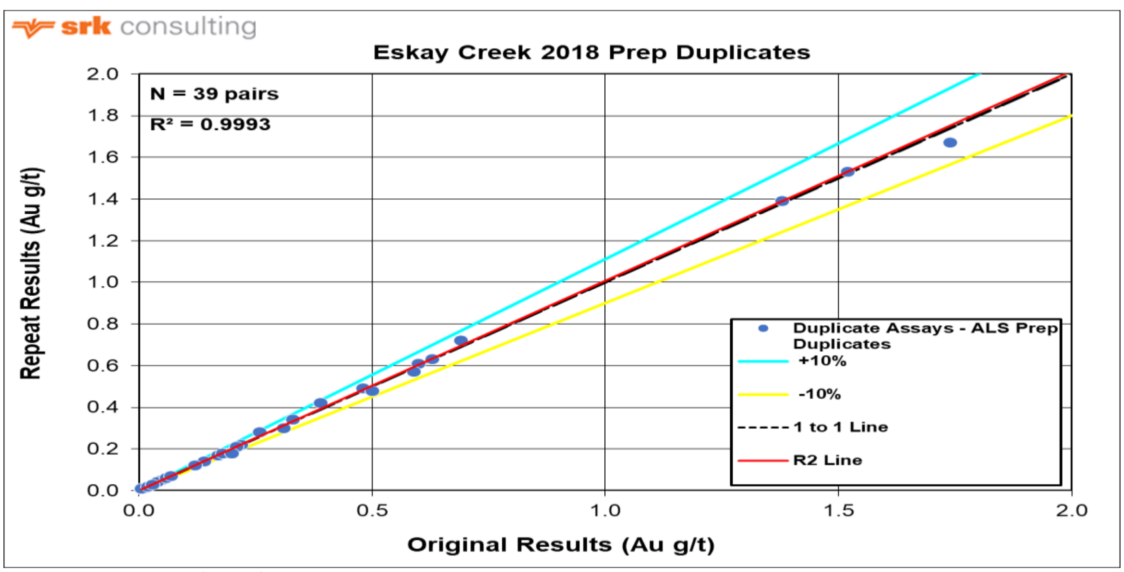
Figure 12-5: Standard CDN-ME-1601 from the 2018 Drilling Campaign



Note: Figure prepared March 23, 2021

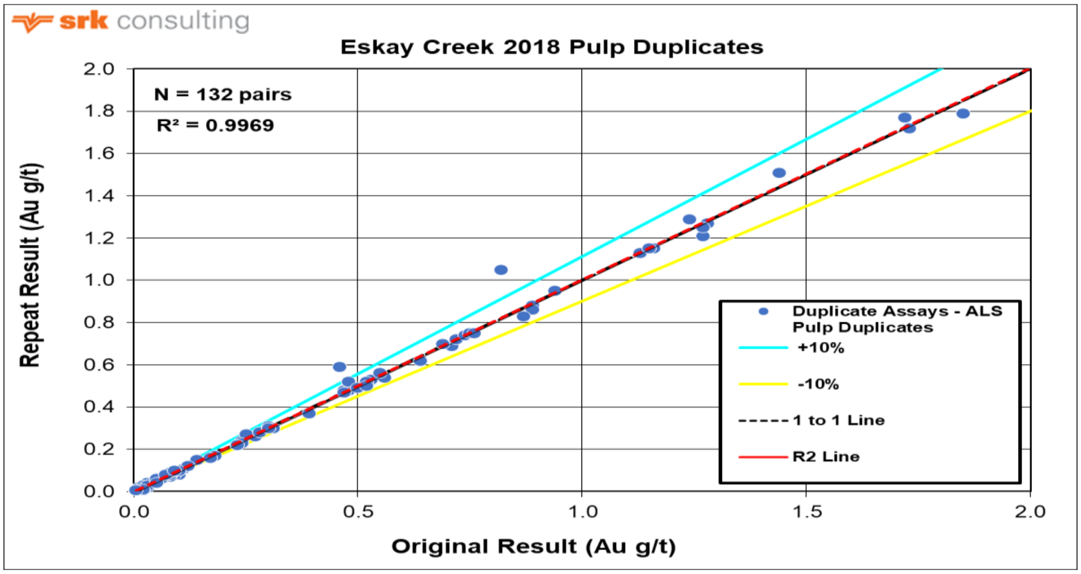
Preparation (rejects) and pulp duplicates were routinely run at ALS as part of the laboratory's internal QA/QC procedures. Paired preparation and pulp data performed within acceptable tolerance criteria at both lower-grade and higher-grade values (Figure 12-6 and Figure 12-7).

Figure 12-6: Gold Prep Duplicate Samples from the 2018 Drilling Campaign



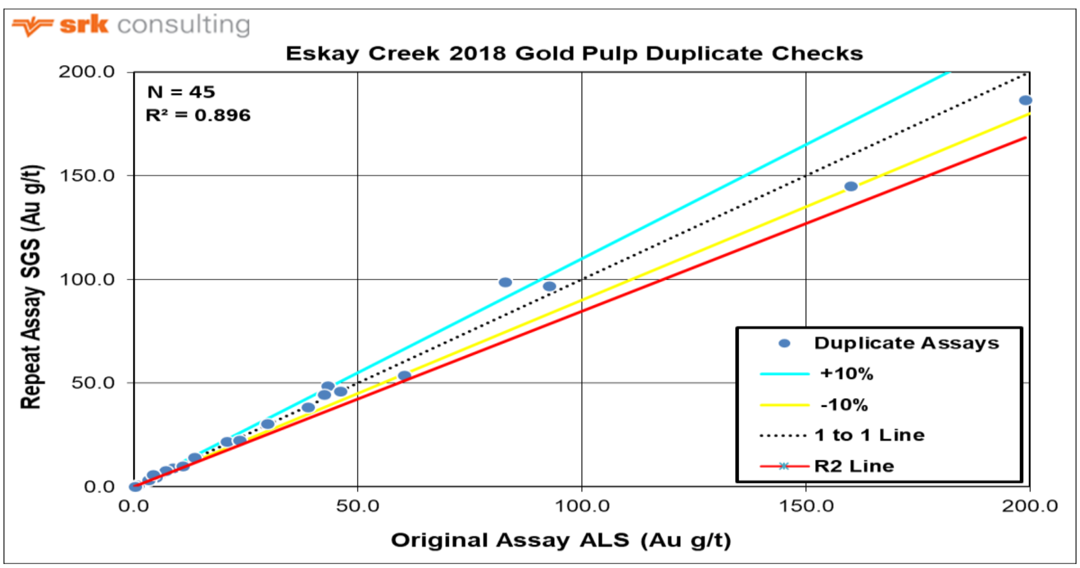
Note: Figure prepared March 23, 2021

Figure 12-7: Gold Pulp Duplicate Samples from the 2018 Drilling Campaign



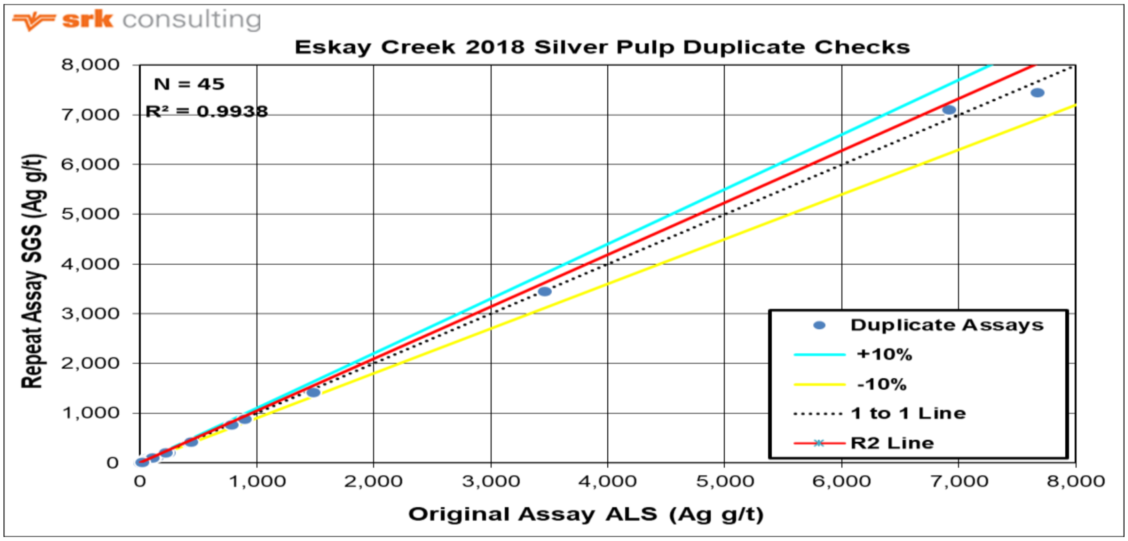
Note: Figure prepared March 23, 2021

Figure 12-8: Gold Pulp Duplicate Check Samples from the 2018 Drilling Campaign



Note: Figure prepared March 23, 2021

Figure 12-9: Silver Pulp Duplicate Check Samples from the 2018 Drilling Campaign

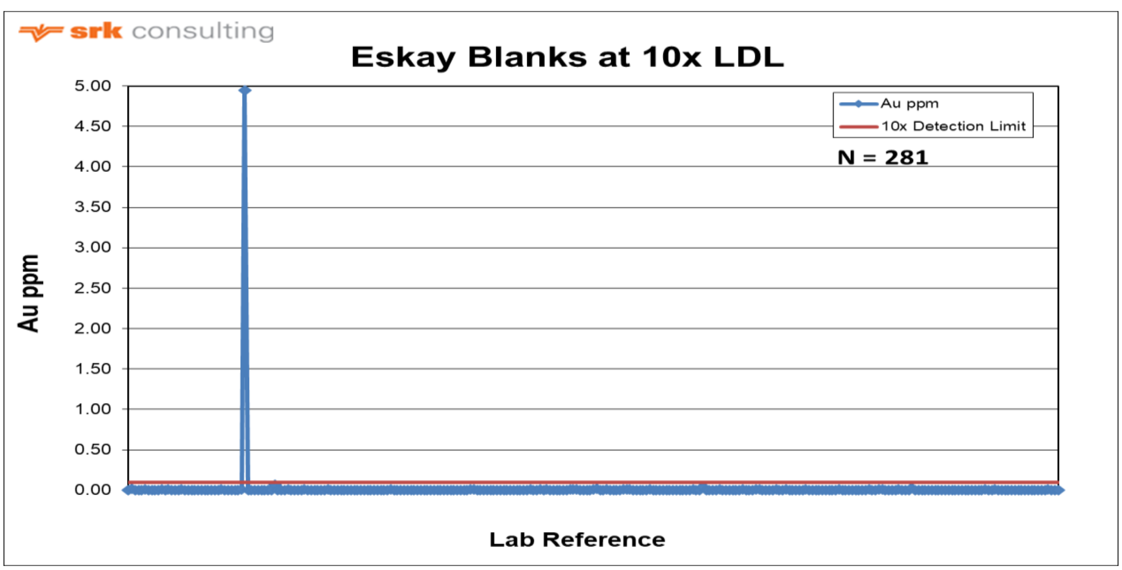


Note: Figure prepared March 23, 2021

12.1.4.2 2019 QA/ QC

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 12-10). 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 no re-assays were run.

Figure 12-10: 2019 Drilling Campaign Blank Results

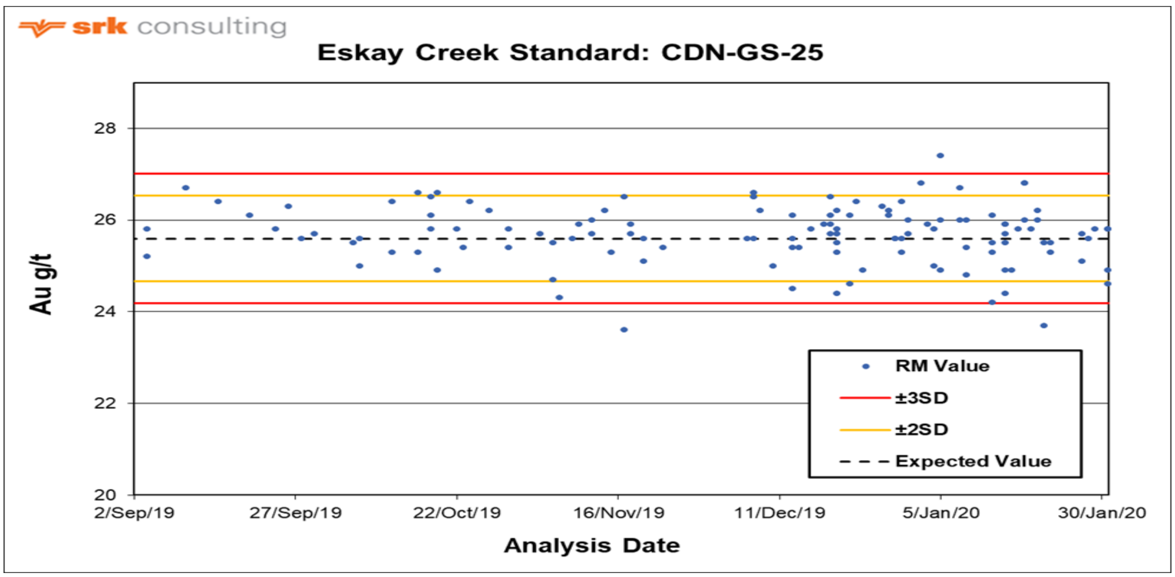


Note: Figure prepared March 23, 2021

Five commercially produced CRMs were inserted into the sample stream during the 2019 drilling program. An analysis of three CRM charts for high, medium, and low gold grades showed no obvious errors or bias (Figure 12-11, Figure 12-12, and Figure 12-13). 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 12-11).

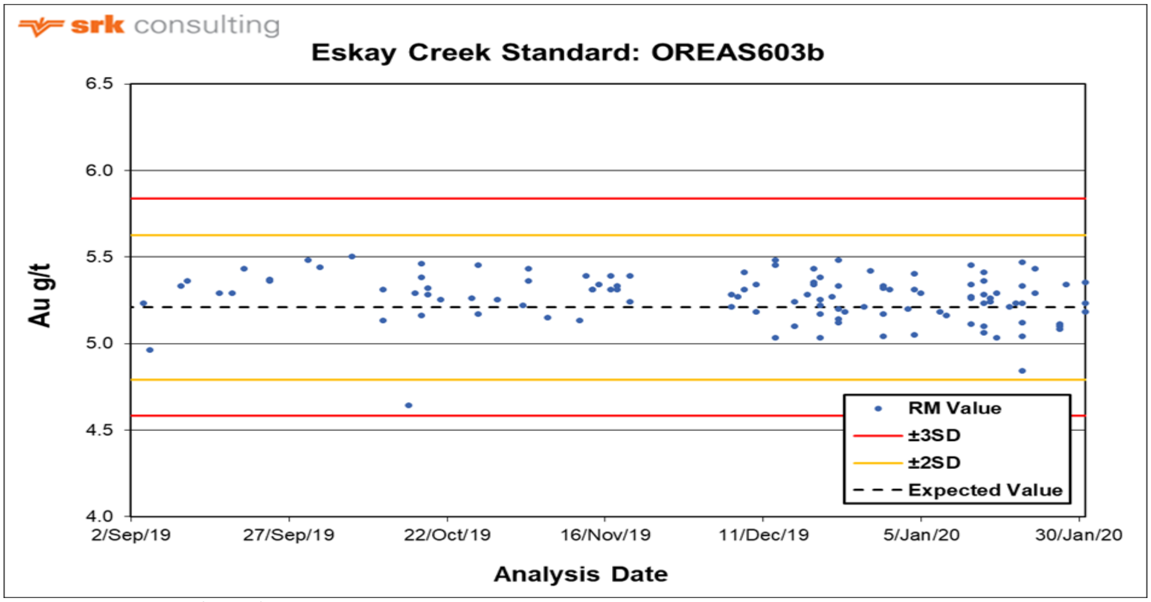
CRM OREAS603b demonstrated acceptable results for gold with all samples falling within the three standard deviations value (Figure 12-12). Similarly, CRM CDN-ME-1312 results are evenly spread about the expected value and occur wholly within the three standard deviation limits (Figure 12-13).

Figure 12-11: Standard CDN-GS-25 from the 2019 Drilling Campaign



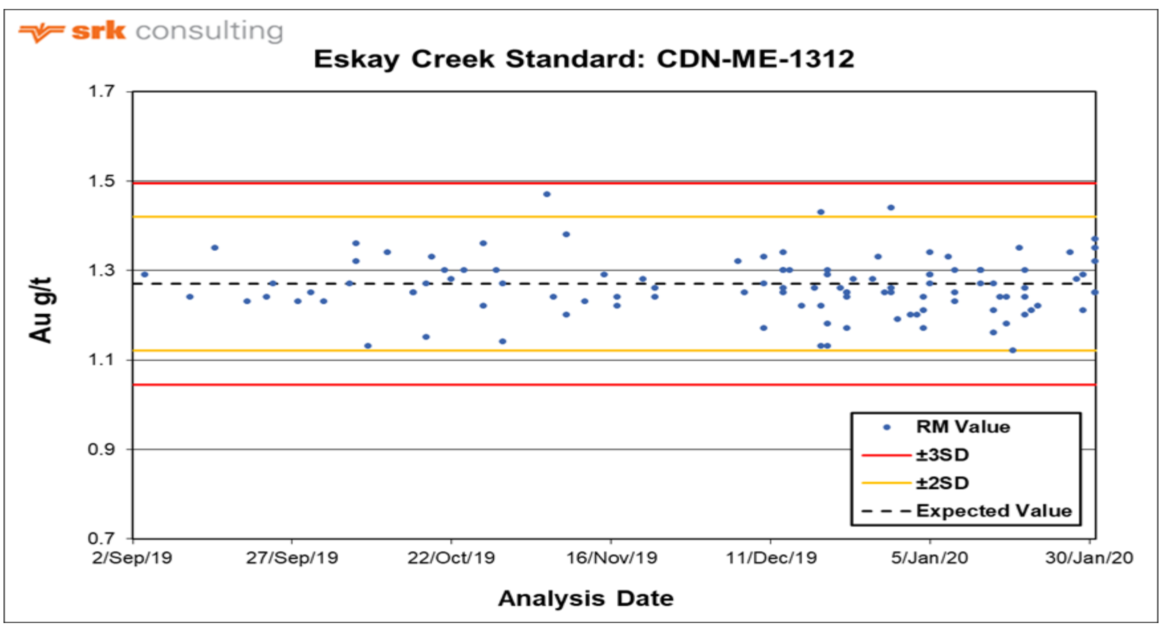
Note: Figure prepared March 23, 2021

Figure 12-12: Standard OREA S603b from the 2019 Drilling Campaign



Note: Figure prepared March 23, 2021

Figure 12-13: Standard CDN-ME-1312 from the 2019 Drilling Campaign

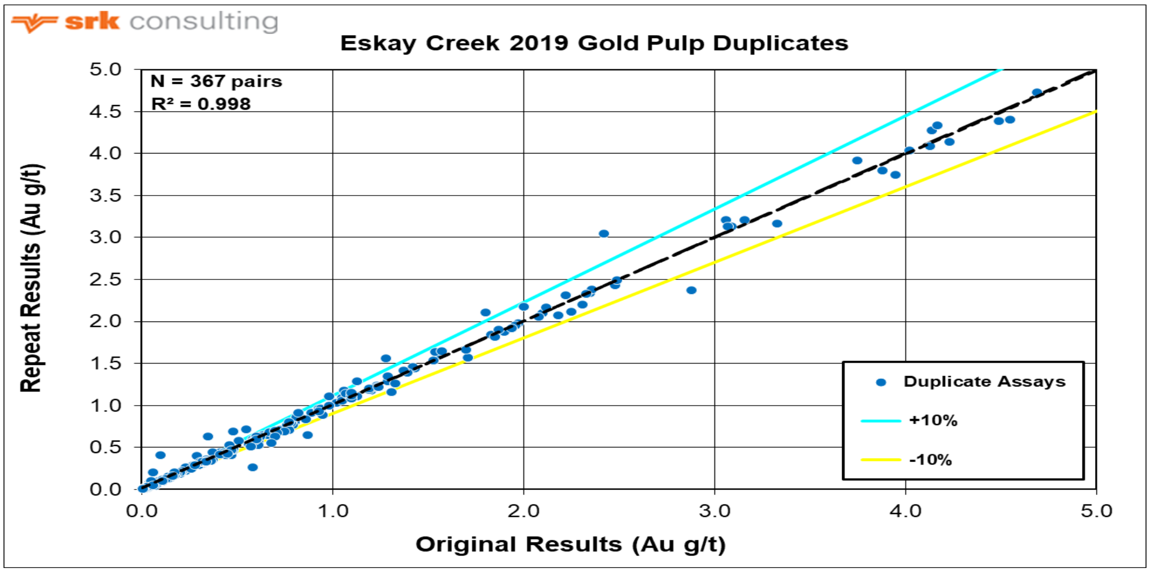


Note: Figure prepared March 23, 2021

Paired preparation and pulp data performed in 2019 occurred within acceptable tolerance criteria at both lower-grade and higher-grade values (Figure 12-14 and Figure 12-15).

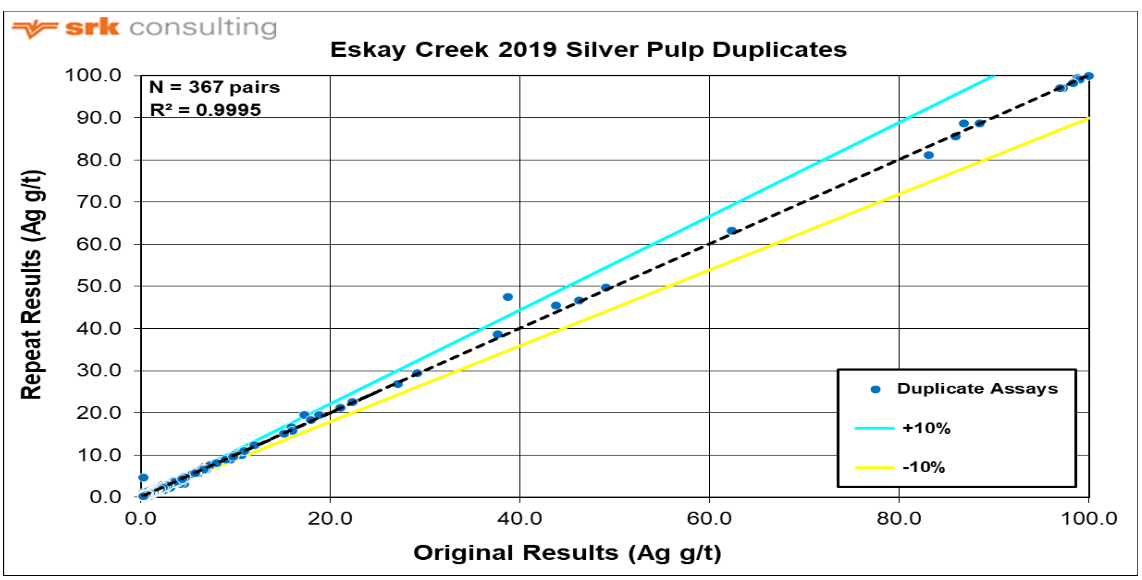
At the end of the 2019 Eskay Creek drilling program, a random selection of 2.5% of all assay samples, of which 1.5% occurred within moderate to higher gold grades, were selected and sent to SGS Canada, (Skeena, 2019b). A total of 215 pulps were checked against pulps originally processed at ALS Vancouver,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

Figure 12-14: Gold Pulp Duplicate Samples from the 2019 Drilling Campaign



Note: Figure prepared March 23, 2021

Figure 12-15: Silver Pulp Duplicate Samples from the 2019 Drilling Campaign

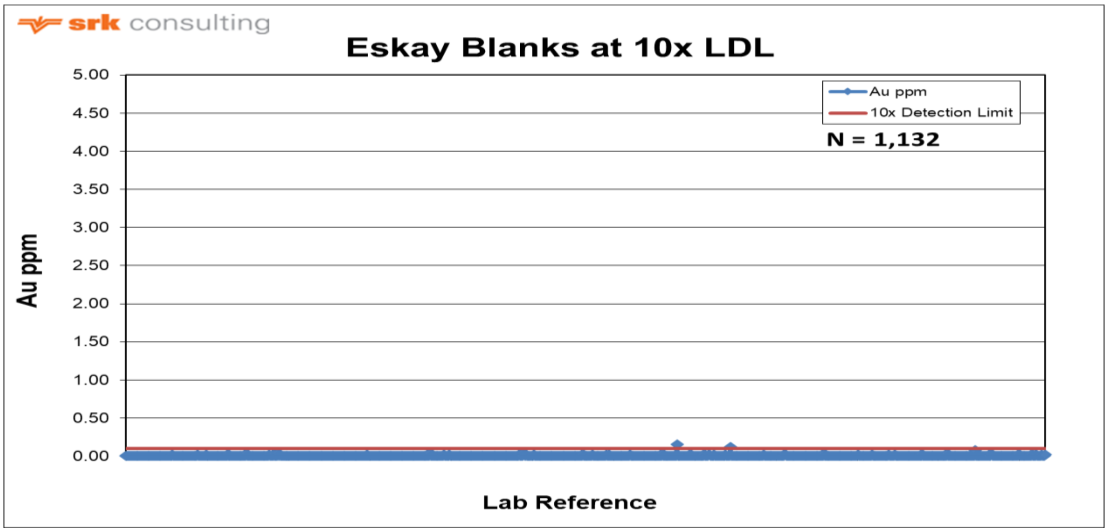


Note: Figure prepared March 23, 2021

12.1.4.3 2020 QA/ QC

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 12-16). Having no effect on the resource estimate, they were, therefore, not retested.

Figure 12-16: 2020 Drilling Campaign Blank Results

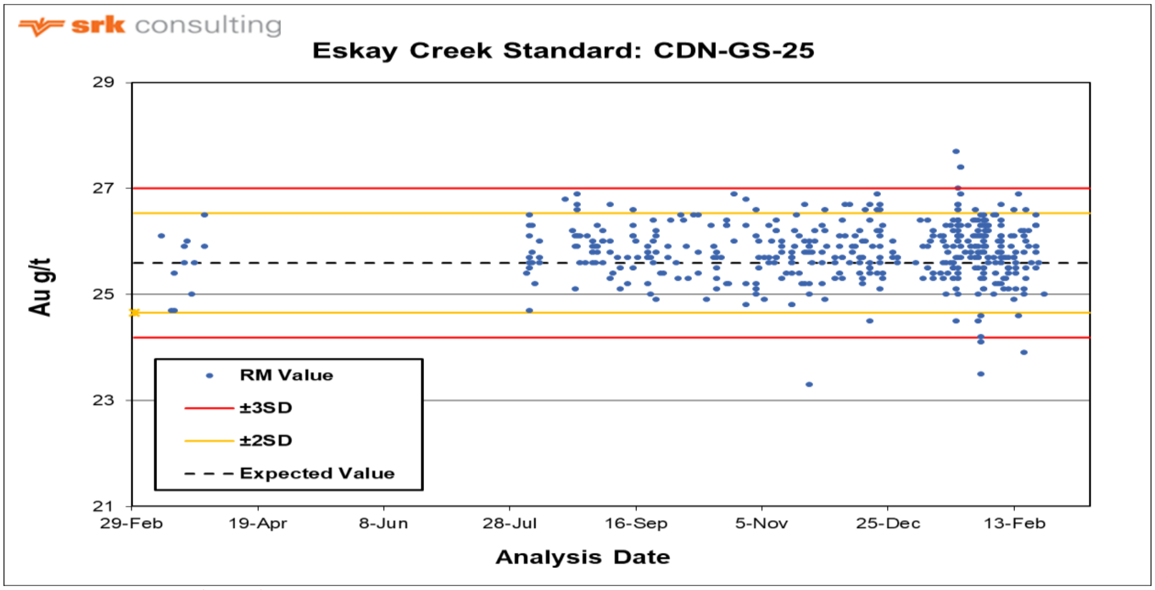


Note: Figure prepared March 23, 2021

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 12-17, Figure 12-18, and Figure 12-19). CRM CDN-GS-25 demonstrated even spread about the expected value for gold, although four samples occurred below the three standard deviation limits (Figure 12-17).

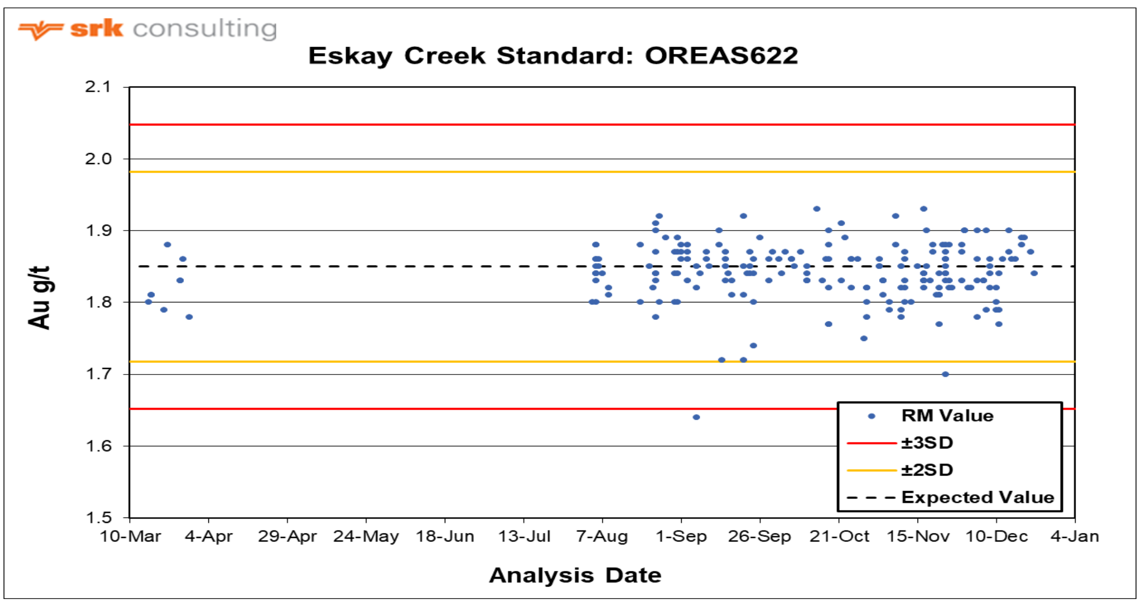
CRM OREAS622 demonstrated acceptable results for gold with all, excepting one sample, falling within the three standard deviation limits (Figure 12-18). Similarly, CRM CDN-ME-1312 results are evenly spread about the expected value and occur wholly within the three standard deviation limits (Figure 12-19).

Figure 12-17: Standard CDN-GS-25 from the 2020 Drilling Campaign



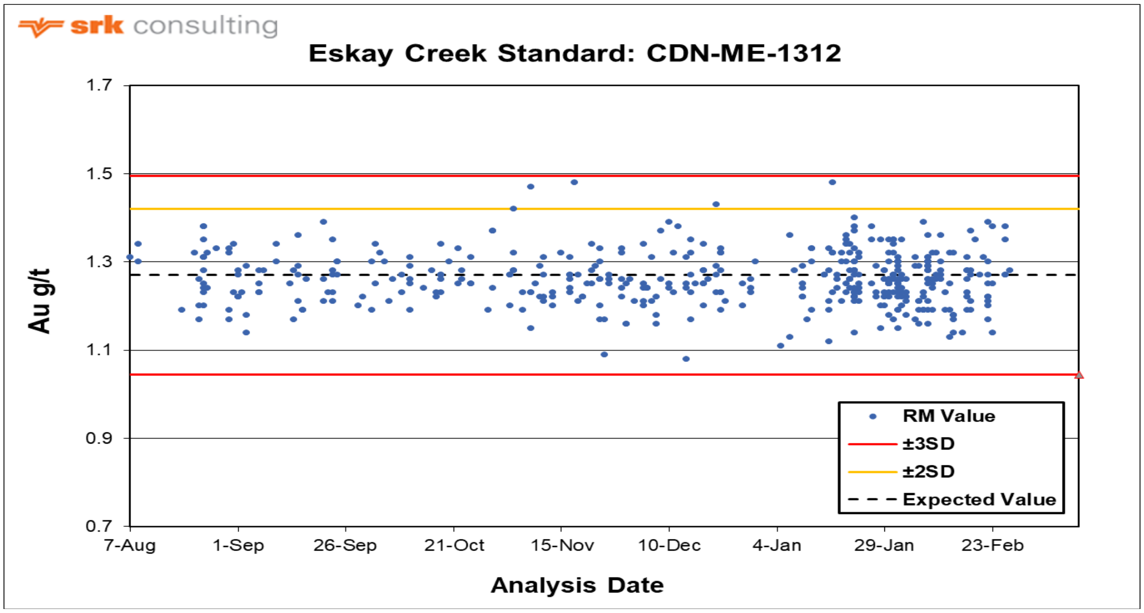
Note: Figure prepared March 23, 2021

Figure 12-18: Standard OREA S622 from the 2020 Drilling Campaign



Note: Figure prepared March 23, 2021

Figure 12-19: Standard CDN-ME-1312 from the 2020 Drilling Campaign



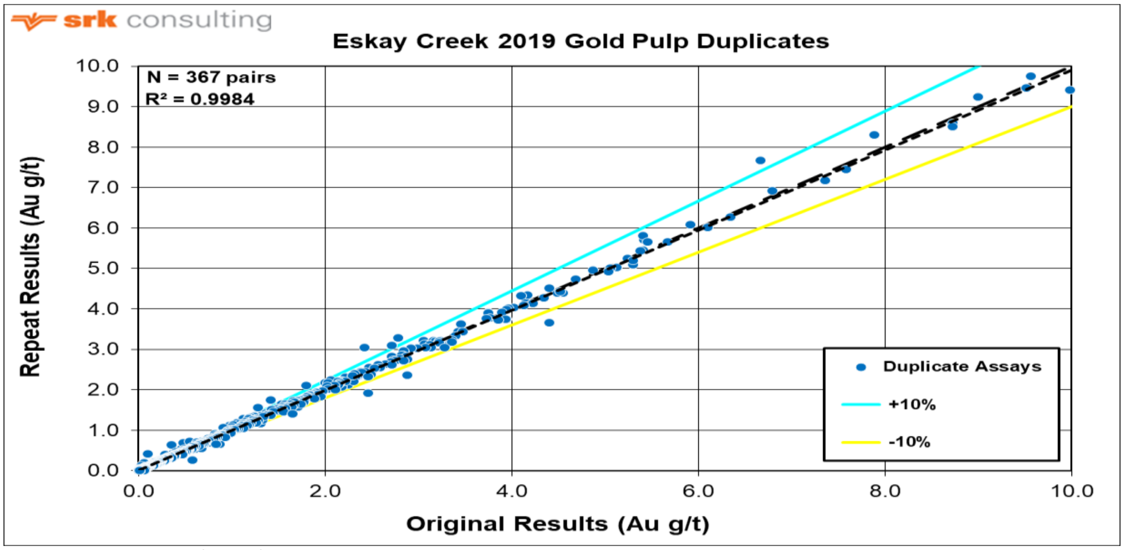
Note: Figure prepared March 23, 2021

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 12-20 and Figure 12-21).

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 11 control blanks and 19 CRMs were processed at SGS. The standard GS-13A performed poorly and will need to be monitored closely in future. However, the control blanks and remaining CRMs performed within acceptable limits. A total of 385 pulp duplicates were checked against pulps originally processed at ALS Vancouver and the paired results were acceptable.

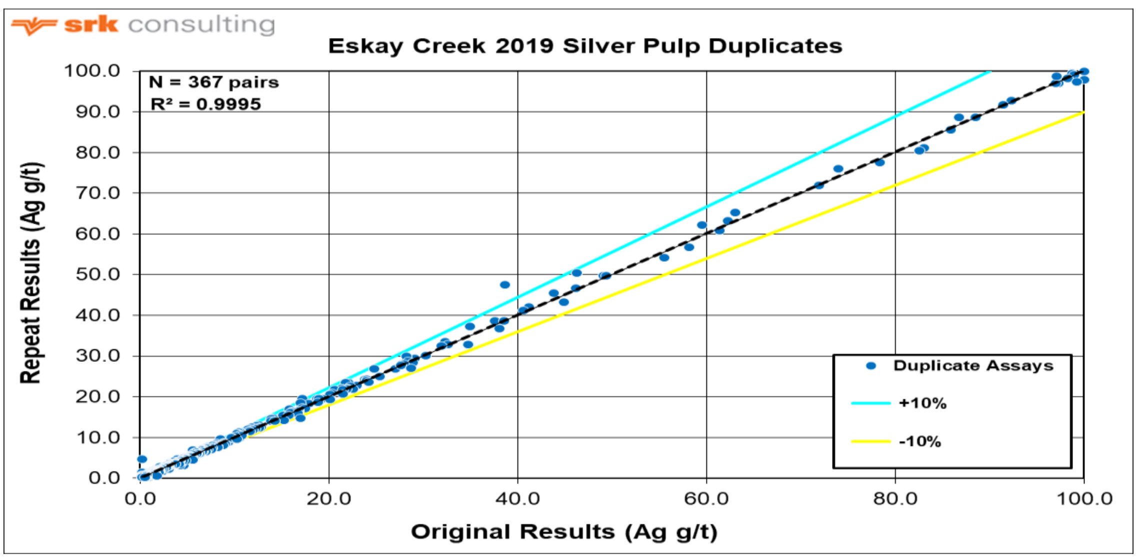
At the end of the 2020 Phase 2 drilling program 4 control blanks and 7 CRMs were processed at SGS. In addition, 530 pulp duplicates were compared with pulps originally processed at ALS Vancouver. All quality control checks for Phase 2 were found to be acceptable.

Figure 12-20: Gold Pulp Duplicate Samples from the Combined 2020 Drilling Campaigns



Note: Figure prepared March 23, 2021

Figure 12-21: Gold Pulp Duplicate Samples from the 2020 Combined Drilling Campaign

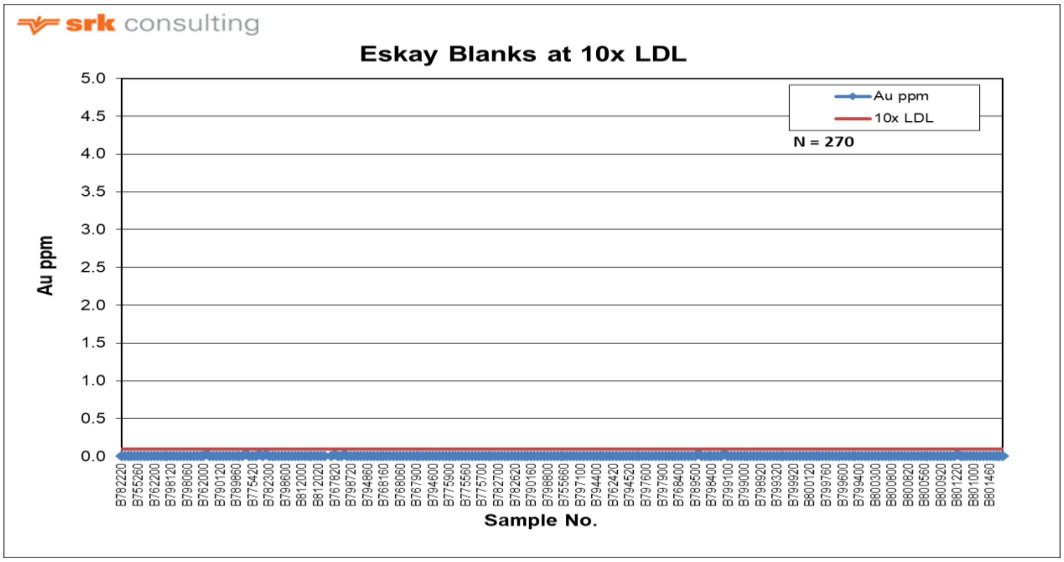


Note: Figure prepared March 23, 2021

12.1.4.4 2021 QA/ QC

A total of 270 control blanks were inserted during the two 2021 drilling campaigns (Phase 2 and Phase 3). Zero samples registered slightly above the 10x detection limit (Figure 12-22).

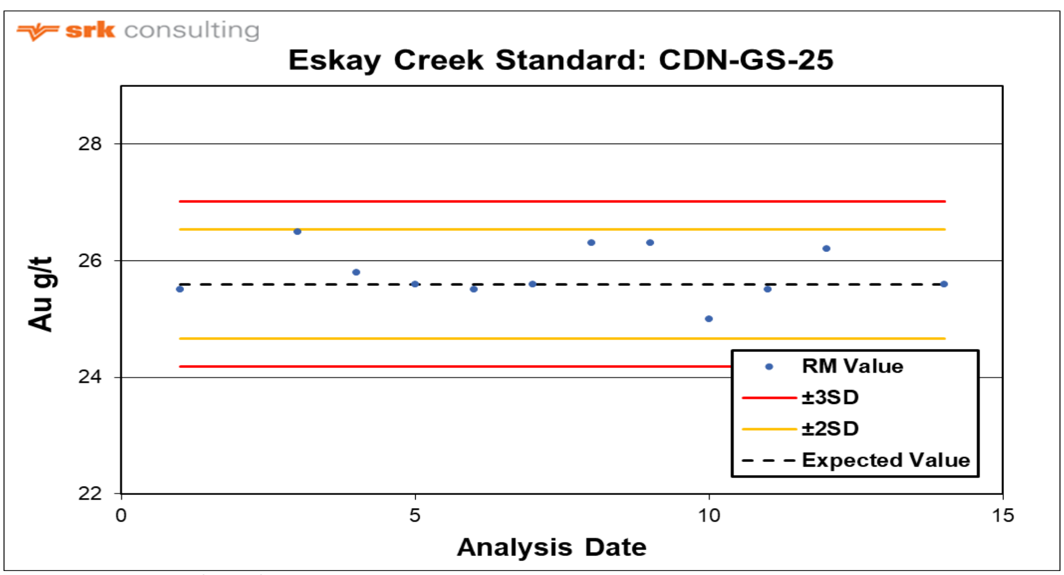
Figure 12-22: 2021 Drilling Campaign Blank Results



Note: Figure prepared March 17, 2022

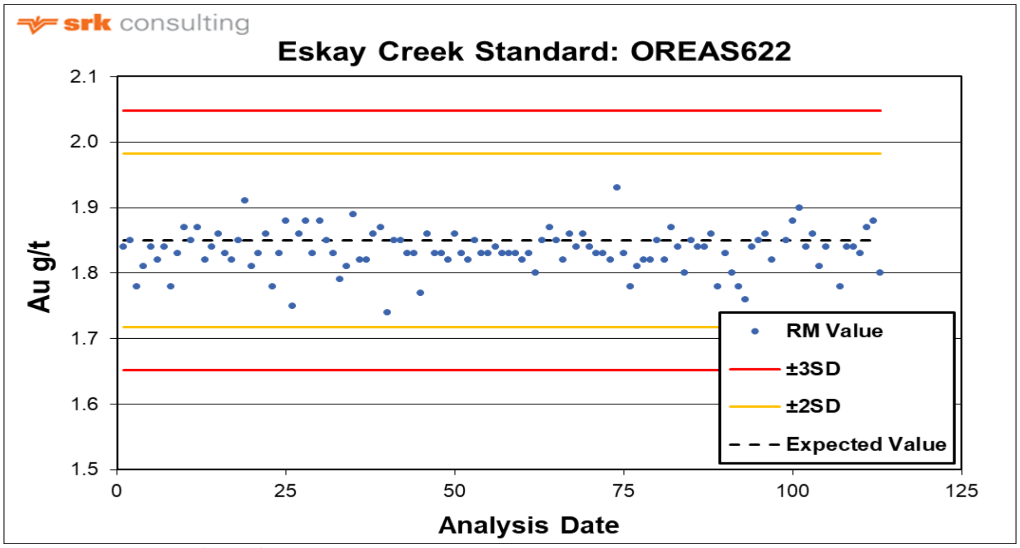
Five different types of CRMs were inserted into the sample stream during the 2021 drilling programs. An analysis of three CRM charts for high, medium, and low gold grades showed no obvious errors or bias (Figure 12-23, Figure 12-24 and Figure 12-25). CRM OREAS622 demonstrated acceptable gold results within 3-standard deviations, however, there is a slight low-grade bias (Figure 12-24).

Figure 12-23: Standard CDN-GS-25 from the 2021 Drilling Campaign



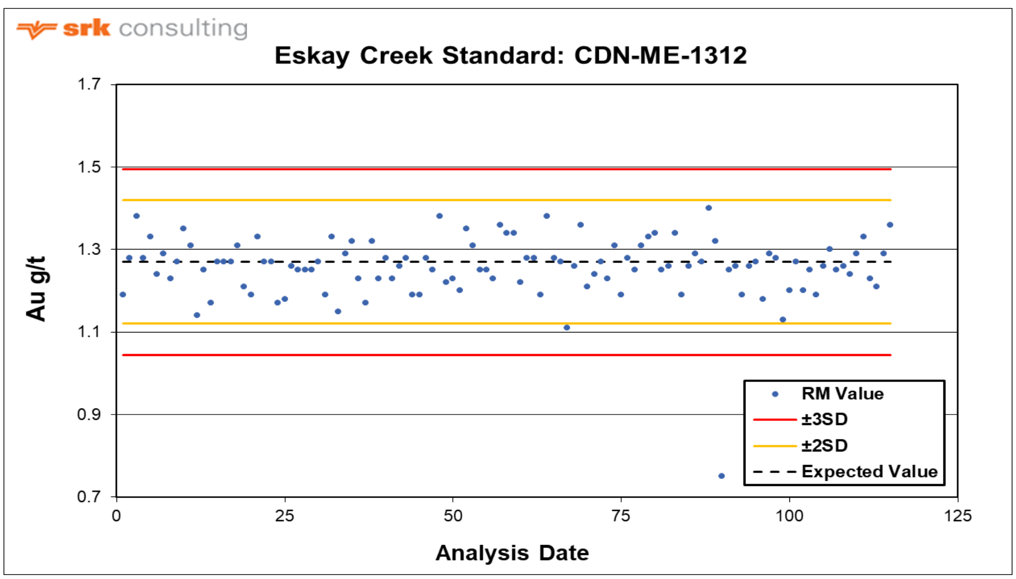
Note: Figure prepared March 17, 2022

Figure 12-24: Standard OREAS622 from the 2021 Drilling Campaign



Note: Figure prepared March 17, 2022

Figure 12-25: Standard CDN-ME-1312 from the 2021 Drilling Campaign

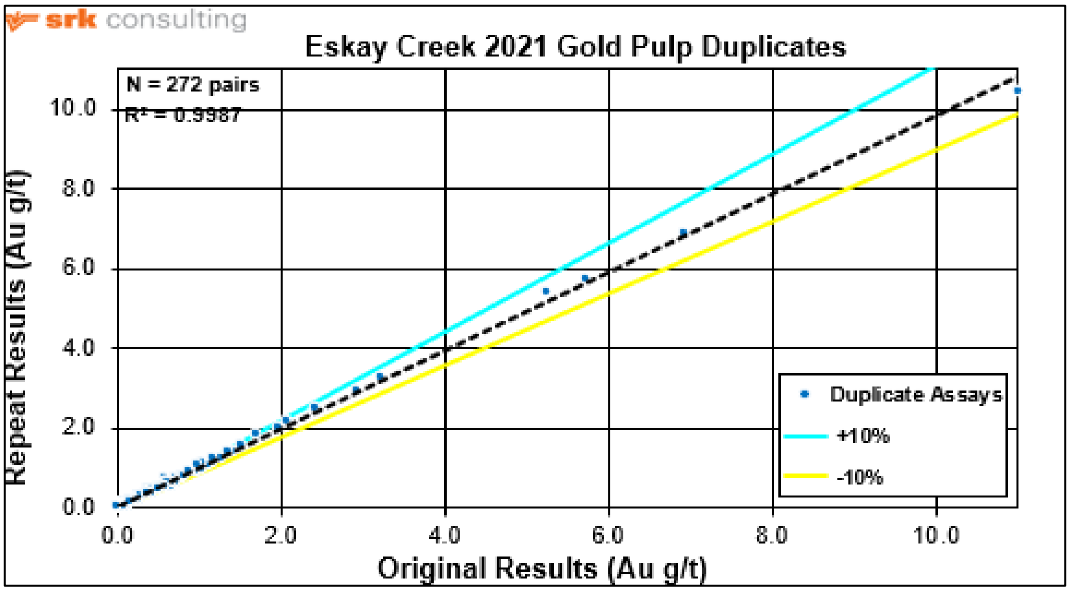


Note: Figure prepared March 17, 2022

Paired preparation and pulp data performed during the 2021 Phase 2 and Phase 3 drilling campaigns occurred within acceptable tolerance criteria at both lower-grade and higher-grade values (Figure 12-26 and Figure 12-27).

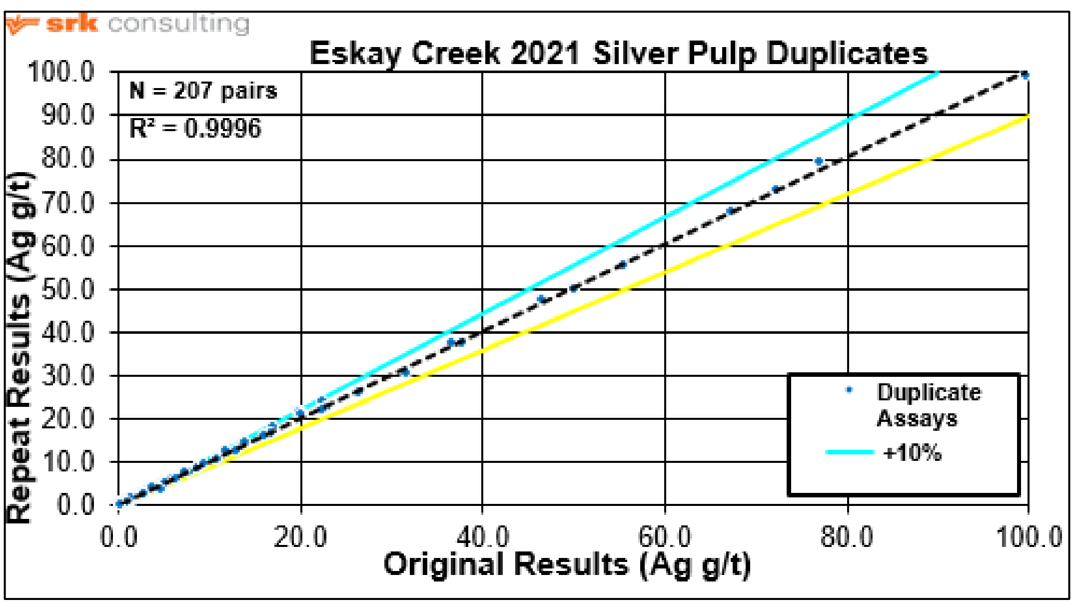
At the end of the 2021 Phase 3 drilling program, a random selection of approximately 2.5% of all assay samples, of which 1.5% occurred within moderate to higher gold grades, were selected and sent to SGS Canada. A total of 4 control blanks and 9 CRMs were processed at SGS, all of which performed within acceptable limits. A total of 120 pulp duplicates were checked against pulps originally processed at ALS Vancouver and the r-squared value was 0.9866; an acceptable result (Figure 12-28).

Figure 12-26: Gold Pulp Duplicate Samples from the Combined 2021 Drilling Campaigns



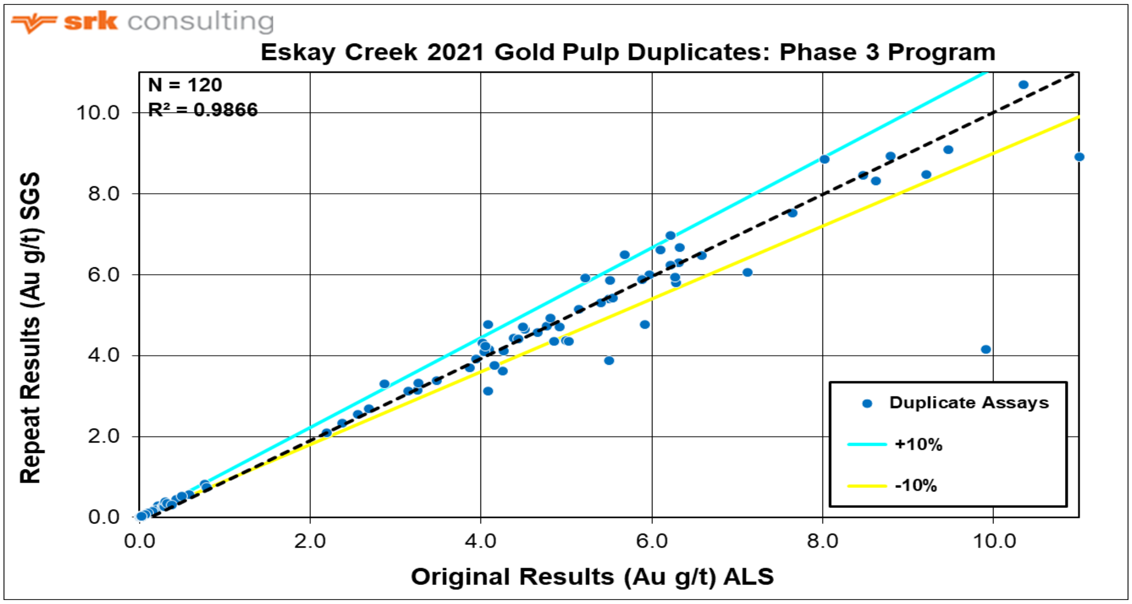
Note: Figure prepared March 17, 2022

Figure 12-27: Silver Pulp Duplicate Samples from the Combined 2021 Drilling Campaigns



Note: Figure prepared March 17, 2022

Figure 12-28: Gold Pulp Check Duplicate Samples from the 2021 Phase 3 Drilling Program



Note: Figure prepared May 18, 2022

12.1.5 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 2020/2021 Phase 2, and 2021 Phase 3 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

NOTE: This section includes summaries of previous technical reports – issued by Skeena following completion of the preliminary economic assessment (PEA) in 2020 and prefeasibility study in 2021 (Skeena 2020, Skeena 2021). A more thorough review of the most recent, feasibility testwork program is included in Section 13.4.

13.1 Background

As part of the 2019 PEA, testwork was completed by Blue Coast Research (Blue Coast) in Parksville B.C., 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 processes 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.

In 2021, Base Met continued their investigation of Eskay Creek material, based on 59 Variability composites from nine identified mineralised zones (including 109 and NEX zones, previously untested). As well as a more detailed variability testwork program on the modified, mill-float-mill-float (MF2) flowsheet, Base Met reviewed the target grind sizes (primary, regrind for rougher concentrate and secondary for deslimed rougher tailings). An additional bulk sample was pilot plant treated to generate sufficient sample mass for regrind mill evaluation and thickener/filtration testwork by Metso:Outotec .

The FS Variability testwork program identified Mudstone and Hangingwall material to perform differently from the main Rhyolite material, thereby requiring separate recovery estimates for the two lithologies. Following the PFS testwork program, these lithologies were previously grouped together.

Finally, additional filtration testing showed the need to include a concentrate dryer to ensure final moisture levels would be maintained below the expected Transportable Moisture Limit (TML).

13.2 2019 PEA Testwork

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

13.2.1 Sample Details

The drilling program in 2019 focused primarily on the 21A mineralised zone with auxiliary drill holes added in Zones 21C and 22. Table 13-1 summarises 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 (C_{org}).

13.2.2 Comminution

Comminution ore breakage testing on each sample consisted of SMC tests (which is used to determine the drop-weight index - DWi), as well as 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 competency and hardness with Mudstone being of low competency and moderately soft (DWi of 3.2 kWh/m^3, BWi of 13.0 kWh/t), while the 22 Zone exhibited BWi values as high as 26.4 kWh/t. The ore competency and hardness values for each sample and test type are summarised in Table 13-2

Table 13-2: Summary of 2019 PEA Comminution Testwork

Composite	Particle SG	DWi (kWh/m³)	Rwi (kWh/t)	BWi (kWh/t)
Hot	3.06	3.2	N/A	13.0
21A Low As	2.69	4.8	14.0	16.1
21A High As	2.69	4.7	15.2	16.2
21C	3.00	5.8	16.4	16.6
22 High As	2.62	7.3	21.0	23.5
22 Low As	2.59	5.9	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 (equivalent to DWi values between 5 and 12 kWh/m^3) with BWi results of 17 to 20 kWh/t, to an unreported closing screen size.

13.2.3 Bulk Flotation

A considerable number of open-circuit, rougher and rougher/cleaner float tests were conducted on all samples. 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 P80 grind sizes were tested (from 338 µm down to 39 µm) with ~60 µm used as the target P80 grind size for further float work. Rougher concentrate was also reground prior to cleaning, with a target P80 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.

Overall, flotation testwork on the selected samples 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.

13.3 2021 PFS Testwork

For the 2021 PFS, additional testwork was conducted by Base Met, with all results summarised 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 Rhyolite material.

13.3.1 Sample Origin & Composition

Head grades for all tested samples are included in Table 13-3, Table 13-4, and Table 13-5. 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 mineralised 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 13-3: 21A Low As Composite Assays

Composite	Cu (%)	Fe (%)	Ag (g/t)	Au (g/t)	As (g/t)	Sb (g/t)	S (%)	S^{-2} (%)
21A Low As	0.01	1.3	53	1.82	344	227	1.42	1.35

Table 13-4: Annual Composite Assays

Composite	Cu (%)	Fe (%)	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)	SO_4 (5)	S^{-2} (%)
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 13-5 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 to 5.7g/t Au, 4 to 512 g/t Au and a wide range of arsenic and antimony values. Sulphur grades ranged from 0.35 to 4.2%.

Table 13-5: PFS 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
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 7 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.

13.3.2 Mineralogy

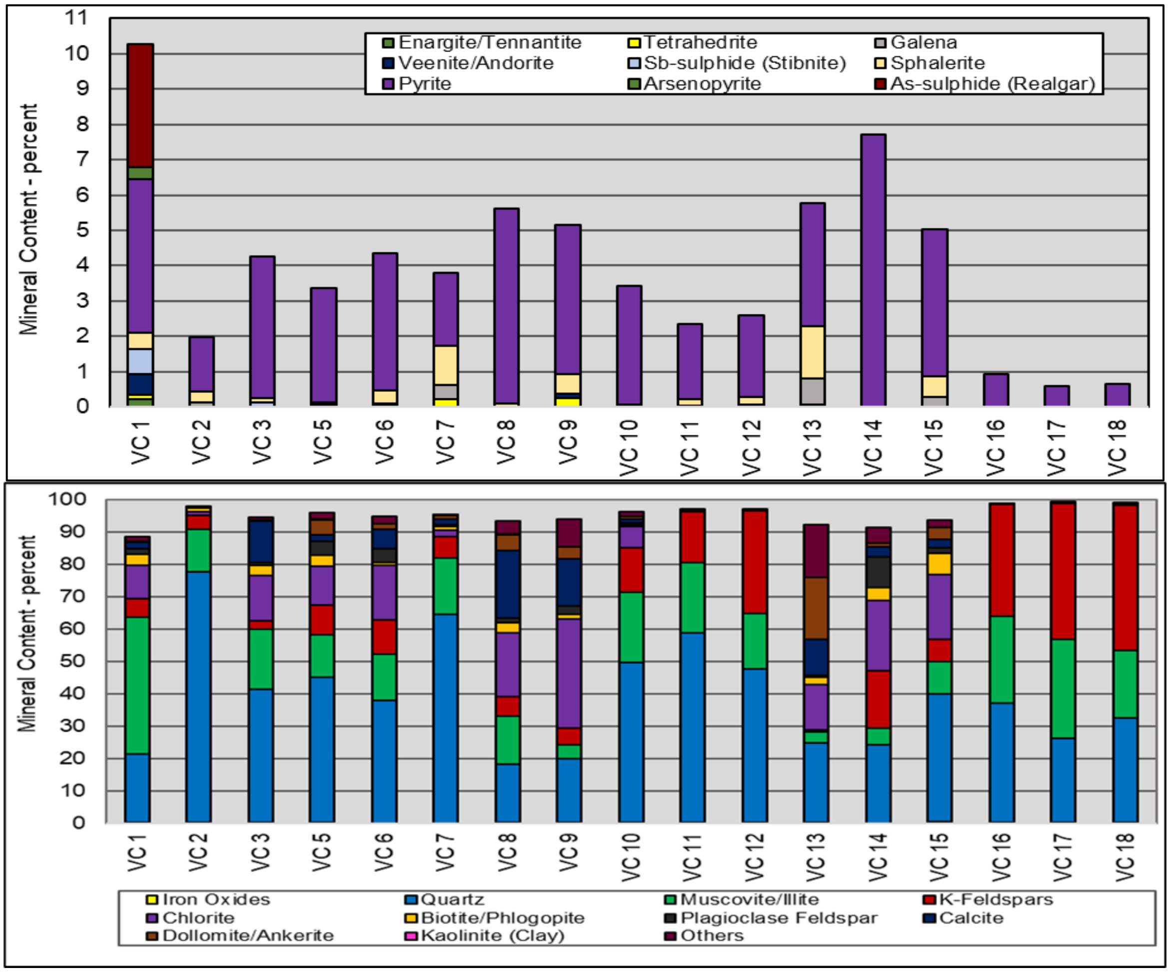
Quantitative mineralogy was obtained on the 21A Low As and three Annual composites using quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), with the modal compositions shown in Table 13-6. All four samples reported <3% pyrite with 36% to 47% quartz and varying amounts of NSG minerals (muscovite/illite, chlorite, biotite/phlogopite).

Table 13-6: 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
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 13-1. The upper chart shows sulphide minerals while the lower chart summarises 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 13-1: PFS Variability Composite Mineralogy (upper: Sulphides; lower: Non-sulphides)



Note: Figure prepared by Base Met, 2021.

13.3.3 Comminution

Comminution testwork was performed on blended composites, representing the main mineralised zones (Table 13-7). HW as well as 21A material was considerably less competent in terms of impact breakage (DWi value) while the 22 Zone sample showed again the highest BWi value (test performed using a closing screen size of 106 µm). The 21A sample was submitted to a number of BWi tests using different closing screens to observe the impact of grind size on grindability. No unusual response was noted with this sample, with a steady increase in kWh/t values for smaller closing screen size (15.7 to 17.3 kWh/t).

Table 13-7: Comminution Test Summary on Zone Composites

Zone	Particle SG	DWi (kWh/m³)	BWi				
			CSS (µm)	F_{80} (µm)	P_{80} (µm)	G_{pr}	BWi (kWh/t)
21A	2.69	4.27	75	1906	53	0.9	17.3
			106	1906	77	1.2	16.0
			150	1906	125	1.6	15.7
21E	2.79	7.10	106	2003	78	1.0	18.5
HW	2.69	3.66	106	1821	77	1.4	14.0
21B	2.65	8.31	106	2015	77	1.0	17.7
21C	2.83	5.88	106	2046	78	1.0	18.7
22	2.68	8.36	106	2276	81	0.6	27.6

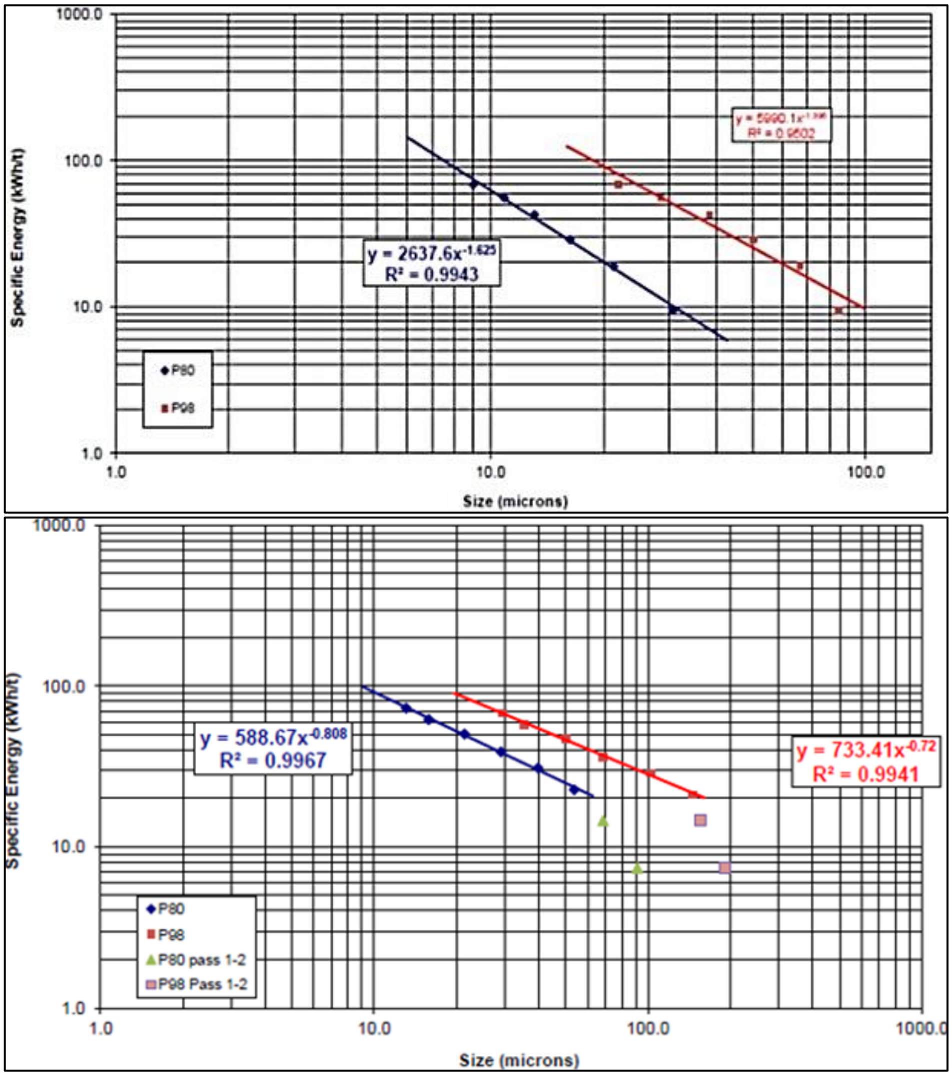
Bond abrasion index testing was done on the three Annual composites, with the results in Table 13-8showing a limited range and moderate abrasivity.

Table 13-8: 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" with a target primary grind P80 size of 100 µm generated sufficient mass of regrind and secondary grind circuit feed samples for IsaMill signature plot testing. In the modified flowsheet for the PFS, two streams require regrinding: rougher concentrate and rougher tailings after desliming. These two samples were submitted to IsaMill tests and the resulting Signature Plots are shown in Figure 13-2.

Figure 13-2: 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 P80 size of 66 µm (measured by sieve and cyclosizing methods) but P80 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 deslimed rougher tailings sample, sieve analysis reported a feed P80 size of 122 µm while laser sizing reported a P80 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. 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.

13.3.4 PFS Flotation Testwork

Table 13-9 summarises the list of separation tests conducted by Base Met (BL594 Base Met, 2021a) to assist with tables and figures shown in this subsection that references test numbers. Table 13-10 lists the tests done to investigate rejecting NSG minerals from the final concentrate (BL777 Base Met, 2021b).

Table 13-9: List of Separation Tests Conducted (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
87	New 21A	Cleaner
88	21A 2020	Cleaner
89 to 92	Annual Comps	Cleaner

Table 13-10: List of Additional Depressant Tests Conducted (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.

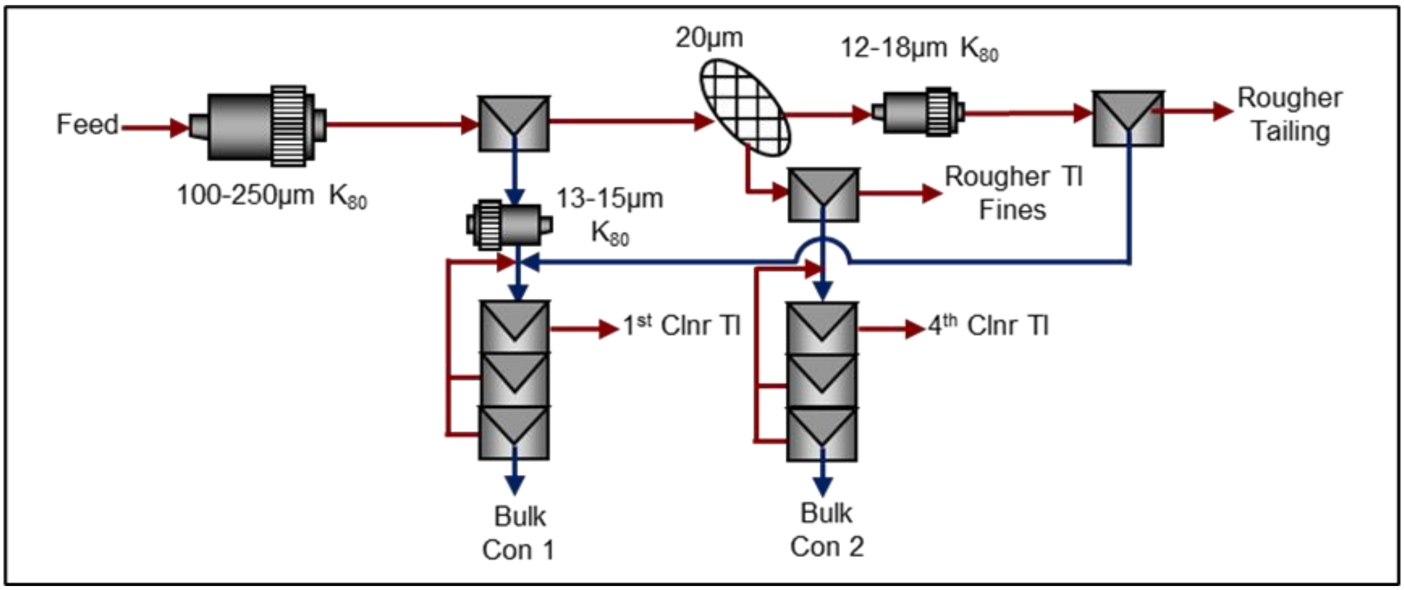
Tests 70 to 86, 89 to 92 were done on the original drill core samples referred to as Variability composites.

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 P80 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 P80 size was set to 100 µm.

The final, mill-float-mill-float (MF2) flowsheet selected for the 2021 PFS is shown in Figure 13-3 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 13-3: 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 13-3. 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.

13.3.5 Flotation Variability Testing

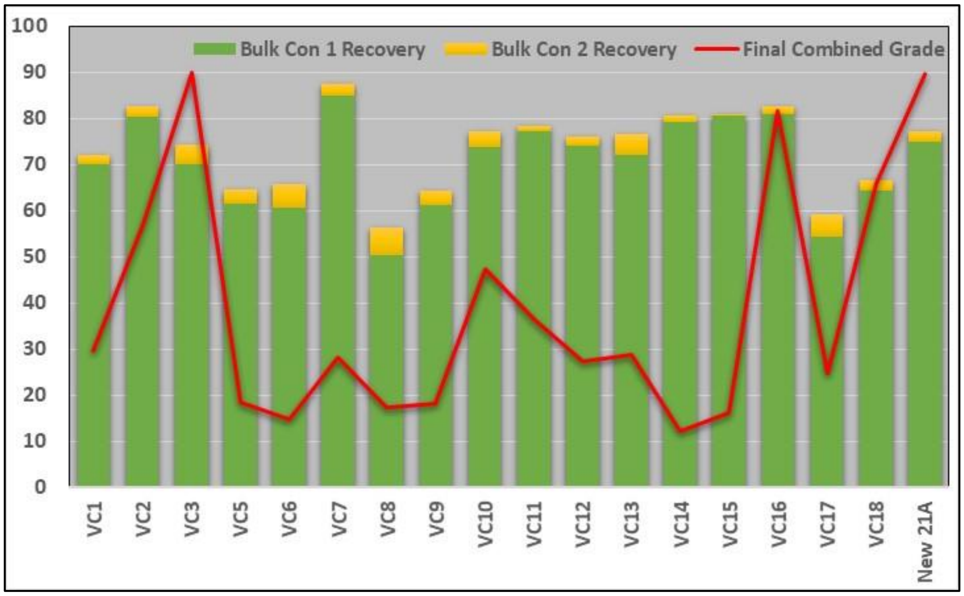
Each of the Variability composites were tested using the open circuit flowsheet shown in Figure 13-3. These drill core samples came from a range of zones, including 21B, 21E and HW not included in the 2019 PEA testwork program. These tests were all conducted at the design criteria, primary grind P80 size of 100 µm (see Table 13-11).

Table 13-11: PFS Variability Composite (VC) Open Circuit Cleaner Tests (Primary Grind P80 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
21A	New 21A	87	89.6	10147	10961	33596	31.6	77.2	97.8	73.0	92.7	76.7

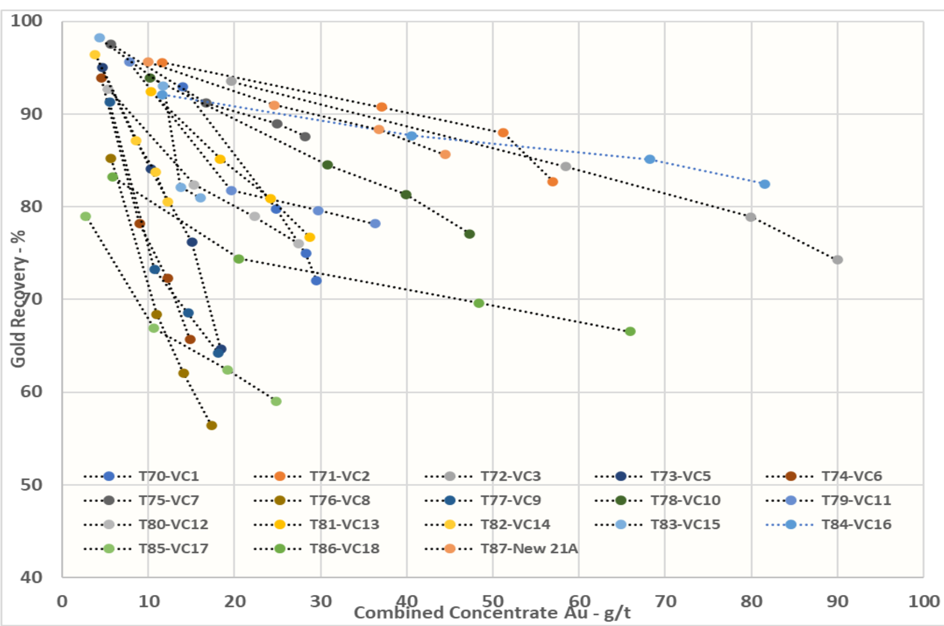
Figure 13-4 summarises the results showing the individual and combined concentrate grades and gold recovery. These results produced a range of recovery versus final concentrate grade curves (see Table 13-11 and Figure 13-5). Base Met observed a trend between the final concentrate grade and Au/(S+Fe) 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. 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.

Figure 13-4: Variability Composite Recovery and Combined Concentrate Grades



Note: Figure prepared by SRK, 2021.

Figure 13-5: Variability Composite Recovery vs. Concentrate Grade



Note: Figure prepared by SRK, 2021.

Additional investigation was done by Base Met to determine if some of these NSG minerals could be depressed. A review of the different mineralised 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.

13.3.6 Solid/Liquid Separation

A number of settling, pressure filtration and vacuum filtration tests were conducted by Base Met on concentrate samples generated from bulk testing as well as a number of Variability composites. The 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 13.3.1. Flocculant scoping test results are shown in Table 13-12 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 13-12: Bulk Test Final Concentrate Flocculant Scoping Tests

Sample	Flocculant	Flocculant Type	Final Density (% Solids, w/w)	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 flocculant dosage

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

Table 13-13: Final Concentrate Static Settling Tests

Sample ID	Dosage (g/t)	Initial Density (% Solids, w/w)	Final Density (% Solids, w/w)	Settling Rate (mm/s)
T88 Final Con	7.4	50	15.6	33.2
	11.0	50	15.3	30.1
	7.4	25	15.3	30.0
	7.4	75	15.3	34.2
	7.4	50	10.8	31.0
	11.0	75	15.3	30.9
T70-Final Con VC1	6.8	50	9.3	47.7
T72-Final Con VC3	7.3	50	9.2	37.4
T73-Final Con VC5	4.6	50	9.3	38.0

Sample ID	Dosage (g/t)	Initial Density (% Solids, w/w)	Final Density (% Solids, w/w)	Settling Rate (mm/s)
T74-Final Con VC6	6.0	50	9.3	37.1
T75-Final Con VC7	6.5	50	9.2	45.5
T76-Final Con VC8	5.1	50	9.2	38.5
T77-Final Con VC9	6.3	50	9.3	36.5
T78-Final Con VC10	5.3	50	9.0	42.9
T79-Final Con VC11	6.1	50	9.3	39.3
T80-Final Con VC12	5.7	50	9.2	37.9
T81-Final Con VC13	6.2	50	9.3	38.0
T82-Final Con VC14	4.4	50	9.1	59.9
T83-Final Con VC15	5.0	50	9.2	52.4
T87-Final Con New 21A	5.1	50	9.2	47.0

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 13-14).

Table 13-14: Bulk Test Final Concentrate Dynamic Settling Tests

Test Procedure	Loading Rate ($t/m^2/hr$)	Flocculant Dosage (g/t)	U/F Density (% Solids, w/w)	Overflow Solids (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 flocculant concentration

Pressure and vacuum filtration tests were completed on the bulk test final concentrate (Table 13-3 and Table 13-4). 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 13-15: Bulk Test Final Concentrate Pressure Filtration Tests

Sample	Sample Mass (g)	Blow Time (sec)		Cake Thickness (mm)	Cake Moisture (%)	Filter Rate (kg/m²/hr)
		Total	Filter Time			
Test 88 Final 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 13-16: Bulk Test Final Concentrate Vacuum Filtration Tests

Test	Feed Mass (g)	Final Moisture (%)	Filter Rate (kg/m²/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

13.3.7 Historical Recovery Estimates

From the 2021 PFS testwork results performed on a much wider range of sample compositions from different zones, a set of equations were used for metal recoveries. Note this recovery model is superseded by the recovery model shown is Section 13.4.11.

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 Feed [g/t]))

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 are estimated based on their relative upgrading to gold:

- 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 mineralised blocks missing gold, iron, or sulphur assays to categorize the material, the 2019 PEA recoveries were assigned. 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 13-5.

13.4 2022 FS Testwork

For the 2022 FS, additional testwork was conducted by Base Met, with all results summarised in one primary report (Base Met, 2022a) and 4 supplementary testing reports (Base Met, 2022b-e).

The objectives of this program were to evaluate the modified, MF2 flowsheet on a much wider range of Variability composites – including separate samples of Rhyolite, Mudstone and Hanging Wall lithologies. In addition, a bulk "MC Composite" sample was processed through a pilot plant to generate sufficient sample mass for regrind and secondary mill evaluation testing and additional thickener and filter testing. The supplementary test programs included additional testing on Mudstone samples, repeat open circuit cleaner and locked cycle tests on selected variability samples, as well as additional bench scale filtration tests.

The first part of the Base Met program was to confirm the grind size requirements of four Master composites (21A, 21B, 21C and Mudstone). A series of tests were performed on each composite, evaluating 100 µm and 150 µm primary grinds, 15 µm, 25 µm and 35 µm regrind sizes and 25 µm, 35 µm and 45 µm secondary grind sizes. In addition, deslime cutpoints of 20 µm, 38 µm and 53 µm were evaluated on each composite.

13.4.1 Sample Origin & Composition

The FS testwork program was based on 59 Variability composite (VC) samples collected from half core intervals of 2019 through 2021 drilling. From these VCs, mineralised zone and lithology composites were prepared for comminution testing as well as Master composites for each major zone for flotation testing (see Table 13-5 for Master composite assays). A Year 1–5 Blend composite was assembled for locked cycle test purposes during subsequent testing, these head assays are also reported below. The Blend composite was assembled from available variability samples to target expected feed grades and zone distributions.

Table 13-17: FS Master Composite Head Assays

Zone/Lithology	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)	Fe (%)	C_{org} (%)
21A	3.74	58	732	1846	1.60	1.14	1.28
21B	3.92	7	203	40	1.28	0.98	1.10
21C	1.54	34	103	70	0.74	0.63	0.61
21Be	4.05	29	425	102	2.14	1.81	1.87
MS	4.04	86	1220	232	3.24	2.85	1.60
Blend Year 1-5	4.31	102	1018	727	2.18	1.90	0.52

Head assays for the 59 Variability composites are shown in Table 13-18, with sample numbers from each mineralised zone collected in proportion to reserves tonnes and samples of both Rhyolite and Mudstone collected from the major mineralised zones.

Head grades ranged from 0.5 to 15.5 g/t Au, 1 to 1193 g/t Ag, 26 to 6054 g/t As, 14 to 6054 g/t Sb and 0.28 to 7.0% S. In addition, organic carbon (C_{org}) values up to 3%, specifically in the Mudstone and Hanging Wall samples.

Table 13-18: FS Variability Composite Head Assays

Comp ID	Lithology	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)	Fe (%)	C_{org} (%)
21A_MET_1	Rhyolite	7.75	147	6,054	472	1.39	1.23	0.027
21A_MET_3	Rhyolite	2.73	215	365	1,265	1.72	1.14	0.009
21A_MET_4	Rhyolite	6.27	1,193	631	9,685	3.14	1.58	0.03
21A_MET_5	Rhyolite	1.02	52	531	428	1.56	1.17	0.007
21A_MET_6	Mudstone	0.90	2	4,718	167	2.83	2.27	1.38
21A_MET_7	Rhyolite	7.21	20	894	10,135	1.84	1.11	0.01
21A_MET_8	Rhyolite	4.07	3	903	726	1.61	1.16	<0.01
21A_MET_9	Rhyolite	0.59	22	134	22	0.68	0.61	<0.01
21A_MET_10	Rhyolite	0.52	1	26	19	0.68	0.62	<0.01
21B_MET_1	Rhyolite	5.04	4	183	23	1.10	0.97	0.08
21B_MET_2	Rhyolite	1.80	2	41	34	0.42	0.42	<0.01
21B_MET_3	Rhyolite	2.56	2	72	21	0.59	0.59	<0.01
21B_MET_4	Mudstone	2.61	39	791	91	3.82	3.10	3.05
21B_MET_5	Rhyolite	1.38	1	160	14	1.47	1.18	<0.01
21B_MET_6	Rhyolite	2.86	5	212	42	1.24	0.98	<0.01
21B_MET_7	Rhyolite	1.13	2	191	24	1.01	0.90	<0.01
21B_MET_8	Rhyolite	1.13	4	190	28	1.11	0.94	<0.01
21B_MET_9	Mudstone	4.75	79	1,262	226	4.27	3.14	2.23
21B_MET_10	Mudstone	10.1	205	1,575	352	4.61	3.85	2.52
21B_MET_11	Rhyolite	0.99	8	345	38	1.14	1.07	<0.01
21B_MET_12	Rhyolite	1.48	3	76	15	0.73	0.67	<0.01
21B_MET_13	Rhyolite	3.50	25	289	70	1.56	1.20	<0.01
21B_MET_14	Mudstone	0.93	9	365	39	1.99	1.56	0.461
21Be_MET_1	Rhyolite	9.46	67	500	163	1.79	1.33	<0.01
21Be_MET_2	Rhyolite	0.76	8	201	36	0.78	0.81	0.019
21Be_MET_3	Rhyolite	7.69	49	1,869	108	7.04	6.75	0.011
21Be_MET_4	Rhyolite	0.46	114	74	179	0.57	0.53	<0.01
21Be_MET_5	Rhyolite	2.81	23	221	88	0.77	0.93	<0.01
21Be_MET_6	Rhyolite	2.37	138	279	1,562	1.51	1.16	<0.01
21Be_MET_7	Rhyolite	0.75	8	94	65	1.04	0.86	<0.01
21C_MET_1	Mudstone	5.55	18	517	105	2.09	1.42	1.18
21C_MET_2	Rhyolite	0.85	1	32	16	0.46	0.39	<0.01

Comp ID	Lithology	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)	Fe (%)	C_{org} (%)
21C_MET_3	Rhyolite	1.25	6	245	36	1.03	0.70	<0.01
21C_MET_4	Rhyolite	0.65	68	54	213	0.71	0.56	<0.01
21C_MET_5	Rhyolite	1.58	28	29	61	0.50	0.45	<0.01
21C_MET_6	Rhyolite	0.73	1	188	31	1.35	1.09	0.045
21C_MET_7	Rhyolite	0.86	90	144	125	0.69	0.52	<0.01
21C_MET_8	Mudstone	2.95	164	364	337	3.07	2.78	0.424
21C_MET_9	Mudstone	1.04	7	662	113	2.50	1.98	0.069
21C_MET_10	Rhyolite	3.72	3	185	38	1.18	0.91	0.022
21C_MET_11	Mudstone	1.21	6	623	148	2.39	1.84	0.569
21C_MET_12	Rhyolite	2.40	19	340	49	0.95	0.78	<0.01
21E_MET_1	Rhyolite	2.08	36	807	469	1.44	1.08	0.204
21E_MET_2	Rhyolite	1.58	4	2,206	110	0.79	0.66	0.024
21E_MET_3	Rhyolite	0.99	2	53	29	0.28	0.46	0.018
21E_MET_4	Rhyolite	2.53	45	95	194	0.36	0.41	0.018
22_MET_1	Rhyolite	3.85	69	290	98	1.06	0.91	0.014
22_MET_2	Rhyolite	0.97	19	3,061	87	0.43	0.45	0.017
22_MET_3	Rhyolite	2.93	5	2,622	1,982	1.23	0.78	0.013
22_MET_4	Rhyolite	0.78	41	1,076	189	0.66	0.46	0.016
HW_MET_1	Hangingwall	15.5	329	314	3,038	2.90	2.75	1.38
HW_MET_2	Hangingwall	2.81	314	596	1,010	1.63	1.39	1.45
HW_MET_3	Hangingwall	1.54	58	946	584	5.49	5.00	2.79
HW_MET_4	Hangingwall	0.60	8	380	38	3.55	3.30	0.827
109_MET_1	Rhyolite	9.87	116	435	159	4.69	4.20	0.261
NEX_MS01	Mudstone	5.32	293	603	415	2.61	2.38	0.712
NEX_RHY02	Rhyolite	3.08	42	511	349	4.79	3.35	0.145
NEX_MS03	Mudstone	3.94	71	365	171	3.54	2.69	0.422
NEX_RHY04	Rhyolite	2.24	28	329	186	2.92	2.40	0.282

13.4.2 Mineralogy

Quantitative mineralogy was measured on all Master and Variability composites using quantitative evaluation of materials by scanning electron microscopy (QEMSCAN), with the modal compositions shown in Table 13-19. Higher arsenopyrite and sphalerite levels were detected in the 21A and Mudstone composites, while pyrite levels ranged from 1.9 to 7.8% (Mudstone sample). Sericite/muscovite ranged from 22 to 49% with clays higher in the Mudstone sample at almost 5%. Barite was also observed in the 21Be and Mudstone samples.

Table 13-19: FS Master Composite Modal Mineralogy

Mineral	Mineral Composition (Weight %)				
	21A Master	21B Master	21C Master	21Be Master	Mudstone
Chalcopyrite	0.02	0.01	0.02	0.01	0.03
Sphalerite	0.15	0.10	0.09	0.04	0.44
Stibnite	0.17	0.00	0.00	0.00	0.00
Galena	0.07	0.02	0.02	0.02	0.07
Arsenopyrite	0.13	0.00	0.01	0.01	0.12
Pyrite	3.01	2.60	1.86	5.44	7.76
Quartz	42.4	44.0	57.8	43.6	45.1
Plagioclase	0.25	0.18	0.16	1.22	1.56
K-Feldspar	0.63	5.89	15.1	8.92	1.10
Biotite/Phlogopite	1.19	0.41	0.13	4.13	0.79
Sericite/Muscovite	49.2	44.9	22.0	27.8	28.3
Chlorite	0.46	0.11	0.08	1.43	1.24
Clays	1.50	1.22	1.67	1.81	4.83
Other Silicates	0.18	0.20	0.18	0.68	1.64
Barite	0.02	0.02	0.03	1.36	0.40
Other	0.56	0.37	0.83	3.52	6.58

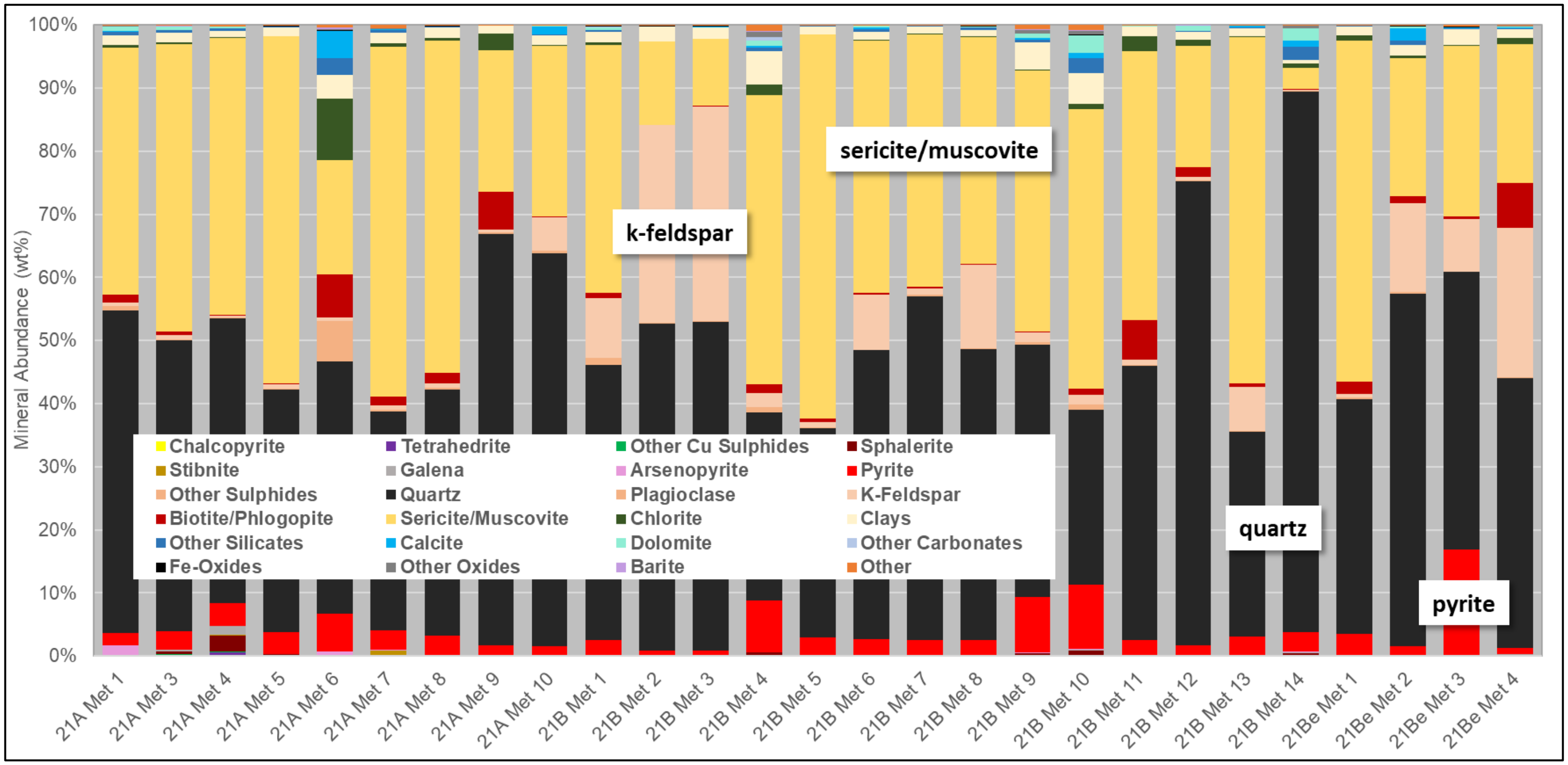
The mineralogy results for the Variability composites are shown graphically in Figure 13-6 and Figure 13-7.

The nine 21A zone samples were relatively consistent, with two including significant levels of biotite/phlogopite and chlorite. Fourteen 21B zone samples and seven 21Be zone samples showed high quartz at times, with occasional biotite/phlogopite and pyrite in a few samples. Twelve 21C zone samples showed elevated quartz in general, while four 21E zone samples were mixed in composition. The four 22 zone samples (noted for their higher BWi values) were comprised mainly of quartz, k-feldspar and sericite/muscovite.

The four HW zone samples were distinctly different from the other zones, but also variable in composition; two of the HW samples were high in pyrite. All four NEX zone samples were very high in pyrite and quartz–despite being split between Mudstone and Rhyolite lithologies.

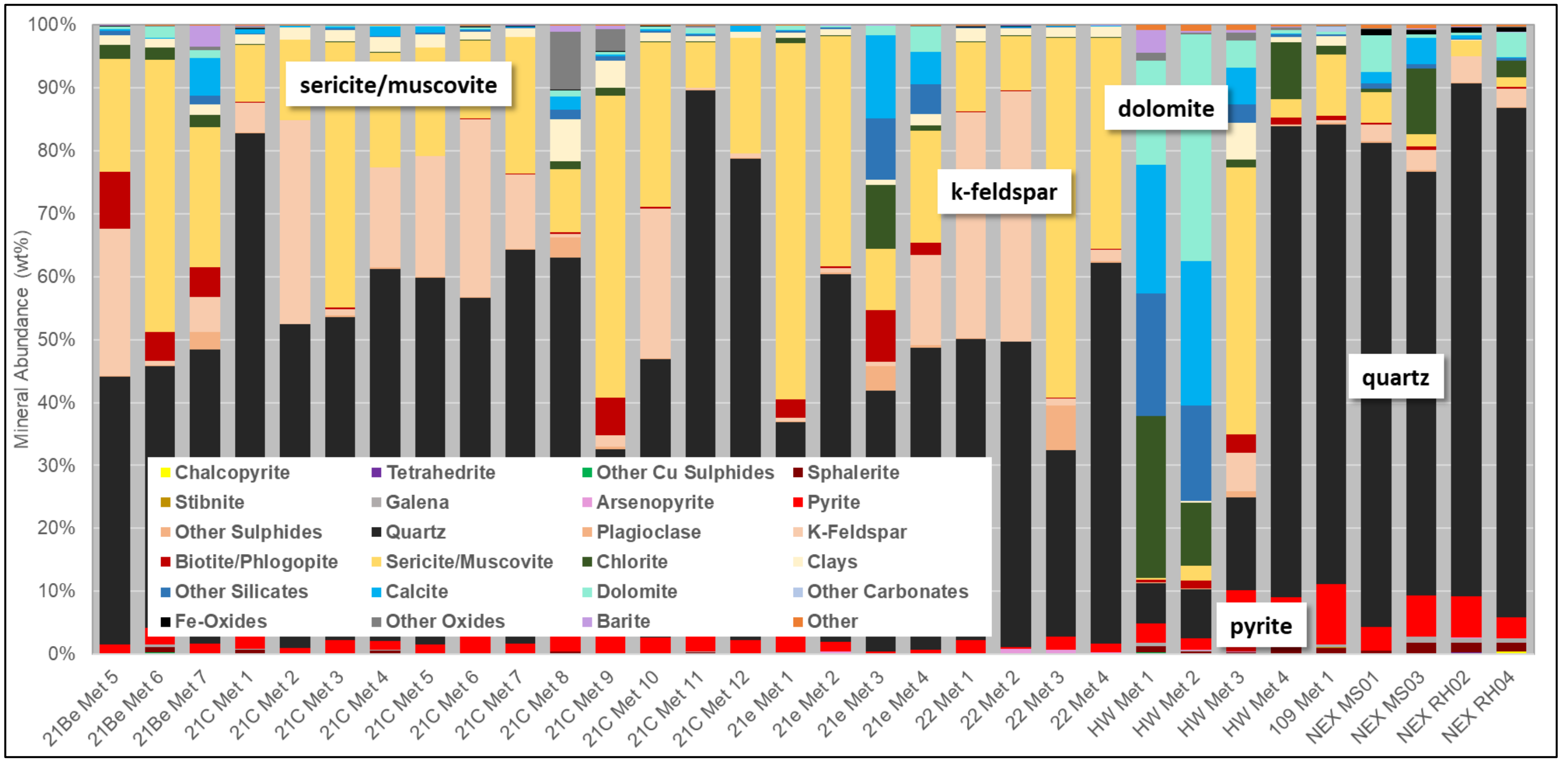
Overall, there is a lack of consistent mineralogy within each of the major mineralised zones; however, NEX and Hanging Wall zones are quote different from the remainder – and this is evident in the flotation results. While Mudstone samples showed a consistently higher Corg assay, modal mineralogy was similar to Rhyolite samples except for the higher pyrite (>5% in general).

Figure 13-6: FS Variability Composite Mineralogy (Part 1)



Source: Base Met, 2002.

Figure 13-7: FS Variability Composite Mineralogy (Part 2)



Source: Base Met, 2002.

13.4.3 Comminution

From the Variability composites, mineralised zone Master composites were prepared for comminution testing (see Table 13-20).

Table 13-20: Comminution Test Summary on FS Zone Composites

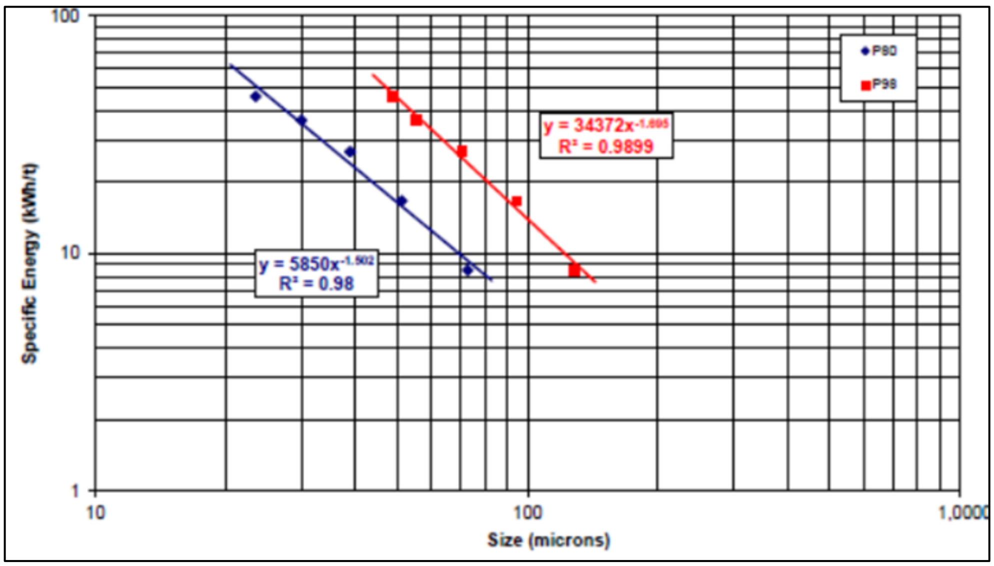
Zone	Lithology	SG	DWi (kWh/m³)	Rod Work Index			Ball Work Index @ 106µm			Abrasion
				F_{80} (µm)	P_{80} (µm)	Rwi (kWh/t)	F_{80} (µm)	P_{80} (µm)	BWi (kWh/t)	Ai (g)
21A	Rhyolite	2.70	6.8	9153	975	17.1	2451	80	19.0	0.365
21A	Rhyolite						2745	76	15.5	0.240
21A	Rhyolite	2.70	4.5	8818	989	14.3	2428	77	15.3	0.067
21A	Mudstone						2817	75	21.1	0.095
21B	Rhyolite	2.61	8.5	9296	970	20.3	2331	81	20.2	0.743
21B	Mudstone	2.71	4.9	8529	958	15.5	2330	77	16.9	0.059
21B	Rhyolite						2768	76	16.1	0.135
21B	Mudstone						2908	77	24.0	0.560
21Be	Rhyolite						2672	76	16.9	0.198
21Be	Rhyolite	2.65	7.7	9370	953	18.0	2334	79	20.4	0.453
21C	Rhyolite	2.64	8.5	9627	959	17.9	2403	81	20.5	0.562
21C	Rhyolite	2.66	9.5	9609	938	18.5	2376	79	16.7	0.570
21C	Mudstone						2855	77	20.7	0.468
21C	Rhyolite						2851	78	16.9	0.368
21E	Rhyolite	2.67	8.5	9283	937	17.3	2353	79	18.7	0.232
21E	Rhyolite						2518	78	15.9	0.121
22	Rhyolite	2.65	7.8	9162	969	18.9	2468	79	20.5	0.462
22	Rhyolite						2870	78	23.8	0.941
HW	HW						2440	77	14.1	0.005
HW	HW	2.80	8.2	9131	944	19.4	2308	77	19.0	0.205
109	Rhyolite						2856	77	16.3	0.508
NEX	Mudstone	2.70	8.9				2403	80	20.2	0.749
NEX	Rhyolite	2.71	8.6				2325	79	22.7	0.839

Where available, samples of both Rhyolite and Mudstone were tested from each zone with Mudstone returning consistently high impact resistance (high DWi values) and BWi values. Pairs of SMC and RWi tests were done but not on every sample.

Abrasion testing was done on all samples and ranged from very low (HW sample) to very high for both Mudstone and Rhyolite samples while sample specific gravity (estimated during SMC tests) fell across a narrow range (2.6 to 2.8).

Following the IsaMill Signature Plot test done by ALS during the PFS program, a second deslimed rougher tailings sample was tested by SGS (SGS Canada, 2022). The secondary grinding mill feed was generated from the MC Composite processed by the pilot plant, with the Signature Plot results shown in Figure 13-8. The SGS sample was somewhat finer than the ALS sample, with an F_{80} size of 105 µm compared with the earlier sample F_{80} of 145 µm. (The MC Composite tailings sample also included more -10 µm material at 11% compared with the ALS sample with 5%.)

Figure 13-8: IsaMill Signature Plot (Rougher Tail after Deslime)



Note: Figure prepared by SGS Canada for Base Met, 2022.

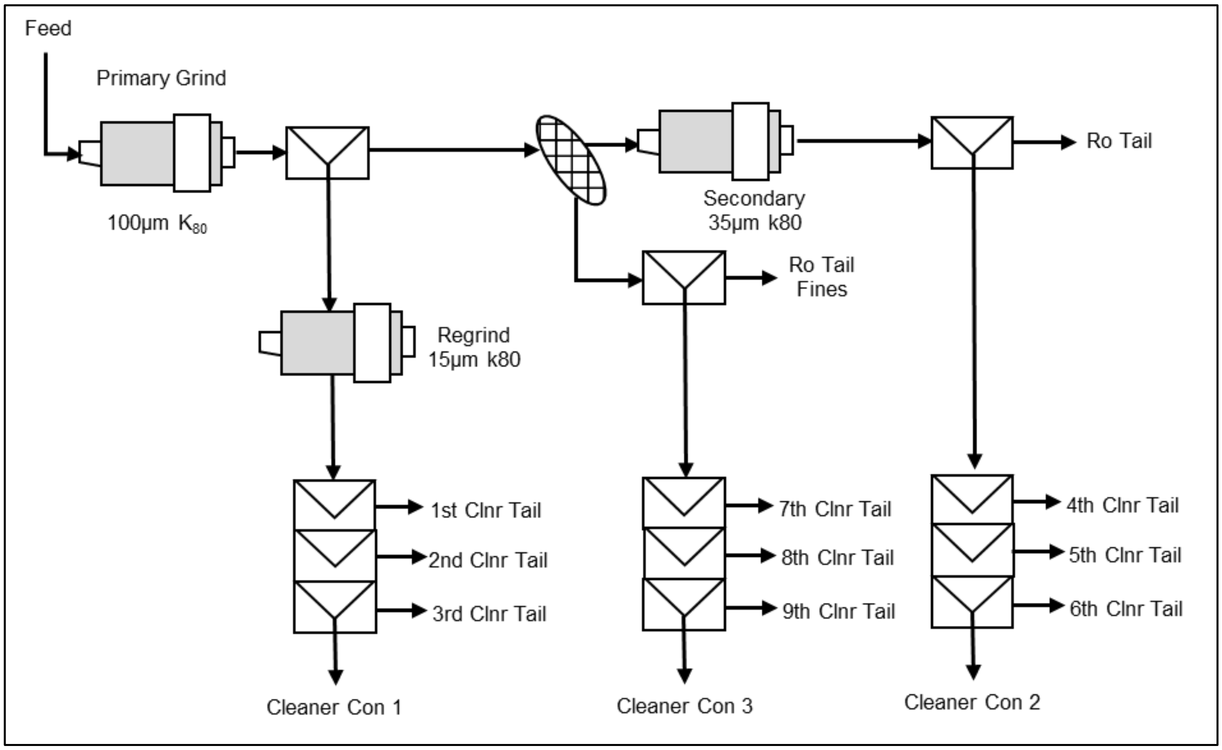
The SGS test was conducted with 4.5 mm ceramic media and did not note any issues, running at 39% w/w solids. To achieve the target secondary grind P80 size of 35 µm, the SGS test indicated a specific energy requirement of 28 kWh/t. The earlier ALS test done with 2.8 mm media indicated 33 kWh/t would be required to achieve a P80 size of 35 µm with the feed needing to be diluted to avoid viscosity issues.

The latest process design criteria incorporate IsaMills for both regrind and secondary grinding duties (the latter in series with a ball mill). This is a change from the PFS design which utilised HIGmills for these duties. HIGmill grinding tests were completed by SGS on samples of both rougher concentrate and deslimed rougher tailings (generated from the MC Composite pilot run). However, the HIGmill test results were not consistent with expectations or that reported in the IsaMill tests. As the final design flowsheet includes IsaMills, the HIGmill test results are not included in this summary report.

13.4.4 Master Composite Flotation

Each of the Master Composites were tested under a range of primary (100 to 150 µm), regrind (15 to 35 µm) and secondary (25 to 45 µm) grind sizes. The MF2 flowsheet used for this variability test program is shown in Figure 13-9.

Figure 13-9: MF2 Flowsheet for FS Variability Testwork



Note: Figure prepared by Base Met, 2022.

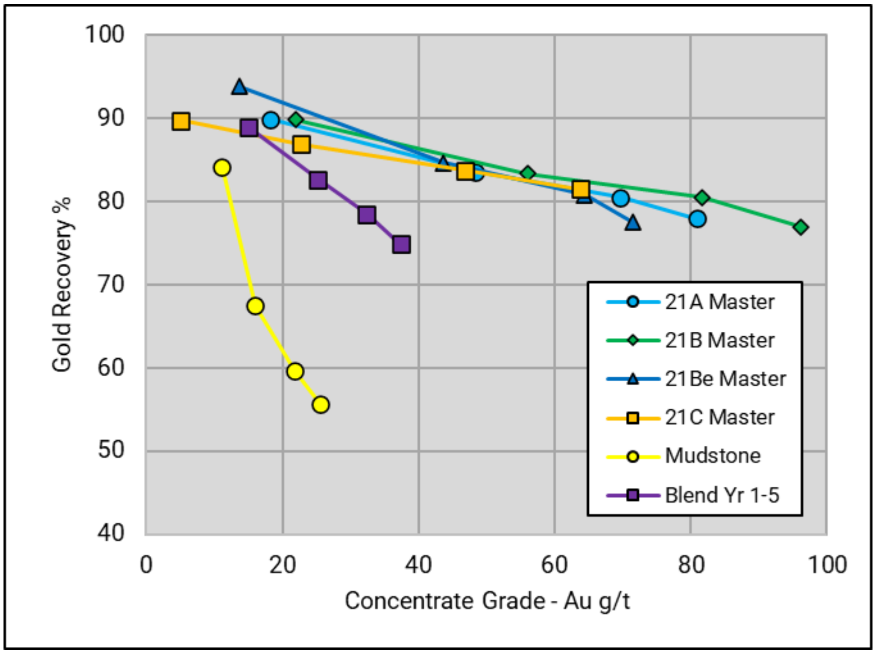
One of the observations of this investigation was the main mineralised zones were all quite insensitive to primary grind. However, due to expected power requirements for the secondary grinding circuit, the primary target grind P80 size was maintained at 100 µm. The regrind P80 size was held at 15 µm to avoid gold losses while the secondary grind P80 was coarsened somewhat to 35 µm. The results in Table 13-21 show Master Composite performance under these target grind sizes. The open circuit cleaner test results for the Year 1–5 Blend Composite is included in this table; however, this sample was only tested at the final grind conditions.

Table 13-21: FS Master Composite Overall Performance Under Final Grind Size Conditions

Composite	Overall Performance – Combined Concentrates									
	Grade (% or g/t)					Recovery (%)				
	Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
21A Master	80.9	1,634	15,361	49,941	27.6	77.9	91.7	75.9	94.7	74.8
21B Master	96.1	234	5,155	977	30.2	76.9	78.4	72.1	69.2	78.3
21Be Master	71.5	433	7,780	1,329	40.4	77.4	78.9	75.9	69.0	77.6
21C Master	63.7	1,133	3,923	2,245	30.2	81.5	90.2	60.2	76.5	71.1
Mudstone	25.6	751	7,103	1,677	18.3	55.6	75.0	47.3	64.4	50.7
Blend Year 1-5	37.4	968	7,034	6,563	18.7	74.9	88.5	65.3	88.5	72.2

The cleaner circuit performances for these tests are shown in Figure 13-10. Four of the Rhyolite samples from the main mineralised zones performed very similarly, with flat recovery vs. grade curves and final concentrate grades >60 g/t Au. The Mudstone sample was significantly different with high gold losses and diluted concentrate grade after cleaning.

Figure 13-10: FS Master Composite Cleaner Performance (Recovery vs. Concentrate Grade)



Note: Figure prepared by Ausenco, 2022

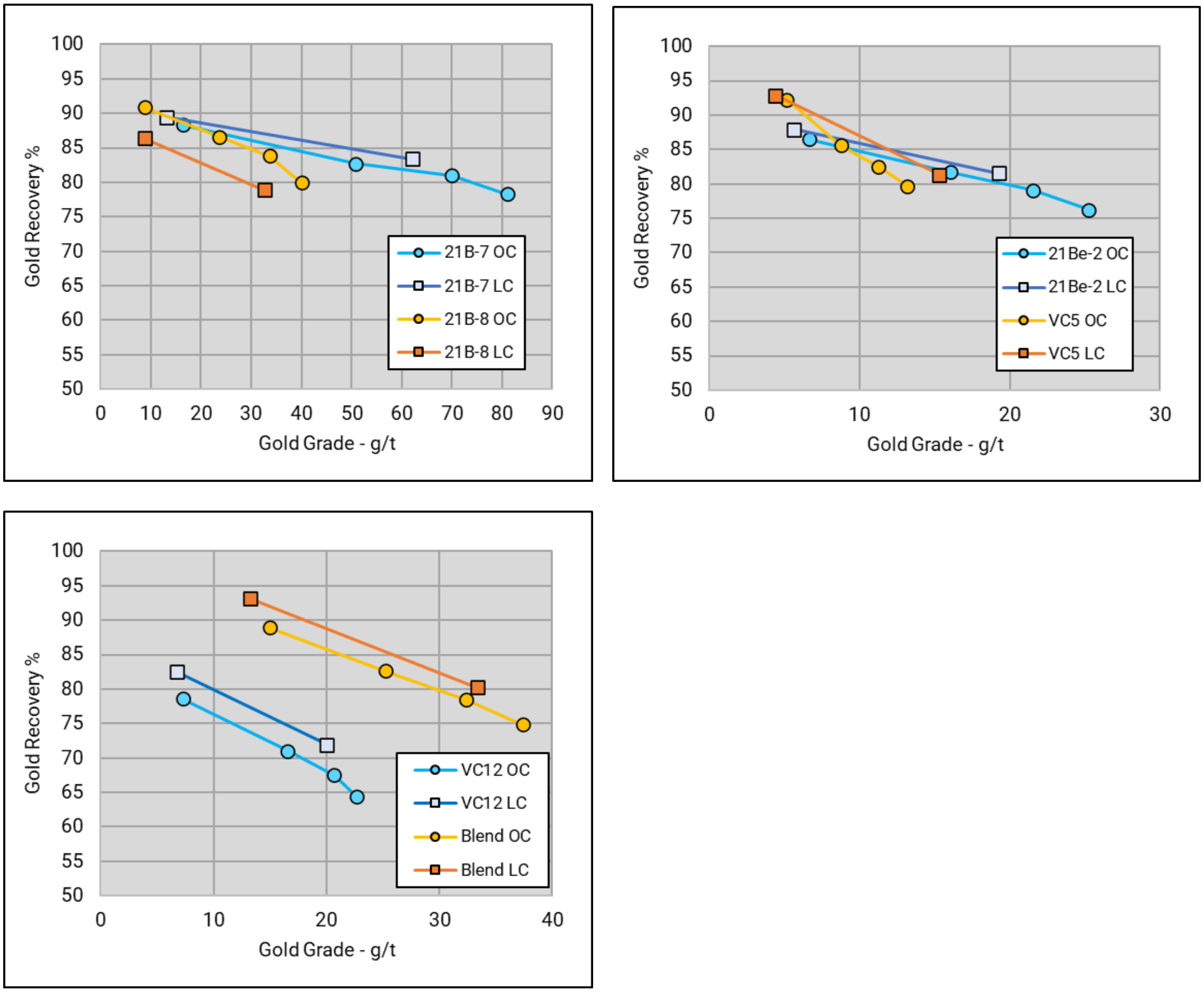
Base Met noted in their report the Mudstone sample had 8 to 90 times greater C_{org} in the three concentrates compared with the other composite samples. Work was completed to address the different performance of the Mudstone material – namely coarser grind and regrind sizes, methods to avoid C_{org} from impacting the concentrate grade (e.g. prefloat and/or depressants), and additional collector – see discussion in Section 13.4.7.

13.4.5 Variability Testing

Following confirmation of the primary, regrind and secondary grind sizes, the MF2 flowsheet was tested on 57 of the Variability composite samples. Two samples (21B Met 9, 21C Met 8) had limited mass after assembling other composites. It should be noted that due to overall sample mass limitations, a single test was conducted on each sample. Primary, secondary and regrind times were often estimated based on reference data, resulting in some variances from target sizes.

After reviewing the initial results of the variability samples, 18 samples that had sufficient remaining sample mass were selected for repeat tests. The repeats were conducted to provide more collector on tests with high cleaner circuit losses, as well as to achieve secondary/regrind product size targets where initial grind time estimates were not sufficient. Three of these samples were also selected for locked cycle tests to confirm closed circuit performance. During this supplementary testing, repeat tests were also conducted on 6 variability samples from the PFS program, two of which were tested using the locked cycle protocol. A locked cycle test was also completed on the Blend Year 1–5 composite. Gold performance for the locked cycle and respective repeat open circuit tests is shown in a series of graphs in Figure 13-11.

Figure 13-11: FS Locked Cycle and Open Circuit Performance (Recovery vs. Concentrate Grade)



Note: Figures prepared by Ausenco, 2022.

Except for sample VC12, the final grade-recovery result for the locked cycle tests was similar to the respective open circuit 2nd cleaner point. Generally, gold recoveries across the cleaner circuits were similar between the two test protocols, final gold recoveries were mostly affected by differences in the rougher circuit recoveries.

Due to the favourable economics of higher gold recoveries at somewhat lower concentrate grades, combined with the locked cycle test observations, variability sample open circuit test data is summarised at the 2nd cleaner concentrate point.

The combined concentrate and recoveries for the main elements are shown in Table 13-22. Tests that were repeats are designated by the letter R. As shown in Figure 13-9, three streams make up the combined final concentrate, following three stages of cleaning on each stream.

In Figure 13-12 and Figure 13-13 (two parts), the recovery to the three streams is shown as stacked columns, with the regrind cleaner circuit achieving the bulk of the recovery (green column). Recovery of gold from the deslime screen undersize ('fines') cleaning circuit (blue column) is generally quite low, but at times, was found to be sufficient to support the inclusion of this circuit in the flowsheet. A few tests used the 2 concentrate flowsheet, so the secondary concentrate (Con 2) is combined with concentrate 1.

Table 13-22: FS Variability Composite Open Circuit MF2 Cleaner Tests (Primary Grind P_{80} of 100 µm)

Variability Comp ID	Test #	Combined 2nd Cleaner Concentrates										
		Mass Pull (%)	Grade (% or g/t)					Recovery (%)				
			Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
21A_MET_1	26	6.1	185	1750	83443	6030	16.0	84.2	82.9	83.5	81.8	70.2
21A_MET_3	25	5.8	49	3059	4384	23088	24.6	81.2	96.2	78.9	97.5	84.1
21A_MET_4	28	10.3	65	10527	4620	90056	28.5	91.1	98.4	80.8	98.2	91.1
21A_MET_5	15	7.1	26	705	5629	6542	18.4	86.4	89.6	76.1	95.0	76.8
21A_MET_6	27 R	17.1	5	16	16445	604	9.4	63.4	74.2	63.5	72.7	61.3
21A_MET_7	20	7.2	91	205	9846	123011	19.5	78.8	74.5	79.3	95.8	84.9
21A_MET_8	21	5.6	80	46	12582	12787	20.7	74.5	64.1	80.6	85.0	79.6
21A_MET_9	16	3.5	44	517	3092	514	17.0	81.9	87.2	78.9	78.1	75.4
21A_MET_10	18	3.8	19	27	601	399	16.0	81.8	87.9	78.5	68.5	79.0
21B_MET_1	22	3.4	143	95	3468	611	23.7	74.6	79.3	69.6	69.0	75.4
21B_MET_2	29	1.6	55	628	1455	3627	17.9	46.5	89.9	49.5	82.4	72.9
21B_MET_3	31 R	5.0	51	28	855	290	9.6	61.4	44.0	70.6	54.6	77.1
21B_MET_4	23 R	13.9	12	157	2624	369	13.6	60.5	53.8	52.3	47.5	55.5
21B_MET_5	17	6.0	25	16	1999	157	21.6	87.0	79.3	77.7	66.9	85.1
21B_MET_6	30	3.7	117	136	4457	971	27.7	74.8	81.3	75.8	74.1	82.3
21B_MET_7	33 R	4.0	70	52	3238	375	22.3	81.0	73.4	80.9	66.3	85.3
21B_MET_8	24 R	4.3	34	72	3509	499	24.9	83.9	82.1	77.4	70.4	86.9
21B_MET_10	19	10.2	43	1553	4521	2731	18.5	48.2	79.8	32.0	67.3	42.4
21B_MET_11	32	4.3	22	151	4952	641	19.6	76.9	77.3	63.4	65.2	71.7
21B_MET_12	34	2.6	71	118	2560	410	24.5	87.6	83.4	83.4	58.0	83.0
21B_MET_13	35	6.2	51	393	3305	1071	20.1	81.5	80.7	69.9	75.1	76.2
21B_MET_14	36 R	13.2	5	53	1498	223	9.2	63.5	71.8	58.9	66.5	62.2

Variability Comp ID	Test #	Combined 2nd Cleaner Concentrates										
		Mass Pull (%)	Grade (% or g/t)					Recovery (%)				
			Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
21Be_MET_1	37	4.5	211	1287	6637	3071	29.4	84.3	86.6	72.0	76.7	78.3
21Be_MET_2	38 R	4.4	22	138	2765	524	12.2	79.1	75.3	65.7	53.1	75.4
21Be_MET_3	39	13.7	54	296	10770	608	42.8	92.5	87.7	87.6	78.9	92.3
21Be_MET_4	40	1.7	24	5316	3064	9037	21.2	57.1	84.4	62.2	80.8	71.5
21Be_MET_5	41	2.3	131	720	6691	2658	22.4	76.9	68.0	63.8	57.5	74.7
21Be_MET_6	42	4.6	58	3040	5709	35258	28.0	77.5	91.0	75.3	94.3	82.8
21Be_MET_7	43	2.9	27	266	2432	2228	20.2	72.1	71.5	65.1	71.6	52.4
21C_MET_1	44	6.7	44	150	2440	638	13.3	54.7	62.2	34.4	40.5	41.8
21C_MET_2	45 R	2.3	37	63	879	579	14.5	66.9	86.0	61.2	62.5	73.1
21C_MET_3	46	4.4	34	79	3477	435	16.8	77.9	63.5	61.7	45.6	72.7
21C_MET_4	47	3.0	18	2030	779	10220	18.6	67.9	97.0	63.6	95.9	77.8
21C_MET_5	48	1.7	79	1564	1217	3306	20.7	79.3	92.8	68.3	85.0	71.2
21C_MET_6	49 R	6.7	13	15	1992	375	17.3	82.4	25.3	81.6	66.2	86.8
21C_MET_7	50 R	2.8	35	2539	3673	4483	20.6	89.0	94.6	74.7	91.3	84.7
21C_MET_9	51	8.5	10	53	5489	915	24.0	77.7	81.5	75.8	65.8	80.4
21C_MET_10	52	4.4	61	53	2654	688	23.1	73.1	83.0	72.5	72.4	82.1
21C_MET_11	53 R	15.2	7.1	30	2748	935	13.4	85.1	79.2	84.1	81.2	86.8
21C_MET_12	54	3.1	72	407	6983	1050	22.8	80.2	78.4	75.0	69.5	74.2
21E_MET_1	55	3.5	58	886	15208	15860	33.8	79.9	84.0	69.4	89.3	79.9
21E_MET_2	56	3.5	47	75	41297	2078	18.3	75.7	69.1	64.2	69.2	71.0
21E_MET_3	57	1.2	71	93	1871	1211	13.1	71.0	54.3	64.1	42.5	56.8
21E_MET_4	58 R	3.6	86	1121	2005	6013	7.2	81.8	80.9	78.9	79.3	73.8
22_MET_1	59	3.9	87	1447	2989	1407	19.2	78.9	89.5	44.0	54.8	80.3
22_MET_2	60	2.8	33	554	96963	2094	9.5	75.5	82.1	87.8	68.8	65.6
22_MET_3	61 R	6.5	40	47	22678	30718	14.1	68.2	59.7	60.7	86.8	75.4
22_MET_4	62	2.9	29	978	22531	4332	17.8	74.6	79.0	66.5	72.8	77.3
HW_MET_1	63	7.1	164	3375	1687	34145	18.9	86.1	89.2	43.9	90.0	47.8
HW_MET_2	64	4.2	26	4281	6504	18505	14.1	35.2	70.0	44.4	70.1	37.0
HW_MET_3	65	8.3	8.8	263	2785	2939	20.6	35.3	41.3	26.5	44.7	30.9
HW_MET_4	66 R	19.1	2.3	27	1125	171	12.2	67.9	76.5	62.9	58.9	68.2
109_MET_1	14 R	12.7	62	105	2541	1146	31.2	93.8	96.0	91.7	91.9	93.5

Variability Comp ID	Test #	Combined 2nd Cleaner Concentrates										
		Mass Pull (%)	Grade (% or g/t)					Recovery (%)				
			Au	Ag	As	Sb	S	Au	Ag	As	Sb	S
NEX-MS01	67 R	13.2	29	1933	3424	2540	14.1	77.0	91.8	77.6	85.9	81.5
NEX-RHY02	68 R	15.7	15	205	2198	1885	24.4	79.2	93.1	79.5	91.8	86.8
NEX-MS03	69 R	14.5	24	465	2016	1159	21.5	82.2	92.1	71.8	88.8	85.7
NEX-RHY04	70	13.6	12	179	1559	1300	17.0	76.7	87.4	68.6	87.0	78.2

Figure 13-12: Open MF2 Cleaner Circuit FS Variability Testwork (Part 1)

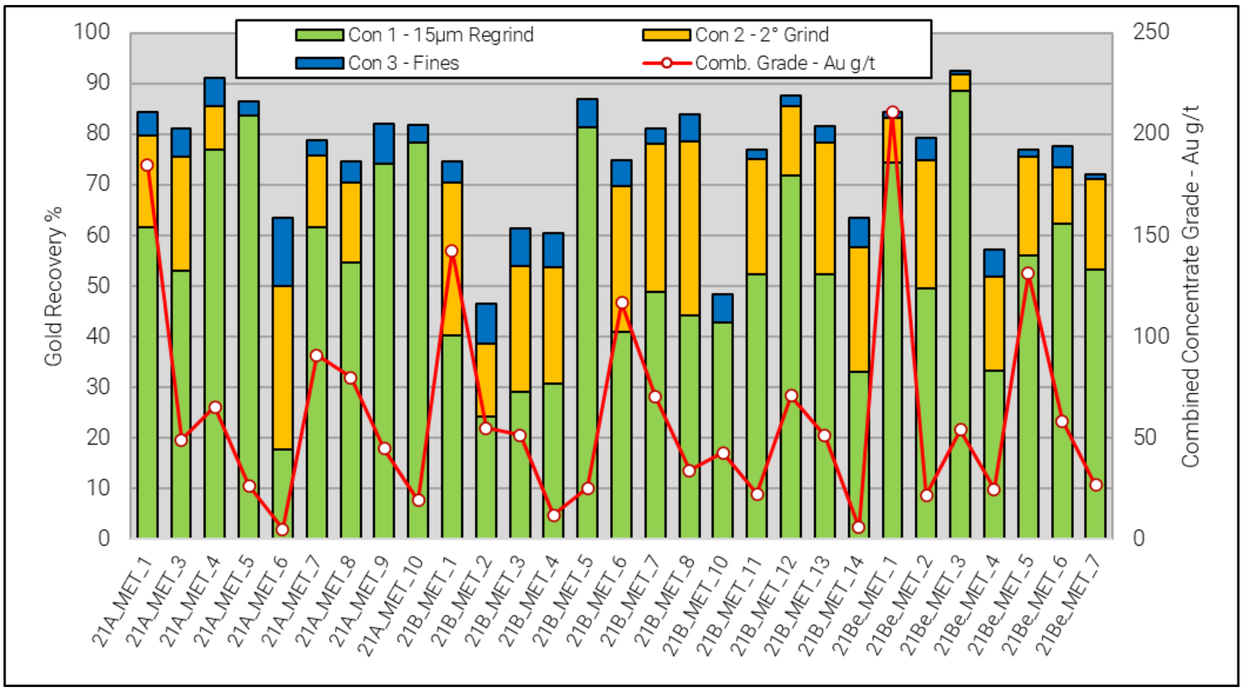
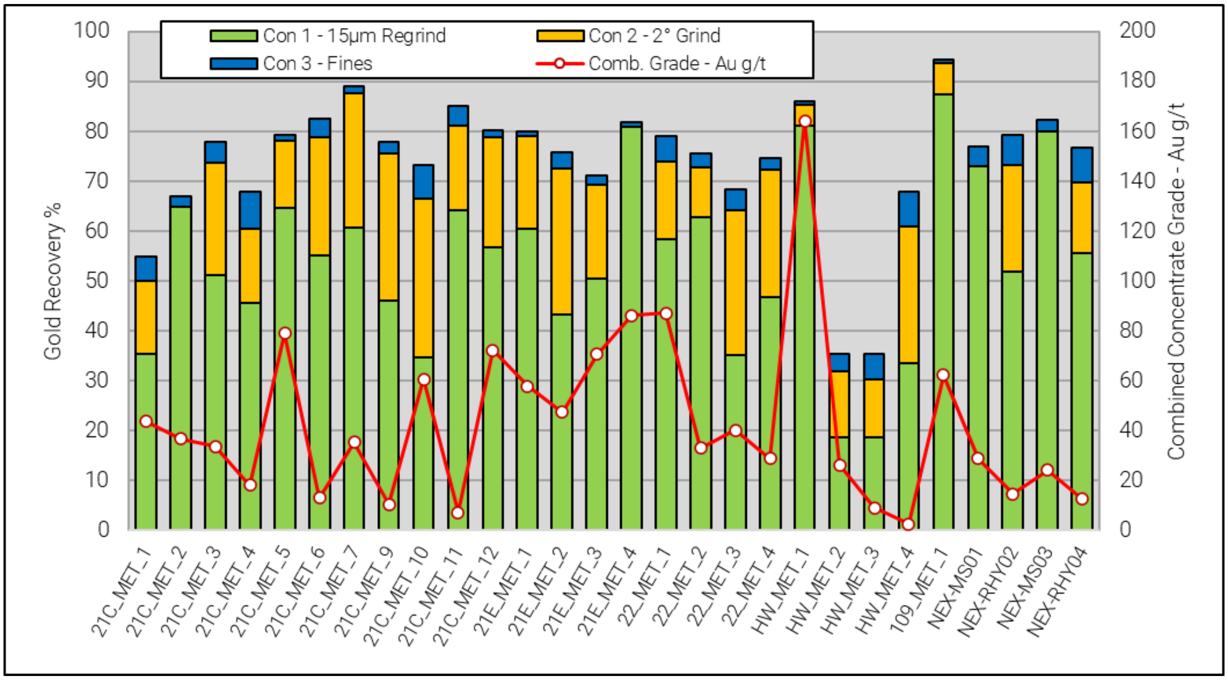


Figure prepared by Ausenco, 2022.

Figure 13-13: Open MF2 Cleaner Circuit FS Variability Testwork (Part 2)



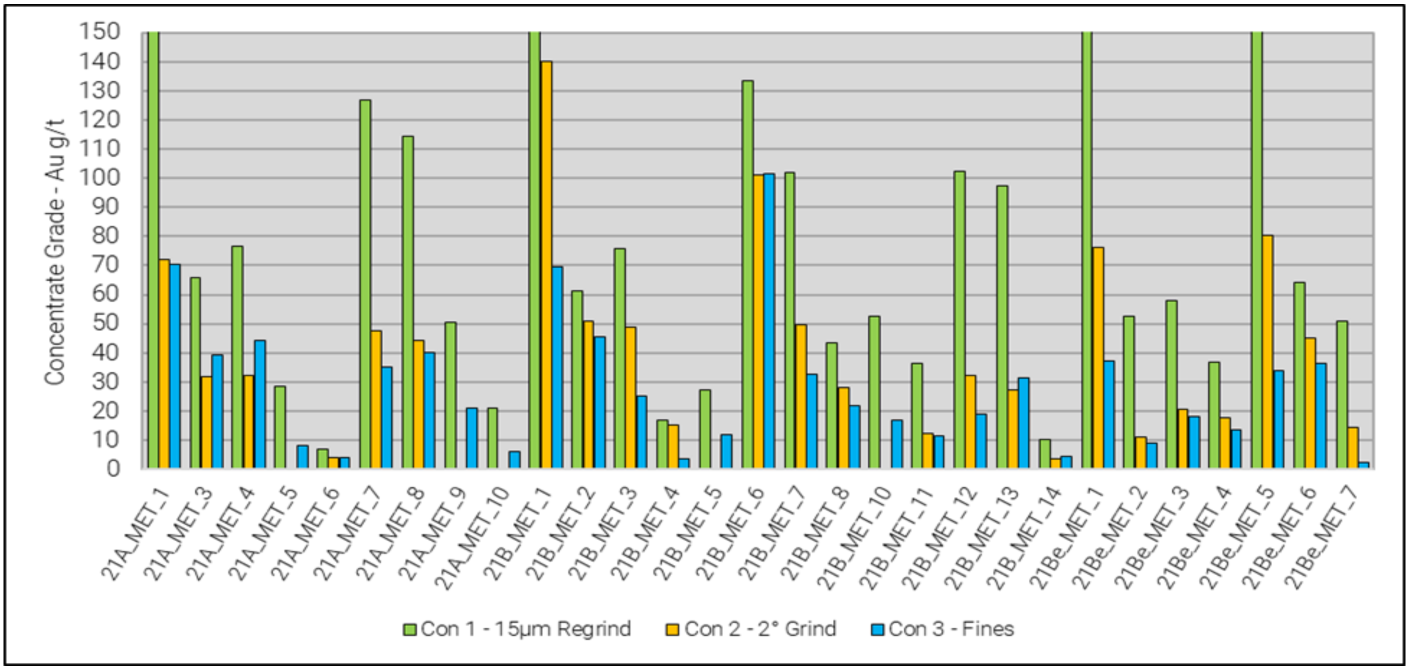
Note: Figures prepared by Ausenco, 2022.

13.4.6 Final Concentrate

The final concentrate grades from the Variability composite testing were variable, with some samples not achieving 20 g/t Au. This was particularly evident for both HW and NEX zone samples.

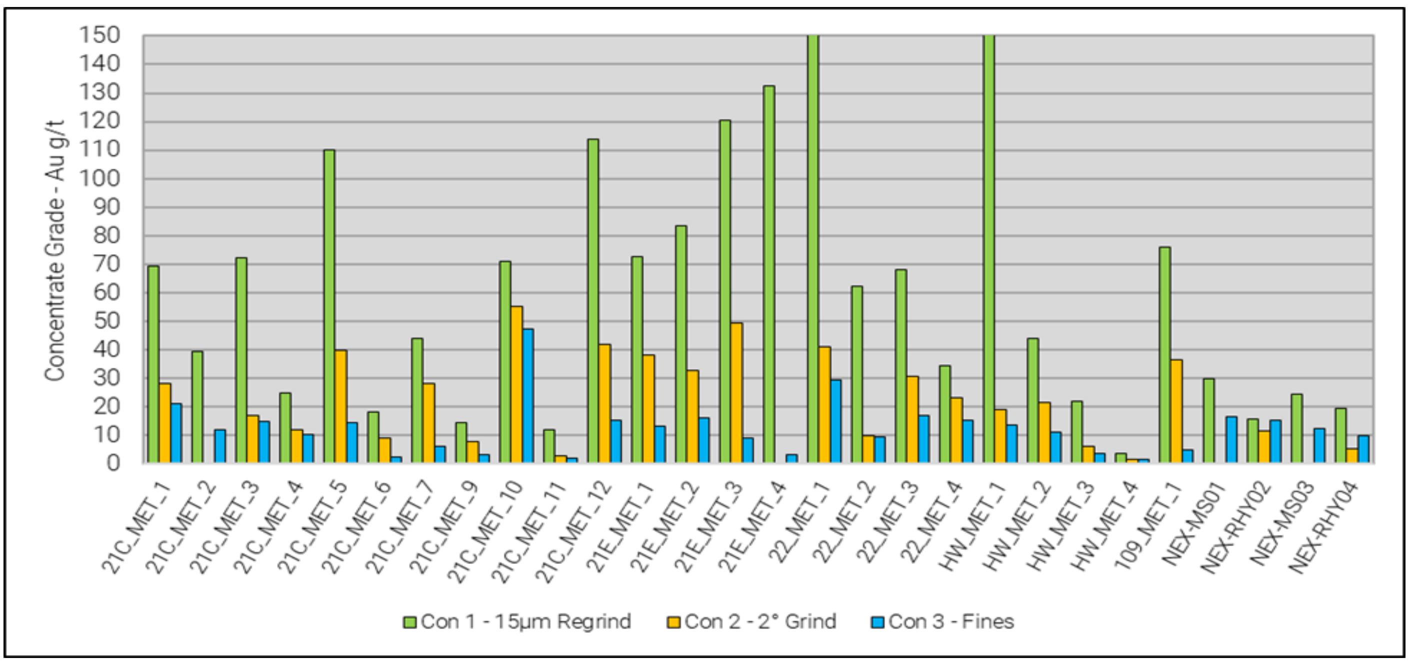
Figure 13-14 and Figure 13-15 (two parts) show the three 2nd cleaner concentrate stream grades for each of the Variability composite tests. At a 20 g/t Au threshold, almost all samples reported Con 1 grades well above this, but were diluted with Con 3 and, at times, Con 2. Note that the concentrate grade axis is limited to 150 g/t for display clarity, 6 of the samples generated Con 1 grades greater than 150 g/t. The highest Con 1 grade of 438 g/t was measured on the 21A_MET_1 sample.

Figure 13-14: FS Variability Testwork – Separate Concentrate Gold Grades (Part 1)



Note: Figure prepared by Ausenco, 2022.

Figure 13-15: FS Variability Testwork – Separate Concentrate Gold Grades (Part 2)



Note: Figure prepared by Ausenco, 2022.

Table 13-23 summarises the range of grades reported for the 57 Variability composites. An average of 52 g/t Au was achieved with almost 20% S.

Table 13-23: FS Variability Composite Combined 2nd Cleaner Concentrate Grades

	Au (g/t)	Ag (g/t)	As (g/t)	Sb (g/t)	S (%)
Average	52.4	954	8,401	8,413	19.5
Minimum	2.3	15	601	157	7.2
Maximum	211	10,527	96,963	123,011	42.8

13.4.7 Mudstone Flotation Conditions

As observed in the variability test program (see Figure 13-10), the Mudstone sample responses were significantly different from the Rhyolite samples; in particular, cleaner circuit losses (after both regrind and secondary grinding) were very high. It was noted that the rougher concentrate P80 size was considerably finer for the Mudstone samples compared with Rhyolite.

It was suggested that Mudstone samples contained a component of soft, friable minerals that were readily overground at the 100 µm target primary grind P80 size. Base Met reported organic carbon (Corg) assays for Mudstone samples between 0.4 and 3.0% while Rhyolite samples were below 0.3% Corg. While it was assumed the carbonaceous mineral was graphite (as logged on the drillcore by Skeena geologists), carbon speciation assay methods determined that very little graphite was present in the Mudstone samples.

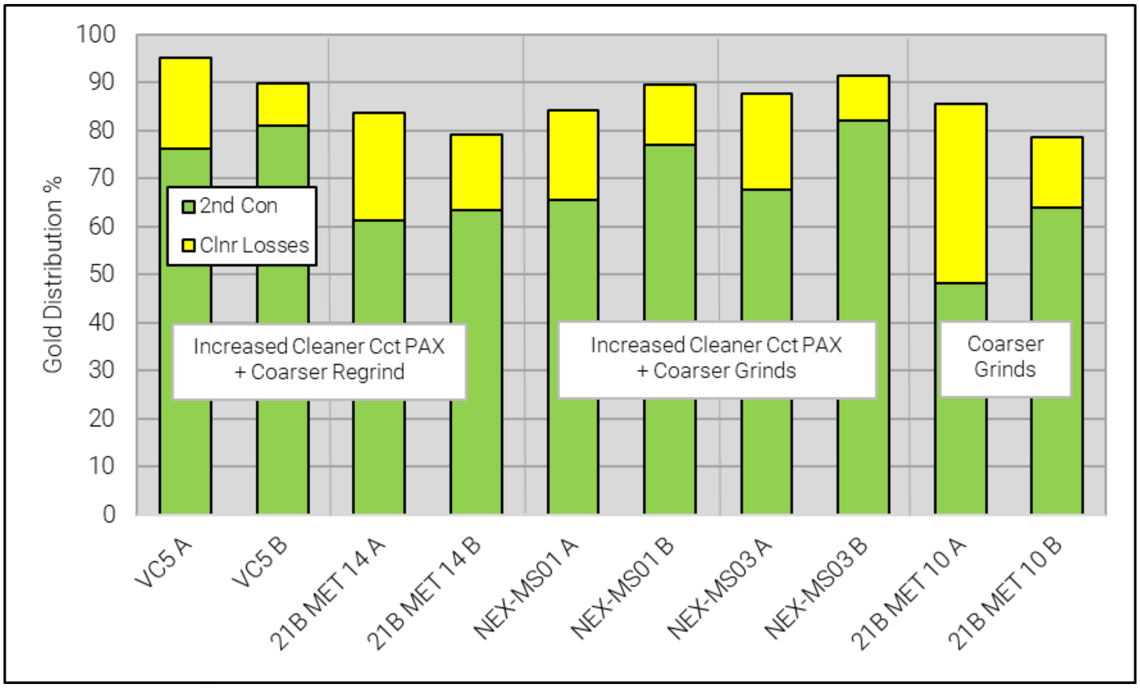
The negative effect of organic carbon on sulphide flotation performance is often attributed to the carbon species "consuming" collector. The understanding is that these minerals have a high affinity to absorb collector on their surfaces, leaving sulphide minerals to compete for adequate collector loading, or simply interfering with collector attachment. This condition is apparent in the development of the Eskay flowsheet on composite samples, as very high PAX dosages were found to be required. Methods to mitigate these issues include adding a pre-flotation stage to remove problematic hydrophobic species in advance of the main circuit and avoiding overgrinding these species, which can exasperate the problem by generating more surface area.

Base Met performed an investigation into Mudstone sample response under a range of float conditions that had the potential to mitigate issues associated with soft minerals and higher organic carbon contents: namely coarser primary grind size, prefloat of Corg and the use of carboxymethyl cellulose (CMC) as a depressant. The response was inconsistent to a prefloat stage ahead of rougher and cleaner circuits, with some Mudstone samples showing unacceptable gold losses. (This confirms the opinion that Corg is not present as graphite.). The use of CMC resulted in high gold losses as well. All test results are summarised in a report by Base Met on program BL1008 (Base Met, 2022c).

Coarsening the primary grind P80 size to 212 µm appeared to increase gold recovery on four Mudstone samples tested, however the 100 µm baseline tests were conducted using the 3-product flowsheet. Additional coarse primary grind testing on six PFS Mudstone samples in BL1043 (Base Met, 2022d) could not replicate this recovery improvement, as all tests were completed using the 2-product flowsheet. It appeared that the additional collector available in the first cleaner circuit of the 2-product flowsheet was contributing to reduced cleaner circuit losses. In the course of retesting several samples in the supplementary programs (Base Met 2022c,d,e) it appeared that both increasing the cleaner circuit collector dosage and limiting overgrinding in the regrind stage contributed to increases in gold recovery, as shown in Figure 13-16. In this figure, the B tests received additional collector and/or less grinding.

The Eskay Creek primary and secondary grinding circuits are sized to achieve P_{80} sizes of 100µm and 35µm, respectively, on 100% Rhyolite feed. However, periods of higher % Mudstone could benefit from operating at a coarser primary grind to lessen overgrinding of softer minerals present in the Mudstone.

Figure 13-16: Comparison of Mudstone Gold Recoveries – Varying Collector Dosage and Regrinding



Note: Figure prepared by Ausenco, 2022

13.4.8 Solid/Liquid Separation

During the PFS testwork program, Base Met conducted both pressure and vacuum filtration tests on a final concentrate sample. These results reported cake moistures of 15 to 25% at a relatively thin cake thickness. To further investigate what could be done to both thicken and filter the concentrate to lower moistures, a second bulk MC Composite run generated a final concentrate sample for dewatering tests. The quality of this final concentrate generated by continuous bulk flotation methods was somewhat poor in terms of sulphur grade compared to bench scale testing. The bulk flotation process may have recovered more entrained fines to the concentrate than is representative of the designed process.

Metso:Outotec is performed both thickening (Metso:Outotec, 2022a) and filtering (Metso:Outotec, 2022b) work on this sample. M:O reported this sample to have a P80 size of 71 µm (by laser) with a solids SG of 2.91 t/m^3; in slurry form, it had a pH of 10.3. Table 13-24 summarises the dynamic settling test results after flocculant screening selected a 956 VHM.

Table 13-24: MC Composite Final Concentrate Dynamic Settling Tests (Metso: Outotec 2022a)

Run	Feed		Coagulant	Flocculant	Underflow		Overflow
	Loading (t/m²/hr)	Liquor RR (m/hr)	Dose (g/t)	Dose (g/t)	% Solids	YS (Pa)	Solids (mg/L)
1	0.25	2.6	75	40	41.2	33	110
2	0.25	2.6	75	30	40.8	31	126
3	0.25	2.6	75	20	39.6	28	177
4	0.25	2.6	50	30	41.1	30	--
5	0.15	1.6	75	30	41.8	37	--

Notes: 45 VHM coagulant; 956 VHM flocculant

Metso elected to use a 45 VHM coagulant in conjunction with the 30 to 40 g/t flocculant dose. Underflow densities were consistent but did not exceed 42% w/w solids.

13.4.9 Concentrate Filtering

Metso used a Larox 100 pressure filter fitted with a 45 mm chamber and AINO T30 filter media. At 16 bar pressure for 28 minutes, they achieved a final cake moisture of 16.8% with a very thin cake of only 3.6mm (see Table 13-25).

Table 13-25: MC Composite Final Concentrate Pressure Filtration Tests (Metso: Outotec 2022b)

Parameter	Units	Run 1
Feed Density	% w/w	24
Chamber Depth	mm	45
Pumping Time	min	3
Pressing Time	min	27.75
Air Drying Time	min	3
Pumping Pressure	bar	7
Pressing Pressure	bar	16
Air Drying Pressure	Bar	10
Cake Thickness	mm	3.6
Cake Moisture	% w/w	16.82

To determine if these poor results were representative of the majority of Eskay Creek final concentrate, Base Met performed additional filter press tests on Master Composite samples (see Table 13-26). In comparison with Metso, Base Met achieved a 18% final cake moisture at 14mm thickness. Other Master composites, generated through bench scale testing, achieved 6% to 15% moistures at generally very thin cake thicknesses. Of note, the Mudstone composite achieved a 15% final moisture.

Table 13-26: Summary of Filter Press Results (Base Met BL783)

Sample	Feed Size (µm P80)	Cake Squeeze (min)	Air Blow (min)	Final % Moisture	Cake Thickness (mm)
MC Comp – Bulk Test	57	-	20	18.1	14
21A Master Comp	34	6	15	6.4	13
MS Master Comp	21	6	15	14.7	17
21B/Be Master Comp	42	2	3	9.9	16

Additional filtration testing was completed by Base Met in the recent BL1053 program, using 3rd cleaner concentrates generated from locked cycle testing, results are summarised in Table 13-27. Several tests were completed using a small 45mm diameter pressure vessel due to sample mass limitations, a single plate filter test (0.016 m^2) with a cake squeeze capability was conducted to compare with the BL783 data. The smaller filter press appears to be limited in terms of final cake moisture compared to the plate filter.

Table 13-27: Summary of Filter Press Results (Base Met BL1053)

Sample	Feed Size (µm P80)	Filter Area (m^2)	Cake Squeeze (min)	Air Blow (min)	Final % Moisture	Cake Thickness (mm)
21B MET 7	27	0.0016	-	10	15.7	22
21B MET 8	25	0.0016	-	10	11.6	22
21Be MET 2	22	0.0016	-	10	15.5	27
VC5	17	0.0016	-	10	19.1	27
VC12	24	0.0016	-	10	14.2	26
Blend Yr 1-5	25	0.0016	-	10	16.8	26
Blend Yr 1-5	25	0.0016	-	20	14.7	27
Blend Yr 1-5	28	0.016	2	20	11.0	25

As a check, the MC Composite Transportable Moisture Limit was tested by SGS Burnaby and reported a TML of 13.6% with a flow moisture of 15.1%

As a consequence of the difficulty in achieving a final concentrate moisture below TML, the current process design criteria include additional filter capacity, with expectations for a thin cake thickness as well as supplementary concentrate drying using a gas-fired dryer.

13.4.10 Reagent Additions

No changes were made to the reagent addition scheme in the recent testwork program. As reported for the PFS testwork, when the modified MF2 flowsheet was developed by Base Met, expected reagent additions are (in g/t mill feed):

- Collector: potassium amyl xanthate (PAX) 725 g/t (in 12 stages);
- Activator: copper sulphate (CuSO4) 650 g/t (in 6 stages);
- Frother: methyl isobutyl carbinol (MIBC) 130 to 250 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.

13.4.11 Updated Recovery Estimates

After the PFS testwork program, equations were presented to summarise the Eskay Creek MF2 flowsheet performance as a function of gold head grade. This was based on variability test results from the 20 samples shown in Table 13-5.

For the FS testwork program, variability samples of Rhyolite and HW/Mudstone were tested from each mineralised zone under the standard grind size conditions (100µm primary, 15µm regrind and 35µm secondary). Due to the significant differences between the Rhyolite and HW/Mudstone response, these lithologies were modelled separately.

Initially, all open circuit cleaner results from both PFS (17) and FS (57) variability samples were used in developing the recovery models. This set of 70 variability tests included 54 Rhyolite, 14 Mudstone, 4 Hanging Wall and 2 Blend samples. Results from repeated tests were used where completed. A total of 24 samples received repeat tests generally to achieve the secondary and regrinding size targets, in some cases cleaner collector dosages were increased. Master composites were excluded as they generally contained varying blends of the two distinct lithologies. As results were grouped and reviewed, the metallurgical results from 16 samples were removed from the data set for a variety of reasons indicated in Table 13-28.

Table 13-28: Variability Samples Excluded From Model Data Set

Reason for Exclusion	Litho Types	Sample Count
Sample a mixture of 2 or more rock types	Blend. RHY	3
Low feed grades, not ore	HW, RHY	3
Excessive cleaner circuit losses, no opportunity to repeat test	HW, MS	4
Excessively coarse secondary grind, low S content in feed	RHY	1
Unusually high mass recovery given very low S content in feed	RHY	2
Low Au recovery compared to other samples of similar Au/S feed characteristics	RHY	3

Models were developed for gold, silver and sulphur recoveries for the two litho groups. Logarithmic equations were generally fit to recoveries as a function of feed grades. Rhyolite concentrate mass recoveries were calculated using an Fe & S multi-variable regression, the gold recovery was then applied to determine the concentrate grades, as this appeared to better match the test data. Gold in concentrate for the HW/Mudstone samples was determined from an Au:(Fe+S) ratio and concentrate mass was subsequently calculated.

The most economical operating point for a given mine period was heavily influenced by NSR terms, specifically payable gold as a function of gold in concentrate. For this reason, it was determined that producing a 1st cleaner concentrate was more appropriate for feeds with higher gold to pyrite ratios while 2nd cleaner concentrates were generally more appropriate for ores with higher sulphide mineral (pyrite) contents. This was accommodated by developing essentially 3 sets of models for each litho group: a 2nd cleaner concentrate model, a 1st cleaner concentrate model, and a mixed data model which selected either the 1st or 2nd cleaner based on a sulphur grade target. The resulting equations are shown in Table 13-29.

Table 13-29: FS Model Equations – Gold, Silver, and Sulphur

Parameter	Data Group	1st Cleaner Concentrate	2nd Cleaner Concentrate	Mixed Cleaner Concentrate
Rhyolite Gold Recovery (%)	Au/(Fe+S) < 1.0	82.6 + 4.87*LN(Au)	78.9 + 5.18*LN(Au)	78.9 + 5.18*LN(Au)
	Au/(Fe+S) < 1.0 > 2.0	76.4 + 7.91*LN(Au)	72.3 + 8.66*LN(Au)	72.3 + 8.66*LN(Au)
	Au/(Fe+S) < 2.0	69.2 + 6.23*LN(Au)	62.2 + 8.06*LN(Au)	62.2 + 8.06*LN(Au)
Rhyolite Con Mass Recovery (%)	All	2.75 - 1.09 *(Fe) + 4.07*(S)	1.21 - 0.99 *(Fe) +3.57*(S)	0.86 - 1.10 *(Fe) +4.42*(S)
Rhyolite Silver Recovery (%)	Ag < 50 g/t	75.1 + 3.27*LN(Ag)	72.1 + 3.55*LN(Ag)	72.1 + 3.77*LN(Ag)
	Ag > 50 g/t	80.7 + 2.44*LN(Ag)	78.6 + 2.64*LN(Ag)	80.2 + 2.52*LN(Ag)
Rhyolite Sulphur Recovery (%)	All	79.7 + 6.69*LN(S)	76.6 + 7.50*LN(S)	77.2 + 8.57*LN(S)
Mudstone Gold Recovery (%)	All	81.83 + 5.90*LN(Au/(Fe+S))	78.3 + 7.23*LN(Au/(Fe+S))	81.44 + 7.48*LN(Au/(Fe+S))
Mudstone Gold in Concentrate (g/t)	All	42.1 * Au/(Fe+S)	55.9 * Au/(Fe+S)	43.1 * Au/(Fe+S)
Mudstone Silver Recovery (%)	All	73.5 + 2.55*LN(Ag)	67.8 + 3.18*LN(Ag)	70.1 + 3.21*LN(Ag)
Mudstone Sulphur Recovery (%)	All	86.2 - 5.46*(S)	81.1 - 5.30*(S)	77.8 -3.08*(S)

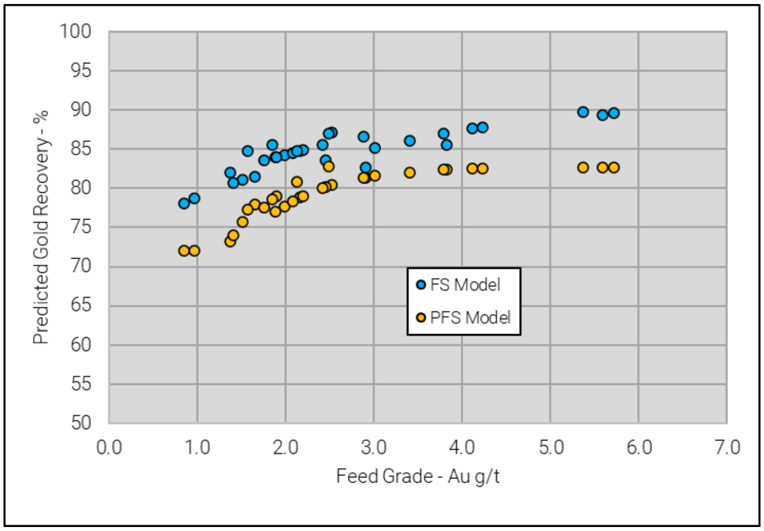
For both lithologies, other metal recoveries are estimated based on their relative upgrading to gold. Lead and zinc upgrading was assumed to be similar to sulphur:gold upgrade ratios. The resulting upgrade coefficients are shown in Table 13-30.

Table 13-30: FS Model Equations – Upgrading of Other Metal

Parameter	1st Cleaner Concentrate	2nd Cleaner Concentrate	Mixed Cleaner Concentrate
Arsenic Upgrade	0.923 * Au Upgrade	0.910 * Au Upgrade	0.909 * Au Upgrade
Antimony Upgrade	0.929 * Au Upgrade	0.917* Au Upgrade	0.913 * Au Upgrade
Mercury Upgrade	0.710 * Au Upgrade	0.710 * Au Upgrade	0.710 * Au Upgrade
Lead Upgrade	0.954 * Au Upgrade	0.942 * Au Upgrade	0.937 * Au Upgrade
Zinc Upgrade	0.954 * Au Upgrade	0.942 * Au Upgrade	0.937 * Au Upgrade

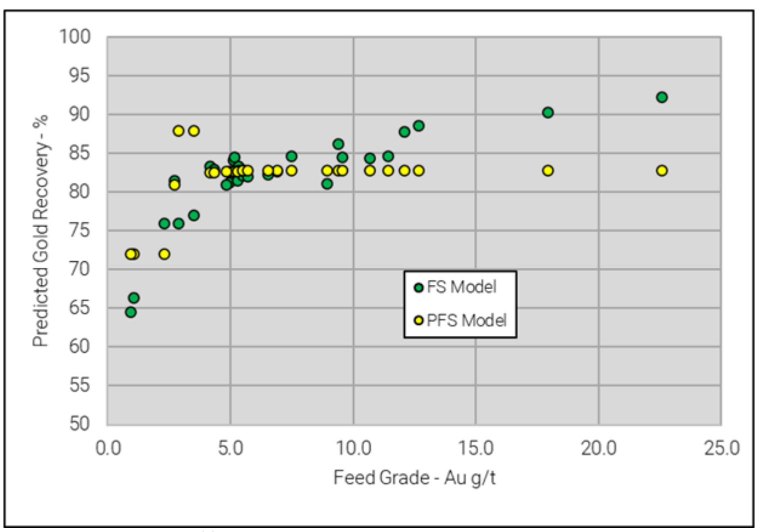
The outputs of these equations for gold recovery versus mine plan head grades is shown in Figure 13-17 and Figure 13-18. The mine plan data includes monthly resolution for Year 1 and quarterly resolution for Years 2 & 3. For Rhyolite, the updated FS curve has improved over the 'global' curve developed in the PFS testwork program. For Mudstone, the recoveries increase considerably at higher gold feed grades using the FS model and show some decrease in recovery at lower head grades compared to the PFS model.

Figure 13-17: Updated Gold Recovery vs. Head Grade - Rhyolite



Note: Figures prepared by Ausenco, 2022.

Figure 13-18: Updated Gold Recovery vs. Head Grade - Mudstone



Note: Figure prepared by Ausenco, 2022.

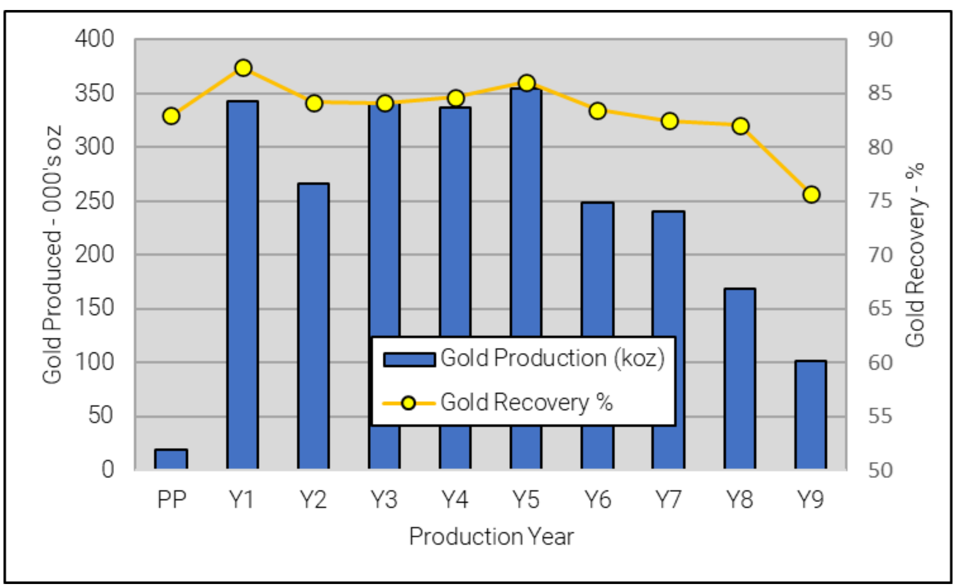
Using the FS mine plan grades and tonnes of Rhyolite and HW/Mudstone material, the updated recovery equations were used to estimate final concentrate tonnes and grades (see Table 13-31). Gold recovery ranges from 87% in Year 1 to 76% in Year 9 as the head grade drops from 4.2 g/t to 1.1 g/t. Silver recovery averages 88% over the mine life. High antimony and mercury levels are expected for the first two years before dropping to below penalty levels. Arsenic penalties are expected for most production years; however, will peak at approximately 2.6% of revenues in Y1 and drop to approximately 0.14% of revenues for the remainder of the mine life. Sulphur in concentrate is expected to range from 18 to 26% over the mine life.

Table 13-31: Estimate of Annual Gold & Silver Recoveries and Final Concentrates Grades

Production Year	Mill Feed			Recovery %		Concentrate							
	Mtpa	Au g/t	Ag g/t	Au	Ag	000t	Au g/t	Ag g/t	Hg g/t	As %	SB %	S %
Y1	3.09	4.17	76	87.2	89.3	196	57.4	1,061	1,575	3.42	2.88	20.2
Y2	3.00	3.28	86	84.2	88.4	192	43.2	1,183	897	0.78	2.38	17.6
Y3	3.00	4.21	115	84.2	88.6	234	45.5	1,298	724	0.68	1.69	17.7
Y4	3.00	4.12	142	84.7	89.8	233	45.0	1,629	445	0.56	1.79	18.0
Y5	3.00	4.26	123	86.1	90.3	234	47.0	1,429	218	0.75	1.07	18.0
Y6	3.70	2.50	65	83.5	87.1	215	36.0	971	165	0.57	0.69	21.0
Y7	3.70	2.45	52	82.5	86.0	249	30.0	674	97	0.43	0.51	21.0
Y8	3.70	1.72	50	82.1	86.4	290	18.0	540	118	0.38	0.52	19.0
Y9	3.72	1.12	27	75.7	82.1	176	18.0	458	259	0.58	0.86	26.0

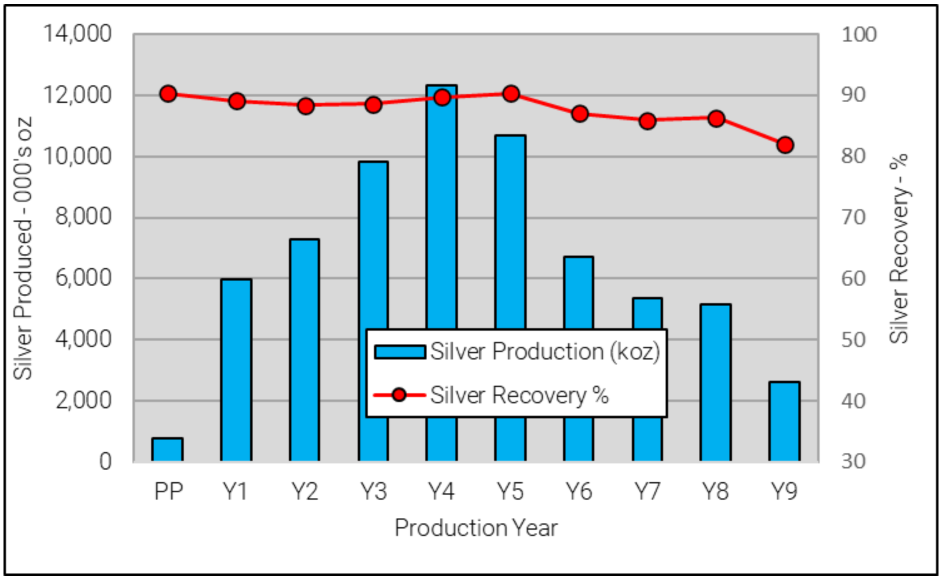
Note: Year 1 includes Pre-production activities

Figure 13-19: Mine Life Gold Production and Recoveries



Source: Ausenco, 2022.

Figure 13-20: Mine Life Silver Production and Recoveries



Source: Ausenco, 2022.

13.5 Concluding Remarks

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

In addition, any/all processing factors or deleterious elements present in the final product that could have a significant impact on potential economic extraction have been discussed and are understood. That is, concentrate grade estimates have been provided for impurities such as arsenic, antimony and mercury and appropriate terms have been provided as part of the Marketing study. In addition, the moisture of the final concentrate will be lowered to below TML levels using supplementary drying of the filter cake.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource model was prepared by Skeena and was independently validated and signed off by SRK. SRK assessed all top capping values, variograms and performed a separate check block model estimate. The resource model is based on 7,583 historical holes and 826 completed surface holes drilled by Skeena from 2018 to August 2021. The updated 2022 mineral resource estimate has a majority component of pit constrained resources. The resource estimation work was completed by Ms. K. Dilworth and was reviewed and accepted by Ms. S. Ulansky, PGeo (EGBC#36085), Senior Resource Geologist with SRK. The effective date of the mineral resource estimate is January 18, 2022.

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 that are not mineral reserves do not have demonstrated economic viability.

The database used to estimate the mineral resources was reviewed by SRK, whereby all available assay certificates were validated and confirmed in an independent database. SRK 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 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 Au-Ag 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 8,409 drill holes totalling 766,799 m. This includes 7,583 historical drill holes within the extents of the resource estimate (north of 8250N), for a total of 6,149 underground drill holes and 1,434 surface drill holes (Table 14-1). An additional 826 surface core drill holes were completed by Skeena from 2018 to August 2021 totalling 115,467.2 m (Table 14-2). The close out for the database was September 10, 2021, once all assays were received for the last hole from Phase 3. An additional 75 holes for 10,727 m have been included in this updated resource model since the 2021 PFS Resource Estimate.

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 (m)	Assays
2018	46	7,737.45	3,315
2019	203	14,091.87	8,593
2020 Phase 1 and 2*	474	80,037.67	34,318
2021 Phase 2* and 3	104	13,600.2	8,110
Total	**826**	**15,467.2**	**54,336**

*Note: Phase 2 covered both 2020 and 2021, however, the drilling metres in Phase 2 have been split in this table according to year.

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 AuEq (where AuEq equaled Au+(Ag/68)) (Barrick, 2005).

Figure 14-1 shows the traces of all surface drill holes in the historical database as well as the traces of surface drilling completed by Skeena from 2018 to August 2021 used in the updated resource model.

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. For the 2022 model update, the lithology model was updated and modified slightly using the recent drilling intervals. The structural model that was created in 2018 by Dr. Ron Uken, a Principal Structural Geologist with SRK, was used (Figure 14-2).

Figure 14-1: Oblique View (left) and Plan View (right) of the Historical and Skeena Core Drill Holes

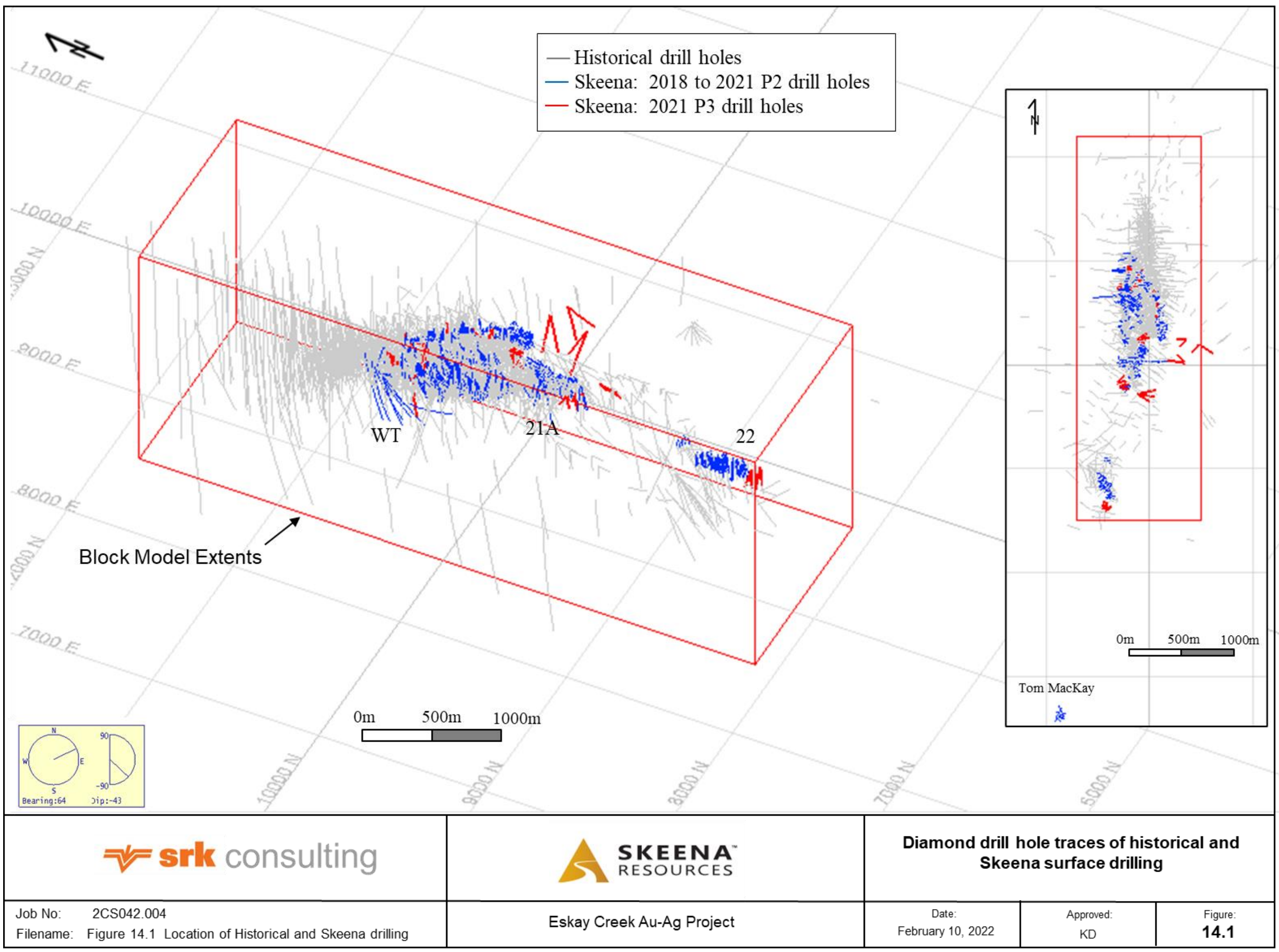
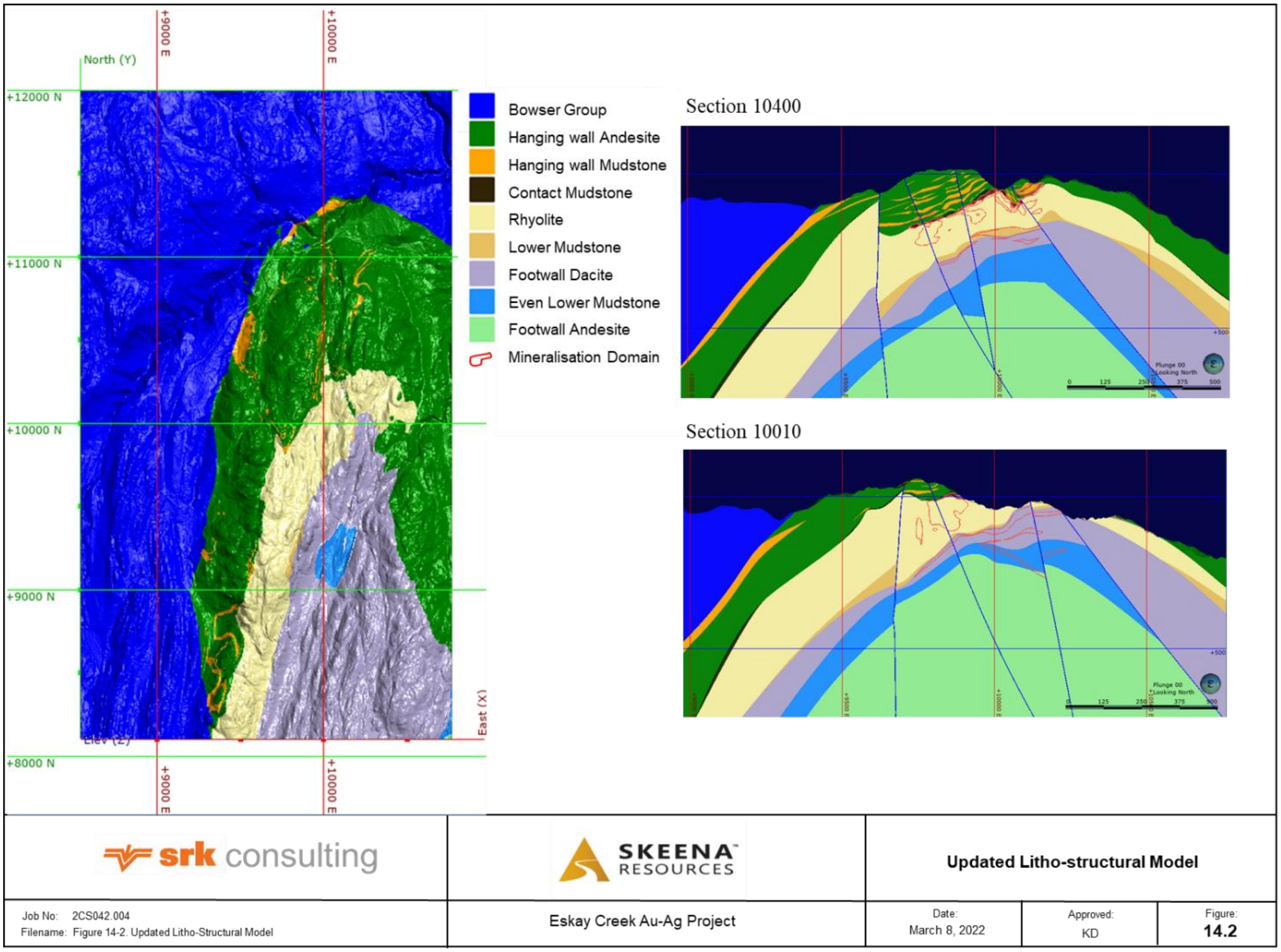


Figure 14-2: Simplified Litho-Structural Model



14.4.2 Mineralization Domaining

Seventy-five (75) new holes were drilled since the 2021 mineral resource estimate and have been incorporated and used to update the latest mineralization domain model. In total, ninety-one solids were created, including ninety mineralization solids and one solid used to restrict the influence of high-grade, mined-out material. A low-grade envelope, as was used in the previous model, was not created.

14.4.2.1Mineralization Domains

Ninety (90) mineralization solids were created to constrain 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. Intervals that were previously included in the low-grade envelope and that had continuity were constrained into new mineralization domains. Leapfrog Geo™ was the software used for initially creating all mineralization domains due to its ability to rapidly and accurately define geological zones whereby a series of geological conditions are honoured.

Three modelling methods were used:

1. Radial Basis Function (RBF) 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.

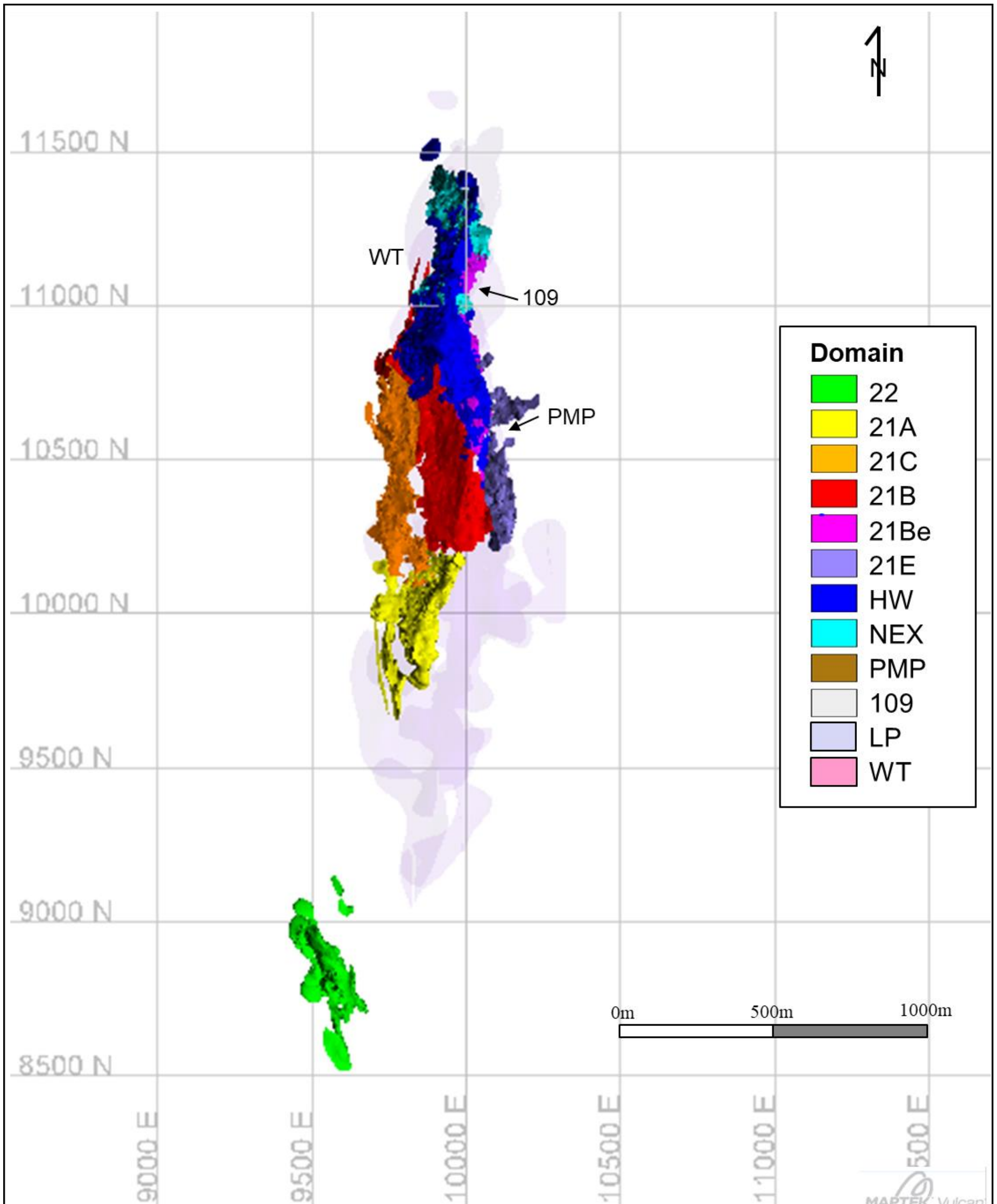
The subsequent wireframes were reviewed in section and level plan view by SRK's QP and they were deemed to be representative of the underlying geology.

The resulting mineralization wireframes differ from the previous solids due to the following changes:

- A new, narrow fault west of the 21A Fault was modelled
- Intervals that were previously included into the low-grade envelope, but have good continuity, were constrained into new mineralization domain solids
- A 1 m restriction domain around the UG workings was used to constrain the high-grade, mined-out mineralization.

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 defined in the Lower Package (LP) lithologies cannot be equated with historically defined mining zones, since they were not defined until 2020.

Figure 14-3: 2022 Model Mineralization Domains



srk consulting	SKEENA RESOURCES	Plan view of the Mineralization Domains		
Job No: 2CS042.004 Filename: Figure 14-3: Mineralization Domains	Eskay Creek Au-Ag Project	Date: Feb 10, 2022	Approved: KD	Figure: 14.3

14.4.2.2 1 m Restriction Domain

Due to the high-grade nature of the mined-out areas at Eskay Creek, a 1 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 1 m high-grade restriction domain. This was done to limit the smearing effect of the high-grade samples into the remaining resource areas.

Figure 14-4 is a representation of the 21B Domain showing the Contact Mudstone, Rhyolite and 1 m restriction domain used for estimation.

14.4.2.3 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 1 m high-grade restriction domain.

Table 14-3 summarizes the coding scheme used.

14.4.3 Topography

The topography surface was created from a 10 cm resolution LiDAR survey.

Figure 14-4: One-metre Restriction Domain Used to Constrain the High-Grade, Mined-Out Material in the 21B Domain

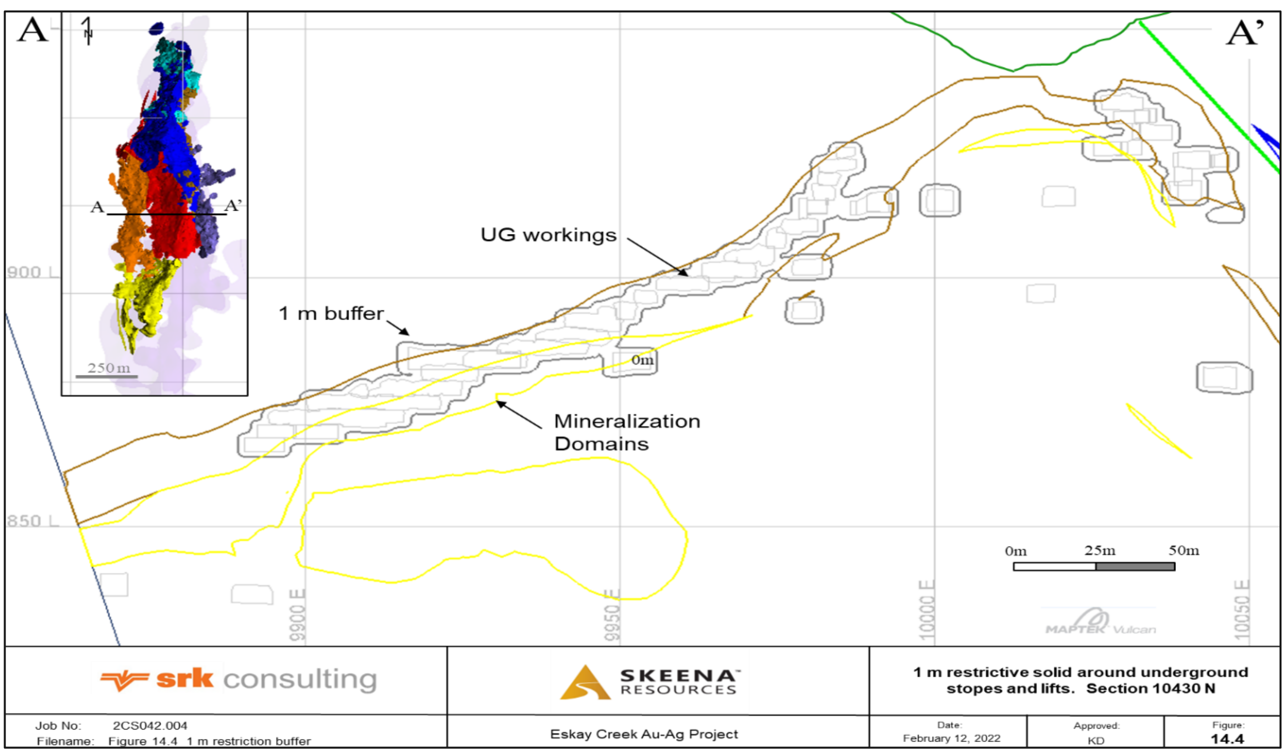


Table 14-3: Mineralization Coding Summary

Domain Name	Domain	Rock Type	Zone	Lithostructural Domain	No. of solids	Est. Zone (outside 1m restriction)	Est. Zone (within 1m restriction)
22	10	Rhyolite	101		3	1011, 1012, 1013	
21A	20	Rhyolite	201		4	2011, 2012, 2013, 2014	
		Contact Mudstone	202		1	2021	
21C	30	Rhyolite	301		6	3011, 3012, 3013, 3014, 3015, 3016	93011, 93012, 93013, 93014
		Mudstone	302		2	3021, 3022	93021, 93022,
		Hanging Wall Mudstone	303		6	3031, 3032, 3033, 3034, 3035, 3036	93031, 93032, 93033, 93034, 93035, 93036
21B	40	Rhyolite	401		8	4011, 4012, 4013, 4014, 4015, 4016, 4017, 4018	94011, 94012, 94013, 94014, 94015, 94016, 94017, 94018
		Contact Mudstone	402		3	4021, 4022, 4023	94021, 94022, 94023
21Be	50	Rhyolite	501		1	5010	95010
		Contact Mudstone	502		1	5020	95020
		Rhyolite N	504		1	5041	95041
		Contact Mudstone N			1	5042	95042
21E	60	Rhyolite	601		4	6011, 6012, 6013, 6014	96011, 96012, 96013, 96014
		Contact Mudstone	602		1	6020	96020
		Hanging Wall Mudstone	603		5	6031, 6032, 6033, 6034, 6035	96031, 96032, 96033, 96034, 96035
		Contact Mudstone	604		1	6042	96042
		Hanging Wall Mudstone	604		2	6043, 6044	96043, 96044
HW	70	Hanging Wall Mudstone	703	4	5	70341, 70342, 90343, 70344	970341, 970342, 990343, 970344
				5	7	70351, 70352, 70353, 70354, 70355, 70356, 70357	970351, 970352, 970353, 970354, 970355, 970356, 970357
				8	4	70381, 70382, 70383, 70384	970381, 970382, 970383, 970384

Domain Name	Domain	Rock Type	Zone	Lithostructural Domain	No. of solids	Est. Zone (outside 1m restriction)	Est. Zone (within 1m restriction)
NEX	80	Rhyolite	801		4	8011, 8012, 8013, 8014	98011, 98012, 98013, 98014
		Mudstone			1	8022	98022
WT	80	Rhyolite	811		6	8111, 8112, 8113, 8114, 8115, 8116	
LP	90	Rhyolite	90		1	900	
		Lower Mudstone	91		1	910	
		Dacite	92		4	921, 922, 923, 924	
		Even Lower Mudstone	93		2	931, 932	
		Footwall Andesite	94		1	940	
PMP	95	Rhyolite	95		3	951, 952, 953	9951
109	99	Rhyolite	99		1	999	9999

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 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 14-5 and show the underground workings used to deplete the current estimate.

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. 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, antimony, iron, and sulphur. The additional elements are important for optimizing 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 14-5: Plan View of the Historical Underground Mine Workings (looking east; domain wireframes are shown for reference; for simplicity, the LP domain is not shown. The Emma adit in the 22 Zone is shown in the inset).

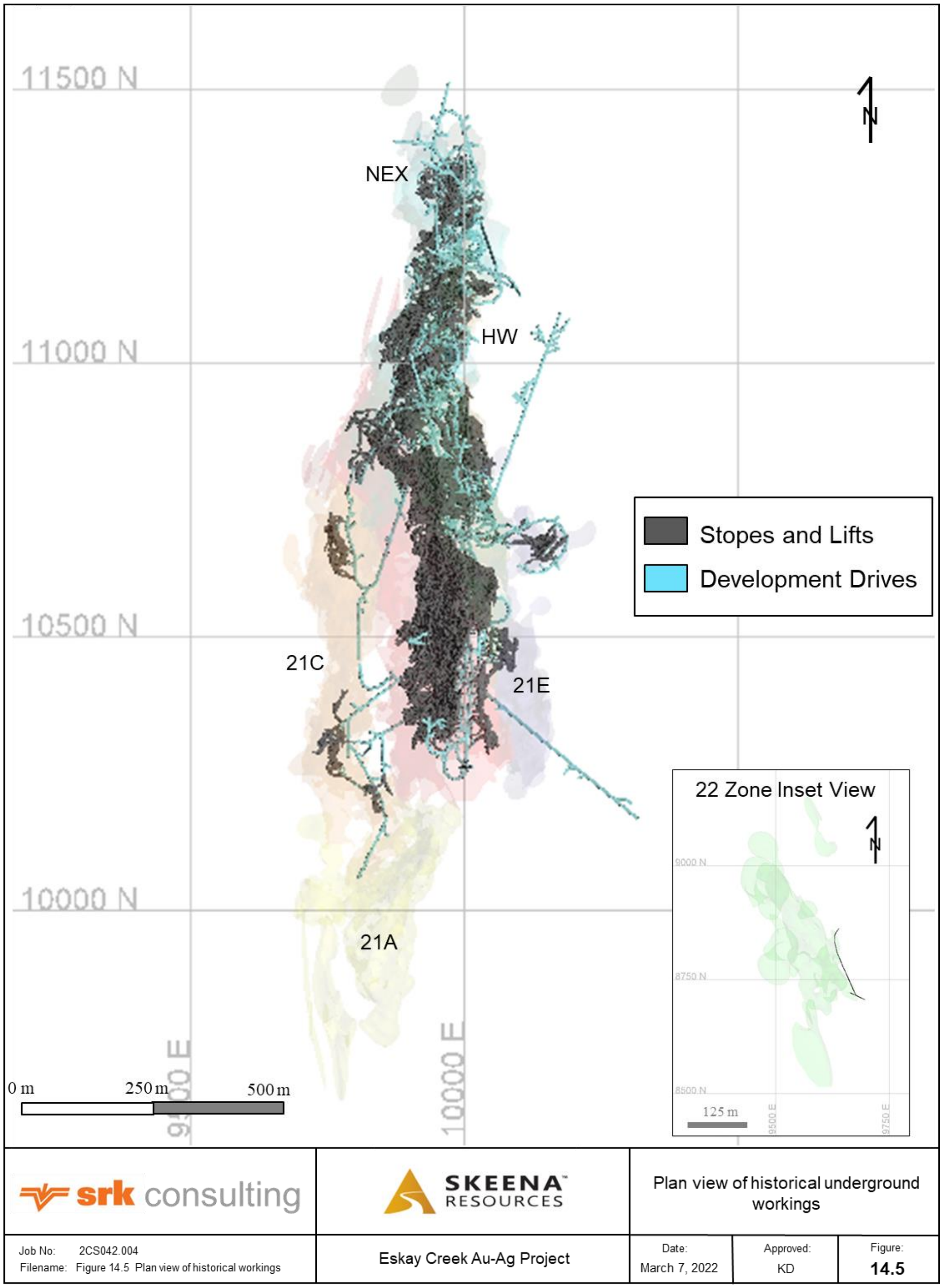
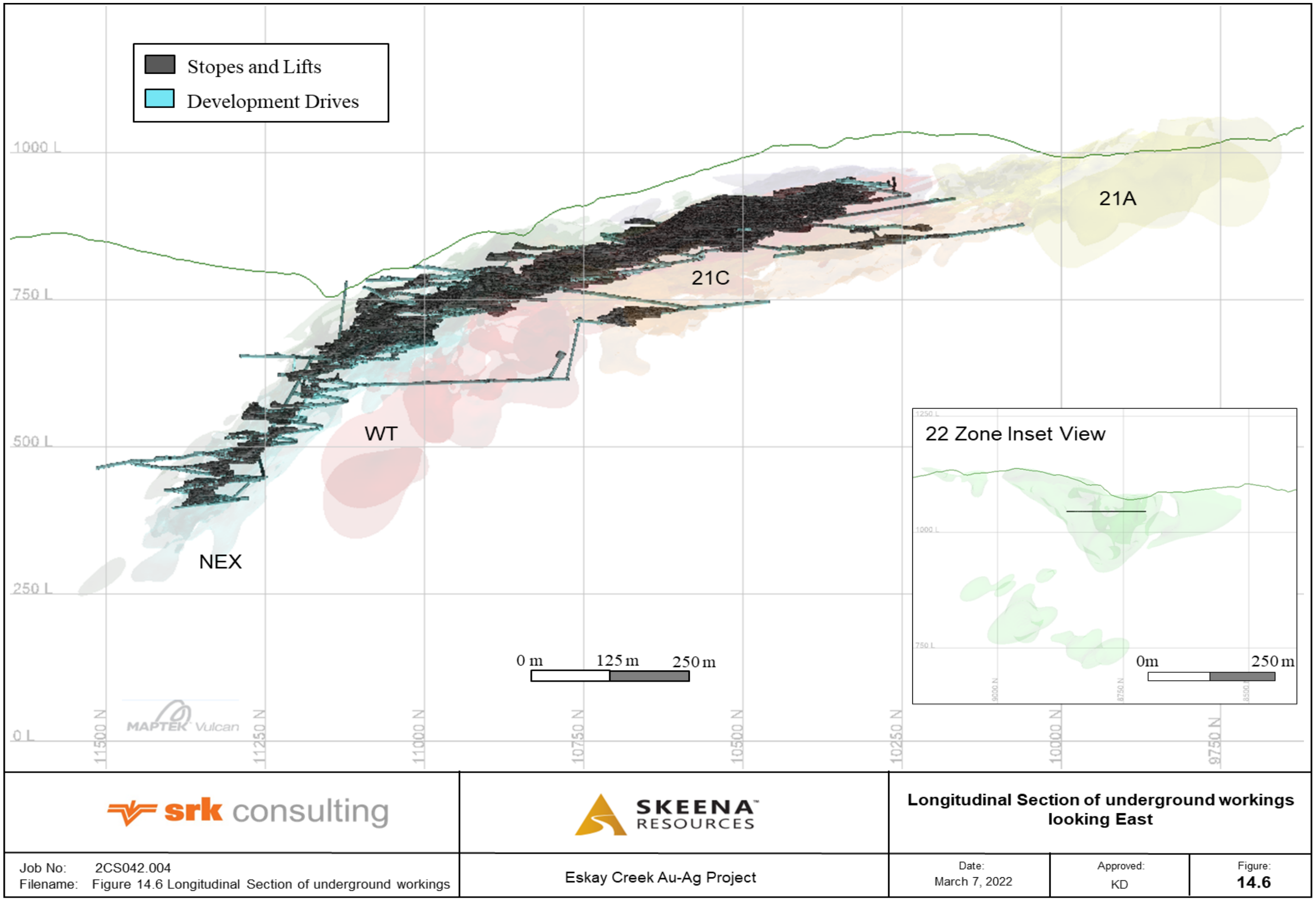


Figure 14-6: Longitudinal View of the Historical Underground Mine Workings (looking east; domain wireframes are shown for reference; for simplicity, the LP domain is not shown. The Emma adit in the 22 Zone is shown in the inset)



14.6 Compositing

To minimize bias introduced by variable sample lengths, assays were composited from assays honouring the relevant mineralization domain boundaries to 2.5 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 2.5-m Composites

A total of 88,716, 2.5 metre composites were coded into mineralization domains. Summary statistics between the assays and 2.5-m composites are shown in Table 14-4.

Table 14-4: Comparison of Assay Data to 2.5-m Composites

		Assays				2.5m-Composites			
Domain	Zone	No. of Samples	Maximum	Mean	CV	No. of Samples	Maximum	Mean	CV
Gold g/t									
22	101	4,536	225.6	1.5	3.2	2,383	58.7	1.5	2.0
21A	201	10,188	238.0	2.6	2.9	4,967	124.2	2.4	2.4
	202	1,092	677.8	19.9	2.1	487	234.7	15.9	1.8
21C	301	29,981	937.0	4.0	2.8	11,832	225.3	3.7	1.9
	302	5,175	1774.4	4.2	10.7	1,959	667.5	3.6	6.7
	303	1,640	122.1	4.0	2.0	714	67.9	3.2	1.7
21B	401	23,082	1652.4	5.1	5.6	9,606	812.9	4.7	5.1
	402	16,957	9659.0	29.4	4.1	6,701	2,516.5	26.4	2.7
21Be	501	19,109	1621.9	9.3	5.3	7,301	790.0	8.6	4.2
	502	7,801	2072.7	19.7	4.1	3,011	1,109.4	17.3	3.3
	504	1,169	361.1	4.5	4.3	434	123.0	3.8	2.8
21E	601	1,750	41.8	1.7	1.5	811	20.1	1.6	1.2
	602	466	450.6	5.4	4.9	199	193.9	5.4	3.7
	603	1,321	22.4	2.0	1.2	710	17.8	1.9	1.0
	604	1,046	115.9	6.5	2.2	389	54.8	5.5	1.7
HW	703	16,766	504.4	5.1	2.9	6,816	234.2	4.5	2.4
NEX	801	27,032	1,380.4	4.3	6.6	10,214	517.9	3.8	4.2
	802	24,652	1,971.1	7.9	5.7	9,367	1,214.9	7.1	4.6
WT	811	3,063	92.8	2.8	1.9	1,242	35.0	2.7	1.4
LP	90	1,149	55.9	1.0	2.3	472	23.2	0.9	1.5
	91	569	1380.0	3.6	16.0	252	331.5	2.5	8.4
	92	4,613	190.3	0.9	3.5	2,048	79.7	0.9	2.3

Domain	Zone	Assays No. of Samples	Assays Maximum	Assays Mean	Assays CV	2.5m-Composites No. of Samples	2.5m-Composites Maximum	2.5m-Composites Mean	2.5m-Composites CV
	93	585	39.9	0.9	3.0	263	27.9	0.9	2.4
	94	198	44.6	0.9	3.9	90	18.9	0.9	2.5
PMP	95	2,902	704.8	6.8	3.1	1,167	331.6	6.1	2.3
109	99	13,585	1625.8	10.7	4.0	5,281	887.7	9.8	2.9
Silver g/t									
22	101	4,536	3,461	6.1	3.0	2,383	1,948	46.4	2.3
21A	201	10,188	7,190	5.0	4.0	4,967	4,941	47.2	3.1
	202	1,092	22,353	7.0	5.3	487	8,759	182.7	4.1
21C	301	29,982	28,419	0.5	6.9	11,832	6,268	41.7	4.0
	302	5,174	36,696	11.0	5.8	1,959	5,164	95.1	2.8
	303	1,640	8,174	16.0	3.1	714	3,895	181.2	2.5
21B	401	23,082	44,767	3.0	5.8	9,606	25,059	228.0	5.1
	402	16,957	43,658	34.0	2.9	6,701	33,903	1,023.6	2.6
21Be	501	19,108	155,086	31.0	6.2	7,301	93,076	450.9	5.2
	502	7,801	54,899	28.0	3.7	3,011	42,787	870.6	3.3
	504	1,169	43,428	0.5	7.1	434	18,820	208.1	5.6
21E	601	1,750	4,470	8.0	3.5	811	1,465	54.3	2.3
	602	466	8,322	17.0	5.5	199	4,848	95.8	4.1
	603	1,321	9,360	25.0	4.5	710	3,488	59.2	2.8
	604	1,046	17,274	1.0	3.8	389	4,875	264.6	2.6
HW	703	16,766	28,093	22.0	4.0	6,816	14,877	226.5	3.3
NEX	801	27,025	45,492	0.5	8.8	10,214	36,657	119.9	7.8
	802	24,652	59,545	14.0	6.4	9,367	49,187	310.8	5.5
WT	811	3,063	2,524	0.5	4.7	1,242	1,222	19.4	3.5
LP	90	1,149	720	3.0	3.0	472	278	10.3	2.0
	91	569	365	7.0	2.0	252	208	15.0	1.5
	92	4,606	470	4.0	2.0	2,048	309	8.4	1.6
	93	585	145	5.0	1.4	263	65	8.6	1.1
	94	198	32	2.5	1.1	90	15	3.5	0.7
PMP	95	2,902	23,117	22.0	4.8	1,167	14,187	158.1	4.0
109	99	13,584	4,457	0.5	6.1	5,281	2,816	14.4	4.9

14.6.2 1=m Composites

For the underground model, 1-m composites were used. Five domains are located under the resource open pit 22 Zone, HW Zone, NEX Zone, WT, and the LP Zone.

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 sample lengths. 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, CV values, degradation plots, and percent metal loss calculations.

14.7.1 2.5-m Composites

Percent metal loss was variable between zones, ranging from as little as 0.2% to as high as 53 % for gold, and 1% to 22% for silver (Table 14-5). For domains with percent metal loss more than 10%, the uncapped mean values were sensitive to extremely high-grade samples. On average, less than 4% gold and 6% silver were lost during the process of capping. Gold grades were capped more aggressively in the LP domain.

Table 14-5: Summary Statistics for 2.5-m Capped and Uncapped Composites by Zone

Domain	Zone	No. of Samples	Cap Value	No. of Cuts	% of Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
Gold g/t										
22	101	2,383	32	3	0.10%	1.46	2.0	1.43	1.8	2%
21A	201	4,967	60	10	0.20%	2.43	2.4	2.36	2	3%
	202	487	130	4	0.80%	15.93	1.8	15.54	1.7	2%
21C	301	11,832	85	7	0.10%	3.71	1.9	3.67	1.8	1%
	302	1,959	60	6	0.30%	3.57	6.7	2.65	2.1	26%
	303	714	25	3	0.40%	3.17	1.7	3.04	1.5	4%
21B	401	9,606	500	5	0.10%	4.68	5.1	4.59	4.7	2%
	402	6,701	600	8	0.10%	26.43	2.7	26.08	2.5	1%
21Be	501	7,301	500	8	0.10%	8.61	4.2	8.49	4.0	1%
	502	3,011	600	5	0.20%	17.33	3.3	17.08	3.2	1%
	504	434	55	6	1.40%	3.81	2.8	3.55	2.5	7%
21E	601	811	9.5	10	1.20%	1.59	1.2	1.55	1.1	2%
	602	199	70	3	1.50%	5.39	3.7	4.21	2.7	22%
	603	710	9	3	0.40%	1.87	1.0	1.85	0.9	1%
	604	389	32	11	2.80%	5.47	1.7	5.094	1.5	7%
HW	703	6,816	180	4	0.10%	4.54	2.4	4.53	2.5	0%
NEX	801	10,214	180	14	0.10%	3.8	4.2	3.61	3.2	5%
	802	9,367	500	5	0.10%	7.05	4.6	6.84	3.9	3%
WT	811	1,242	28	4	0.30%	2.72	1.4	2.7	1.4	1%
LP	90	472	5.5	5	1.10%	0.94	1.5	0.88	0.9	6%
	91	252	8.5	7	2.80%	2.49	8.4	1.16	1.4	53%

Domain	Zone	No. of Samples	Cap Value	No. of Cuts	% of Cut	Uncapped Composites		Capped Composites		% Metal Lost
						Mean	CV	Mean	CV	
PMP	92	2,048	8	12	0.60%	0.92	2.3	0.86	1.1	7%
	93	263	4.8	6	2.30%	0.86	2.4	0.71	1.1	18%
	94	90	4.5	2	2.20%	0.88	2.5	0.66	1.3	25%
	95	1,167	80	6	0.50%	6.09	2.3	5.8	1.6	5%
109	99	5,281	500	3	0.10%	9.81	2.9	9.7	2.7	1%
Silver g/t										
22	101	2,383	600	12	0.50%	46.4	2.3	44.3	2	4%
21A	201	4,967	1,500	6	0.10%	47.2	3.1	46.3	2.7	2%
	202	487	2,700	6	1.20%	182.7	4.1	143	3.1	22%
21C	301	11,832	2,500	6	0.10%	41.7	4	40.9	3.6	2%
	302	1,959	1,700	7	0.40%	95.1	2.8	90.5	2.4	5%
	303	714	2,100	12	1.70%	181.2	2.5	169.4	2.3	7%
21B	401	9,606	18,000	5	0.10%	228	5.1	226.7	5	1%
	402	6,701	20,000	7	0.10%	1023.6	2.6	1,018.3	1.6	1%
21Be	501	7,301	25,000	9	0.10%	450.9	5.2	427.2	4.3	5%
	502	3,011	21,000	14	0.50%	870.6	3.3	838.1	3.1	4%
	504	434	5,000	6	1.40%	208.1	5.6	168.9	4.2	19%
21E	601	811	700	7	0.90%	54.3	2.3	52.3	2.1	4%
	602	199	2,000	1	0.50%	95.8	4.1	81.5	3	15%
	603	710	850	2	0.30%	59.2	2.8	54.6	1.9	8%
	604	389	3,000	7	1.80%	264.6	2.6	248.7	2.4	6%
HW	703	6,816	10,000	10	0.10%	226.5	3.3	223.5	3.1	1%
NEX	801	10,214	8,000	17	0.20%	119.9	7.8	101.6	5.1	15%
	802	9,367	20,000	12	0.10%	310.8	5.5	296.4	4.8	5%
WT	811	1,242	300	12	1.00%	19.4	3.5	16.7	2.5	14%
LP	90	472	90	6	1.30%	10.3	2	9.5	1.5	8%
	91	252	90	3	1.20%	15	1.5	14.1	1.2	6%
	92	2,048	110	9	0.40%	8.4	1.6	8.2	1.4	2%
	93	263	30	9	3.40%	8.6	1.1	8	0.9	7%
	94	90	none	-		3.5	0.7	-	-	-
PMP	95	1,167	2,000	9	0.80%	158.1	4	129.7	2.1	18%
109	99	5,281	600	11	0.20%	14.4	4.9	13	2.8	10%

* % 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.

14.7.2 1-m Composites

For the underground model, 1 m composites were used. Percent metal loss was variable between zones, ranging from as little as 0.2% to as high as 50% for gold, and 2% to 15% for silver. For domains with percent metal loss more than 10%, the uncapped mean values were sensitive to extremely high-grade samples. The LP domain was capped more aggressively.

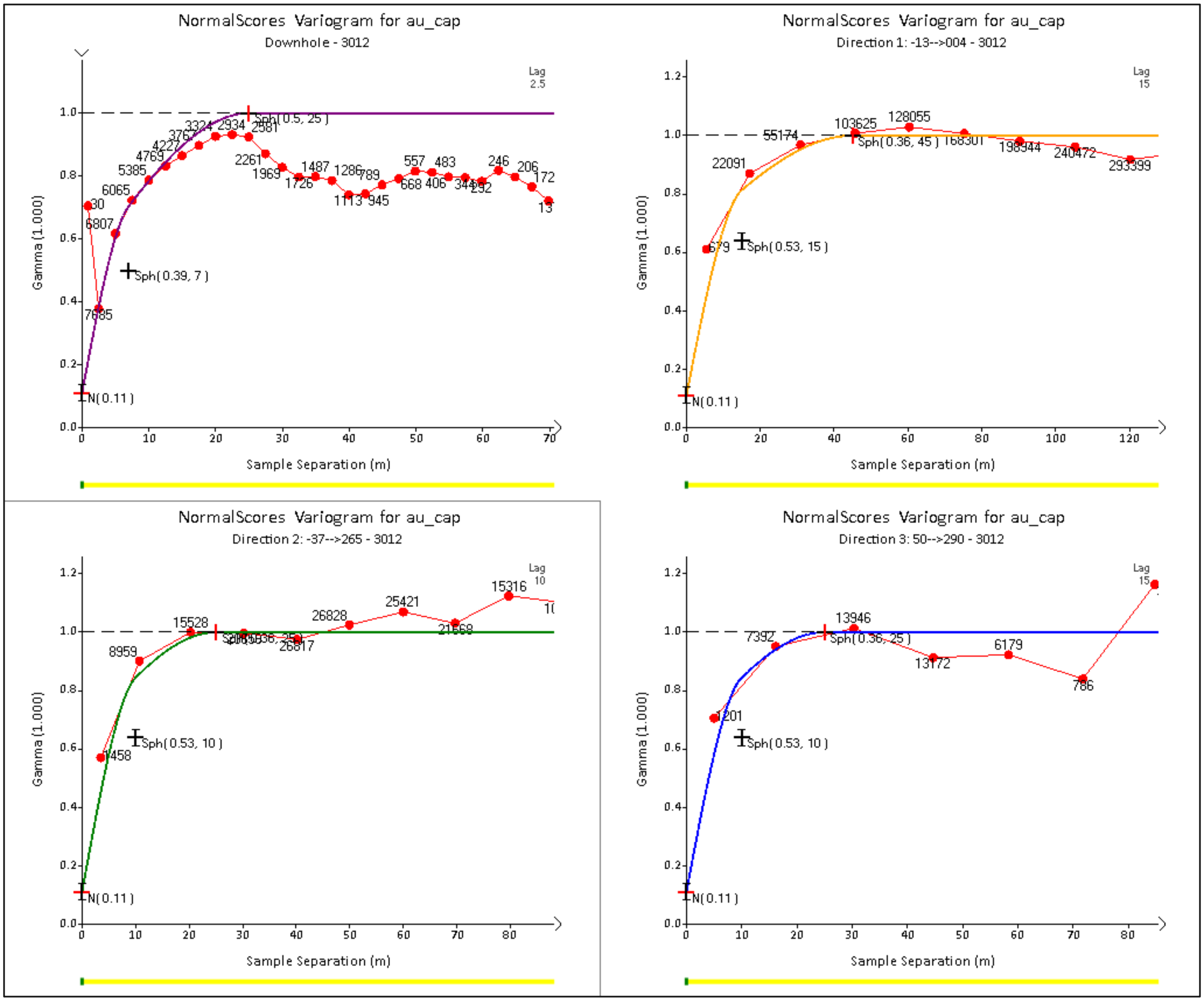
14.8 Variography

Variograms were used to assess for grade continuity, spatial variability in the estimation domains, sample search distances, and kriging parameters. Variograms were prepared using 2.5-m composites for the open pit model and 1 m for the underground model.

14.8.1 2.5-m Composites

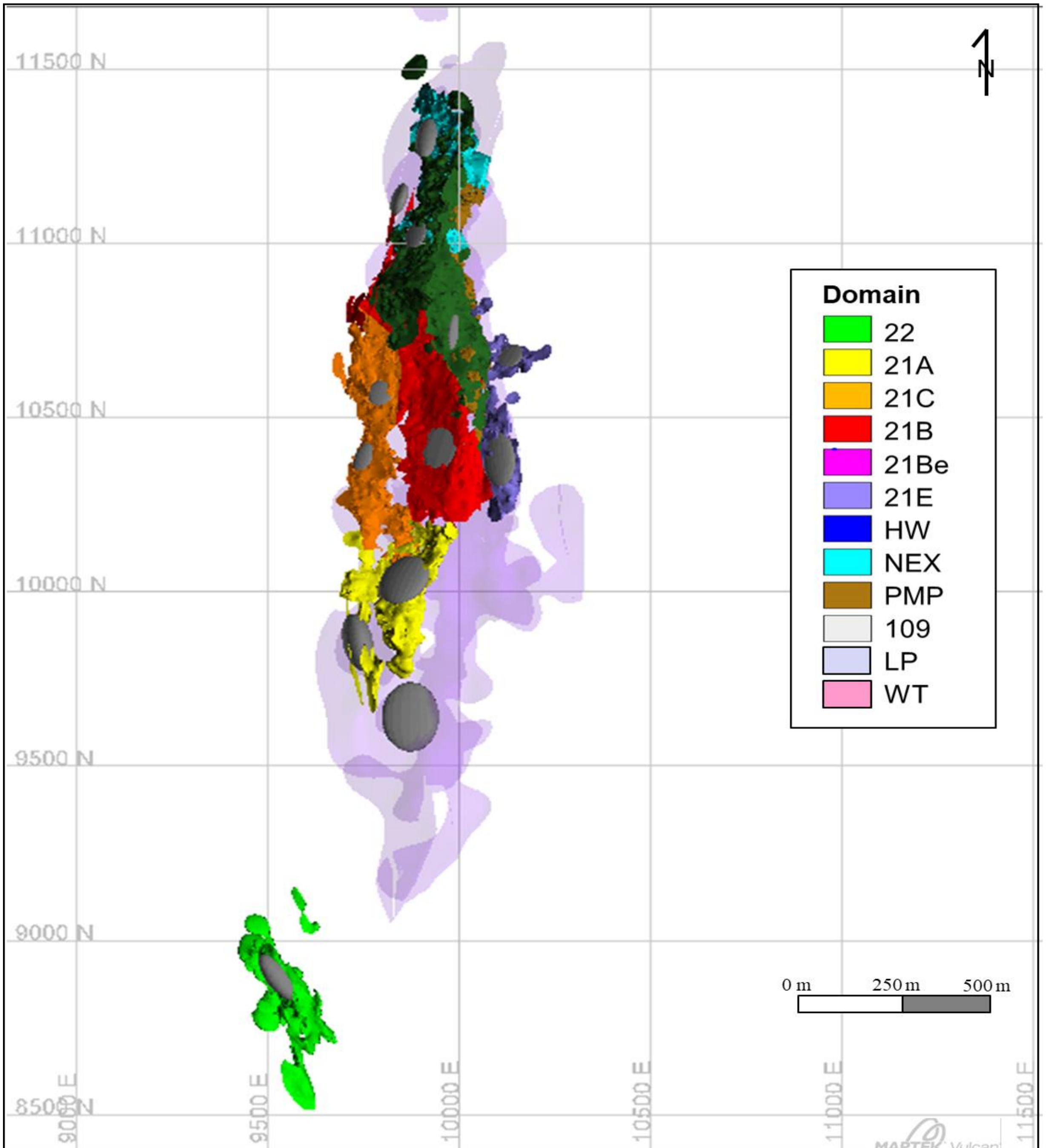
Spatial continuity was assessed using variogram maps and 3D representations of grade continuity. The most suitable orientation was selected based on the general understanding of the attitude of each mineralized zone. Initially, the variograms were produced on normal scores of the composite assay grades. 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 back transformed. Spherical variogram models were used for determining grade continuity. Figure 14-7 shows the gold variogram in the 21C Zone rhyolite, and Figure 14-8 illustrates gold search ellipsoids and ranges used per domain.

Figure 14-7: Gold Variograms in the 21C Zone Rhyolite



Note: Figure prepared by Skeena Resources, 2022

Figure 14-8: Gold Search Ellipses (in grey) Determined by Variography



srk consulting	SKEENA RESOURCES	Gold search ellipses determined from variography		
Job No: 2CS042.004 Filename: Figure 14.8 Gold Search Ellipses	Eskay Creek Au-Ag Project	Date: March 16, 2022	Approved: KD	Figure: 14.8

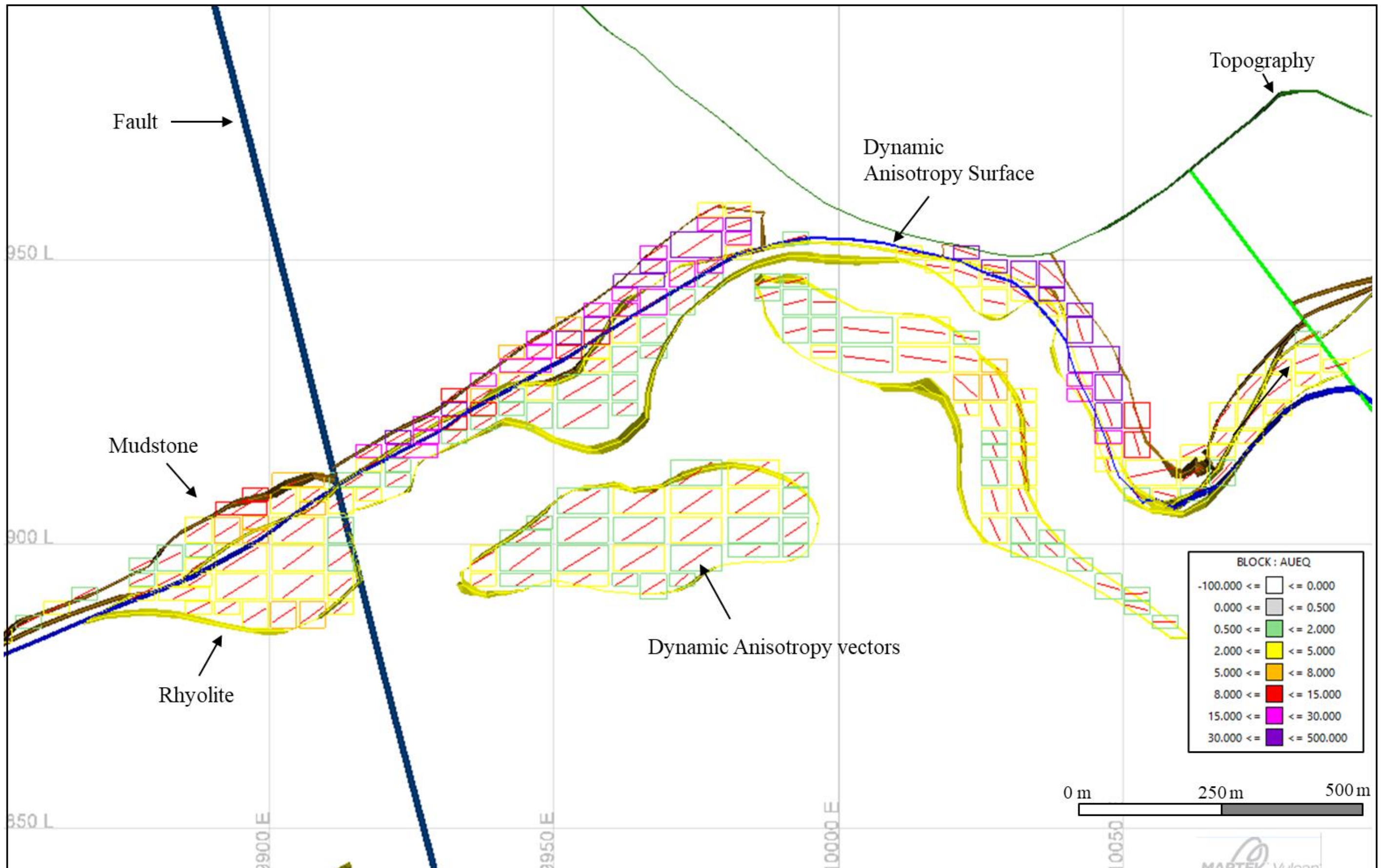
14.8.2 1-m Composites

For the underground model, variograms for the 1-m composites used the same orientations determined from the 2.5 m composites; however, the nugget, sills and ranges were updated accordingly. The Lower Mudstone and Dacite zones were subdivided into steep and shallow solids to aid with variogram creation and estimation.

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 14-9).

Figure 14-9: Dynamic Anisotropy Vectors Used in the Folded 21B Domain (looking north)



srk consulting	SKEENA RESOURCES	Dynamic Anisotropy in the 21B Domain Section 13300 N looking North		
Job No: 2CS042.004 Filename: Figure 14.9 Dynamic Anisotropy	Eskay Creek Au-Ag Project	Date: March 15, 2022	Approved: KD	Figure: 14.9

14.10 Specific Gravity

During 2018 to the end of the 2021 Phase 3 drilling program, Skeena collected 5,432 SG measurements from lithology types from all zones. The density used for tonnage calculation for the 2022 estimate is based on a combination of lithology type and zone, with the mean SG value selected from each ore zone, or, if outside of the ore zones, then average SG values within lithology type was utilized. Where there were fewer than 10 samples, SG was determined by averaging the SG of zones in that lithology. Mineralization solids 301 and 7021, which contain barite, were coded separately. Density was discussed in Section 11.2, and Table 11-6 summarizes the bulk density measurements used in the model.

14.11 Block Model and Grade Estimation

The grade estimate for the January 2022 resource model 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. Five estimation domains below the optimized resource pit were reported as resources potentially amenable to underground mining methods (22, HW, NEX, WT, 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

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 14-6.

Table 14-6: Details of the Open Pit Block Model Dimensions and Block Size

	Bearing	Plunge	Dip	Start Offset			End Offset			Block Size		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8500	-50	1200	3700	1500	10	10	5
Sub-block	90	0	0	9300	8500	-50	1200	3700	1500	5	5	2.5

Ordinary kriging (OK) was used to estimate gold and silver in all domains, apart from 5 zones which were estimated by Inverse Distance (ID): the fault zones west of the 21C Zone, the Even Lower Mudstone and the Footwall Andesite. Two and a half metre (2.5 m) 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 14-7 and Table 14-8. Pass 1 equalled two thirds of the variogram range, pass 2 equalled the variogram range and pass 3 equalled 2.5 times the variogram range. A fourth validation pass at 5 times the variogram range was estimated in the LP domain to aid with validation only.

For pass 1, a minimum of 8 and maximum of 10 composites were used per block. For pass 2, a minimum of 5 and maximum of 15 composites were used per block, and for pass 3, a minimum of 3 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 Zones. A hard boundary was applied within the 1 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 x 4 x 3 was used. A summary of the gold and silver parameters used for estimation are shown in Table 14-7 and Table 14-8Table 14-8.

Table 14-7: Gold Estimation Parameters by Estimation Zone

Zone	Est._Zone	Rock Type	Search Pass	Estimation Method	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
101	1011 to 1013	Rhyolite	1	OK	324.7/17.4/-58.4	53.6	26.8	20.1	8	10	2
			2			80	40	30	5	15	2
			3			200	100	75	3	15	2
201	2011, 2012	Rhyolite	1	OK	Dynamic Anisotropy	53.6	46.9	20.1	8	10	2
			2			80	70	30	5	15	2
			3			200	175	75	3	15	2
201	2013, 2014 (Faults)	Rhyolite	1	OK	348.2/-19.9/-95.3	56.95	43.55	26.8	8	10	2
			2			85	65	40	5	15	2
			3			212.5	162.5	100	3	15	2
202	2021	Contact Mudstone	1	OK	Dynamic Anisotropy	30.15	26.8	10.05	8	10	2
			2			45	40	15	5	15	2
			3			112.5	100	37.5	3	15	2
301	3012, 3013, 3014 (North)	Rhyolite	1	OK	Dynamic Anisotropy	30.15	16.75	6.7	8	10	2
			2			45	25	10	5	15	2
			3			112.5	62.5	25	3	15	2
301	3011 (South)	Rhyolite	1	OK	1.1/-6.7/18.9	33.5	33.5	20.1	8	10	2
			2			50	50	30	5	15	2
			3			125	125	75	3	15	2
301	3015, 3016	Rhyolite	1	ID	0 / 0 / 80	16.75	16.75	6.7	8	10	2
			2			25	25	10	5	15	2
			3			62.5	62.5	25	3	15	2
302	3021, 3022	Contact Mudstone	1	OK	Dynamic Anisotropy	26.8	23.45	6.7	8	10	2
			2			40	35	10	5	15	2
			3			100	87.5	25	3	15	2
303	3031 to 3036	Hang Wall Mudstone	1	OK	12.1/-10/3/22.9	33.5	26.8	10	8	10	2
			2			50	40	15	5	15	2
			3			125	100	37.5	3	15	2
401	4011 to 4015	Rhyolite	1	OK	Dynamic Anisotropy	20.1	13.4	6.7	8	10	2
			2			30	20	10	5	15	2
			3			75	50	25	3	15	2
401	4016 to 4018	Rhyolite	1	OK	3.1/-21.5/-57.5	20.1	16.75	6.7	8	10	2
			2			30	25	10	5	15	2
			3			75	62.5	25	3	15	2

Zone	Est._Zone	Rock Type	Search Pass	Estimation Method	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
402	4021 to 4023	Mudstone	1	OK	Dynamic Anisotropy	46.9	43.55	6.7	8	10	2
			2			70	65	10	5	15	2
			3			175	162.5	25	3	15	2
501	5010	Rhyolite	1	OK	6.7/-18.9/-47.2	25.46	13.4	6.7	8	10	2
			2			38	20	10	5	15	2
			3			95	50	25	3	15	2
502	5020	Contact Mudstone	1	OK	358.2/-21.6/-34.5	26.8	23.45	6.7	8	10	2
			2			40	35	10	5	15	2
			3			100	87.5	25	3	15	2
504	5041/5042	Contact Mudstone	1	OK	30.8/-37.8/-26.6	20.1	10.05	6.7	8	10	2
			2			30	15	10	5	15	2
			3			75	37.5	25	3	15	2
601	6011 to 6014,	Rhyolite	1	OK	354.0/-14.0/32.4	43.55	36.85	13.4	8	10	2
			2			65	55	20	5	15	2
			3			162.5	137.5	50	3	15	2
602	6020	Contact Mudstone	1	OK	354.0/-14.0/32.4	43.55	33.5	13.4	8	10	2
			2			65	50	20	5	15	2
			3			162.5	125	50	3	15	2
603	6031 to 6035	Hanging Wall Mudstone	1	OK	354.0/-14.0/32.4	53.6	33.5	13.4	8	10	2
			2			80	50	20	5	15	2
			3			200	125	50	3	15	2
604	6042 to 6044	Contact Mudstone/ Rhyolite	1	OK	100.8/37.8/26.6	30.15	26.8	6.7	8	10	2
			2			45	40	10	5	15	2
			3			112.5	100	25	3	15	2
703	70341 to 70344	Hanging Wall	1	OK	305.7/-23.9/26.3	13.4	10.05	6.7	8	10	2
			2			20	15	10	5	15	2
			3			50	37.5	25	3	15	2
703	70351 to 70357	Hanging all	1	OK	6.7/-18.9/-47.2	36.85	13.4	6.7	8	10	2
			2			55	20	10	5	15	2
			3			137.5	50	25	3	15	2
703	70381 to 70384	Hanging Wall	1	OK	2.5/-35.9/37.4	30.15	23.45	6.7	8	10	2
			2			45	35	10	5	15	2
			3			112.5	87.5	25	3	15	2
801		Rhyolite	1	OK		50.25	20.1	16.75	8	10	2

Zone	Est._Zone	Rock Type	Search Pass	Estimation Method	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
	8011 to 8014		2		6.9/-41.6/-149.2	75	30	25	5	15	2
			3			187.5	75	62.5	3	15	2
802	8021, 8022	Contact Mudstone	1	OK	25.1/-35.4/45.3	50.25	40.2	20.1	8	10	2
			2			75	60	30	5	15	2
			3			187.5	150	75	3	15	2
811	8111 to 8116	Rhyolite	1	OK	13.1/-34.4/102.1	33.5	26.8	13.4	8	10	2
			2			50	40	20	5	15	2
			3			125	100	50	3	15	2
90	910	Rhyolite	1	OK	336.4/-6.3/24.2	36.85	26.8	13.4	8	10	2
			2			55	40	20	5	15	2
			3			137.5	100	50	3	15	2
91	903	Lower Mudstone	1	OK	345.9/-4.2/24.7	87.1	53.6	16.75	8	10	2
			2			130	80	25	5	15	2
			3			325	200	62.5	3	15	2
92	921 (steep)	Upper Dacite	1	OK	10/-50/-90	77.05	23.45	20.1	8	10	2
			2			115	35	30	5	15	2
			3			287.5	87.5	75	3	15	2
92	922 (shallow)	Upper Dacite	1	OK	Dynamic Anisotropy	67	50.25	16.75	8	10	2
			2			100	75	25	5	15	2
			3			250	187.5	62.5	3	15	2
92	923 (steep)	Lower Dacite	1	OK	10/-50/-90	77.05	23.45	20.1	8	10	2
			2			115	35	30	5	15	2
			3			287.5	87.5	75	3	15	2
92	924 (shallow)	Lower Dacite	1	OK	Dynamic Anisotropy	67	50.25	16.75	8	10	2
			2			100	75	25	5	15	2
			3			250	187.5	62.5	3	15	2
93	931 (shallow)	Even Lower Mudstone	1	OK	2.5/-9.8/28.5	50.25	50.25	16.75	8	10	2
			2			75	75	25	5	15	2
			3			187.5	187.5	62.5	3	15	2
93	932 (steep)	Even Lower Mudstone	1	ID	70/0/40	50.25	50.25	16.75	8	10	2
			2			75	75	25	5	15	2
			3			187.5	187.5	62.5	3	15	2
94	940	Footwall Andesite	1	ID	70/0/40	50.25	50.25	16.75	8	10	2
			2			75	75	25	5	15	2

Zone	Est._Zone	Rock Type	Search Pass	Estimation Method	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
			3			187.5	187.5	62.5	3	15	2
95	951 to 953	Rhyolite	1	OK	346.9/-24.1/-73.5	33.5	16.75	10.05	8	10	2
			2			50	25	15	5	15	2
			3			125	62.5	37.5	3	15	2
99	99	Rhyolite	1	OK	158.8/51.7/133	46.9	30.15	16.75	8	10	2
			2			70	45	25	5	15	2
			3			175	112.5	62.5	3	15	2

Table 14-8: Silver Grade Estimation Parameters by Estimation Zone

Zone	Est. Zone	Rock Type	Search Pass	Estimation Type	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
101	1011 to 1013	Rhyolite	1	OK	324.7/17.4/-58.4	40.2	40.2	23.45	8	10	2
			2			60	60	35	5	15	2
			3			150	150	87.5	3	15	2
201	2011, 2012	Rhyolite	1	OK	Dynamic Anisotropy	46.9	26.8	26.8	8	10	2
			2			70	40	40	5	15	2
			3			175	100	100	3	15	2
201	2013, 2014 (Faults)	Rhyolite	1	OK	348.2/-9.1/115.3	40.2	40.2	20.1	8	10	2
			2			60	60	30	5	15	2
			3			150	150	75	3	15	2
202	2021	Contact Mudstone	1	OK	Dynamic Anisotropy	40.2	20.1	10.05	8	10	2
			2			60	30	15	5	15	2
			3			150	75	37.5	3	15	2
301	3011 (South)	Rhyolite	1	OK	1.1/-6.7/18.9	26.8	23.45	16.75	8	10	2
			2			40	35	25	5	15	2
			3			100	87.5	62.5	3	15	2
301	3012 to 3014 (North)	Rhyolite	1	OK	Dynamic Anisotropy	26.8	13.4	13.4	8	10	2
			2			40	20	20	5	15	2
			3			100	50	50	3	15	2
301	3015, 3016	Rhyolite	1	ID	0 / 0 / 80	16.75	16.75	6.7	8	10	2
			2			25	25	10	5	15	2
			3			62.5	62.5	25	3	15	2
302	3021, 3022	Contact Mudstone	1	OK	Dynamic Anisotropy	40.2	20.1	13.4	8	10	2
			2			60	30	20	5	15	2
			3			150	75	50	3	15	2
303			1	OK	12.1/-10.3/22.9	30.15	16.75	10.05	8	10	2

Zone	Est. Zone	Rock Type	Search Pass	Estimation Type	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
	3031 to 3036	Hang Wall Mudstone	2			45	25	15	5	15	2
			3			112.5	62.5	37.5	3	15	2
401	4011 to 4015	Rhyolite	1	OK	Dynamic Anisotropy	20.1	16.75	10.05	8	10	2
			2			30	25	15	5	15	2
			3			75	62.5	37.5	3	15	2
401	4016 to 4018	Rhyolite	1	OK	348.1/-21.5/-57.5	23.45	16.75	6.7	8	10	2
			2			35	25	10	5	15	2
			3			87.5	62.5	25	3	15	2
402	4021 to 4023	Mudstone	1	OK	Dynamic Anisotropy	46.9	33.5	6.7	8	10	2
			2			70	50	10	5	15	2
			3			175	125	25	3	15	2
501	5010	Rhyolite	1	OK	6.7/-18.9/-47.2	33.5	33.5	6.7	8	10	2
			2			50	50	10	5	15	2
			3			125	125	25	3	15	2
502	5020	Contact Mudstone	1	OK	358.2/-21.6/-34.5	20.1	13.4	10.05	8	10	2
			2			30	20	15	5	15	2
			3			75	50	37.5	3	15	2
504	5041/5042	Contact Mudstone	1	OK	30.8/-37.8/-26.6	26.8	13.4	6.7	8	10	2
			2			40	20	10	5	15	2
			3			100	50	25	3	15	2
601	6011 to 6014,	Rhyolite	1	OK	354.0/-14.0/32.4	30.15	26.8	13.4	8	10	2
			2			45	40	20	5	15	2
			3			112.5	100	50	3	15	2
602	6020	Contact Mudstone	1	OK	354.0/-14.0/32.4	30.15	26.8	13.4	8	10	2
			2			45	40	20	5	15	2
			3			112.5	100	50	3	15	2
603	6031 to 6035	Hanging Wall Mudstone	1	OK	354.0/-14.0/32.4	46.9	33.5	13.4	8	10	2
			2			70	50	20	5	15	2
			3			175	125	50	3	15	2
604	6042 to 6044	Contact Mudstone/Rhyolite	1	OK	100.8/37.8/26.6	0	0	0	8	10	2
			2						5	15	2
			3			0	0	0	3	15	2
703	70341 to 70344	Hanging Wall	1	OK	125.8/-23.9/-26.3	30.15	30.15	10.05	8	10	2
			2			45	45	15	5	15	2
			3			112.5	112.5	37.5	3	15	2
703	70351 to 70357	Hanging all	1	OK	6.7/-18.9/-47.2	0	0	0	8	10	2
			2						5	15	2
			3			0	0	0	3	15	2

Zone	Est. Zone	Rock Type	Search Pass	Estimation Type	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
703	70381 to 70384	Hanging Wall	1	OK	2.5/-35.9/37.4	20.1	20.1	10.05	8	10	2
			2			30	30	15	5	15	2
			3			75	75	37.5	3	15	2
801	8011 to 8014	Rhyolite	1	OK	6.9/-41.6/-149.2	40.2	16.75	10.05	8	10	2
			2			60	25	15	5	15	2
			3			150	62.5	37.5	3	15	2
802	8021, 8022	Contact Mudstone	1	OK	25.1/-35.4/45.3	60.3	26.8	16.75	8	10	2
			2			90	40	25	5	15	2
			3			225	100	62.5	3	15	2
811	8111 to 8116	Rhyolite	1	OK	15.4/-24.6/-101	40.2	23.45	13.4	8	10	2
			2			60	35	20	5	15	2
			3			150	87.5	50	3	15	2
90	910	Rhyolite	1	OK	336.4/-6.3/24.2	67	40.2	20.1	8	10	2
			2			100	60	30	5	15	2
			3			250	150	75	3	15	2
91	903	Lower Mudstone	1	OK	345.9/-4.2/24.7	67	67	13.4	8	10	2
			2			100	100	20	5	15	2
			3			250	250	50	3	15	2
92	921 (steep)	Upper Dacite	1	OK	10/-50/-90	87.1	73.7	13.4	8	10	2
			2			130	110	20	5	15	2
			3			325	275	50	3	15	2
92	922 (shallow)	Upper Dacite	1	OK	Dynamic Anisotropy	67	50.25	16.75	8	10	2
			2			100	75	25	5	15	2
			3			250	187.5	62.5	3	15	2
92	923 (steep)	Lower Dacite	1	OK	10/-50/-90	87.1	73.7	13.4	8	10	2
			2			130	110	20	5	15	2
			3			325	275	50	3	15	2
92	924 (shallow)	Lower Dacite	1	OK	Dynamic Anisotropy	73.7	50.25	16.75	8	10	2
			2			110	75	25	5	15	2
			3			275	187.5	62.5	3	15	2
93	931 (shallow)	Even Lower Mudstone	1	OK	2.5/-9.8/28.5	50.25	50.25	1.34	8	10	2
			2			75	75	2	5	15	2
			3			187.5	187.5	5	3	15	2
93	932 (steep)	Even Lower Mudstone	1	ID	70/0/40	50.25	50.25	16.75	8	10	2
			2			75	75	25	5	15	2
			3			187.5	187.5	62.5	3	15	2
94	940	Footwall Andesite	1	ID	70/0/40	50.25	50.25	16.75	8	10	2
			2			75	75	25	5	15	2

Zone	Est. Zone	Rock Type	Search Pass	Estimation Type	Orientation	Gold Search Radii			No. of Composites		Max Composites per Drill Hole
						X	Y	Z	Min	Max	
			3			187.5	187.5	62.5	3	15	2
95	951 to 953	Rhyolite	1	OK	346.9/-24.1/-73.5	33.5	33.5	16.75	8	10	2
			2			50	50	25	5	15	2
			3			125	125	62.5	3	15	2
99	99	Rhyolite	1	OK	158.8/51.7/133	13.4	13.4	10.05	8	10	2
			2			20	20	15	5	15	2
			3			50	50	37.5	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 14-10 and Figure 14-11 show estimated AuEq block grades in relation to 2.5 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 14-10: Visual Comparison of Block Model Gold Grades vs 2.5 m Composite Gold Grades in the 21B, 21E, and PMP Domains (looking north)

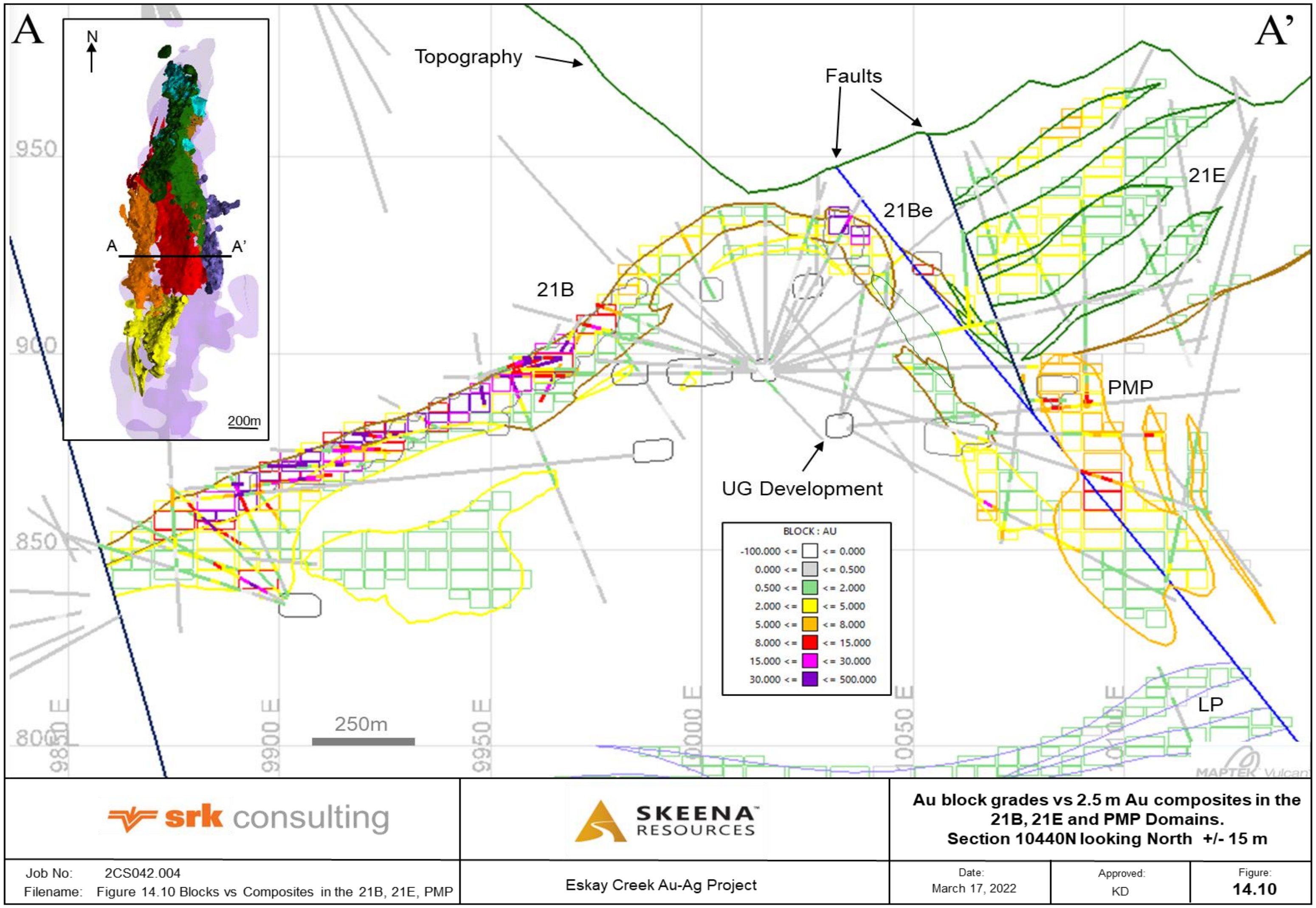
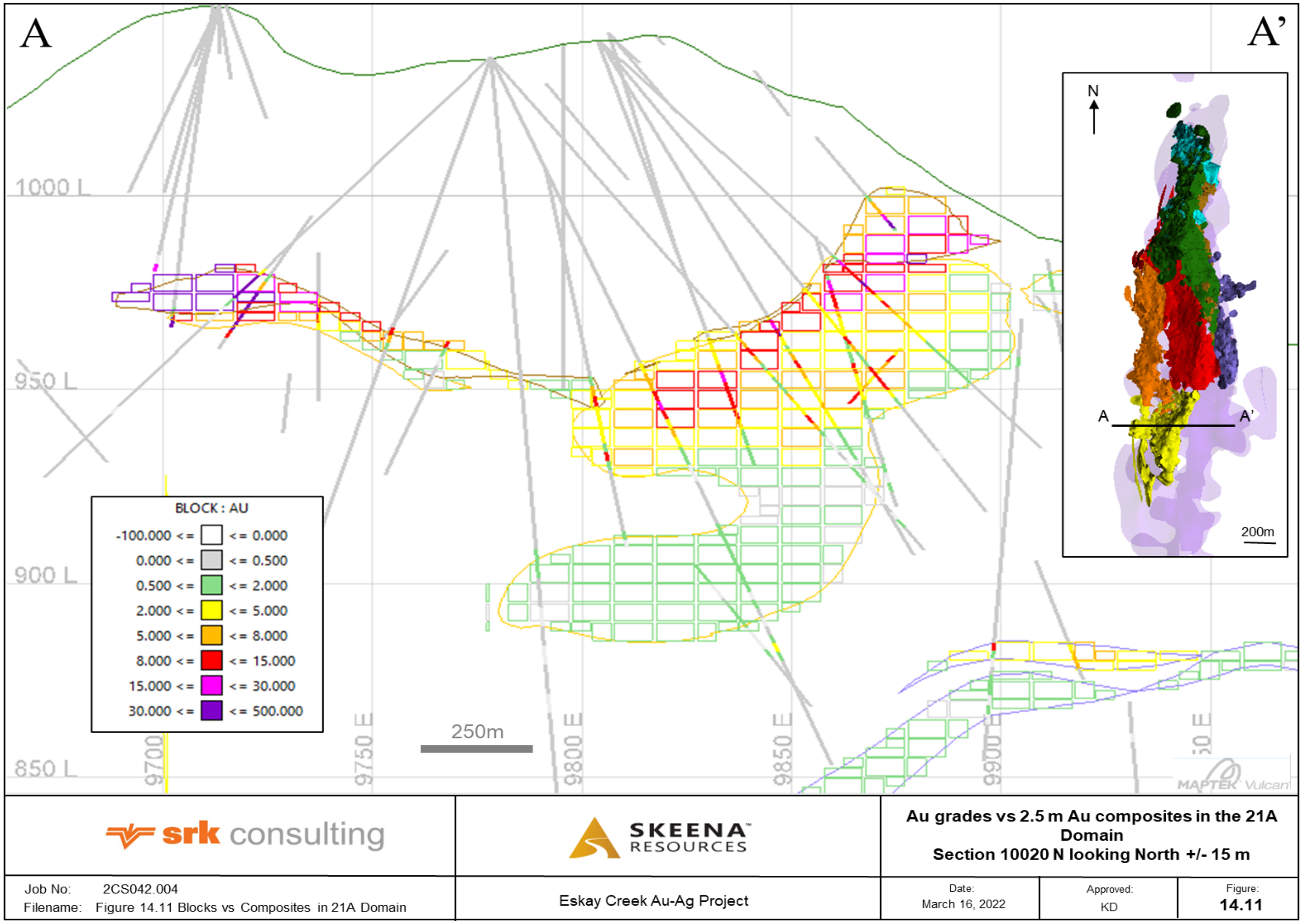


Figure 14-11: Visual Comparison of Block Model Gold Grades and 2.5 m Composite Gold Grades in the 21A Domain (looking North)



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 2.5 x 2.5 x 2.5 m were created and the closest 2.5 m composite up to a maximum distance of 200 m was estimated. For the underground model, parent blocks of 1 m x 1 m 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 (ID2) and NN declustered 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, ID2, and OK estimation methods for gold and silver by estimation zone are summarized in Table 14-9 and Table 14-10. The differences are within acceptable limits.

Table 14-9: Global Bias check for Gold by Zone

Zone	NN_Declus	AU_OK	AU_ID2	OK / NN Declus	OK / ID2
90	0.88	0.89	0.86	1%	3%
91	0.72	0.70	0.71	-2%	-1%
92	0.88	0.95	0.93	8%	2%
93	0.81	0.74	0.74	-8%	0%
94	0.91	0.92	0.92	1%	0%
95	3.49	3.57	3.54	2%	1%
99	6.84	6.75	6.79	-1%	-1%
101	1.25	1.29	1.29	3%	0%
201	1.74	1.74	1.74	0%	0%
202	11.84	11.68	12.11	-1%	-4%
301	2.60	2.61	2.60	0%	0%
302	2.33	2.30	2.32	-1%	-1%
303	2.57	2.59	2.64	1%	-2%
401	2.36	2.39	2.38	1%	1%
402	21.24	20.45	20.70	-4%	-1%
501	4.85	4.90	4.85	1%	1%
502	13.27	12.46	12.80	-6%	-3%
504	2.44	2.39	2.36	-2%	1%
601	1.57	1.57	1.57	0%	0%
602	1.43	1.50	1.55	5%	-3%
603	1.79	1.80	1.84	1%	-2%
604	4.90	4.86	4.99	-1%	-2%
703	3.19	3.17	3.20	-1%	-1%

Zone	NN_Declus	AU_OK	AU_ID2	OK / NN Declus	OK / ID2
801	2.44	2.43	2.43	-1%	0%
802	4.96	4.74	4.80	-4%	-1%
811	2.73	2.93	3.00	7%	-2%
			Overall % Difference	**0%**	**-1%**

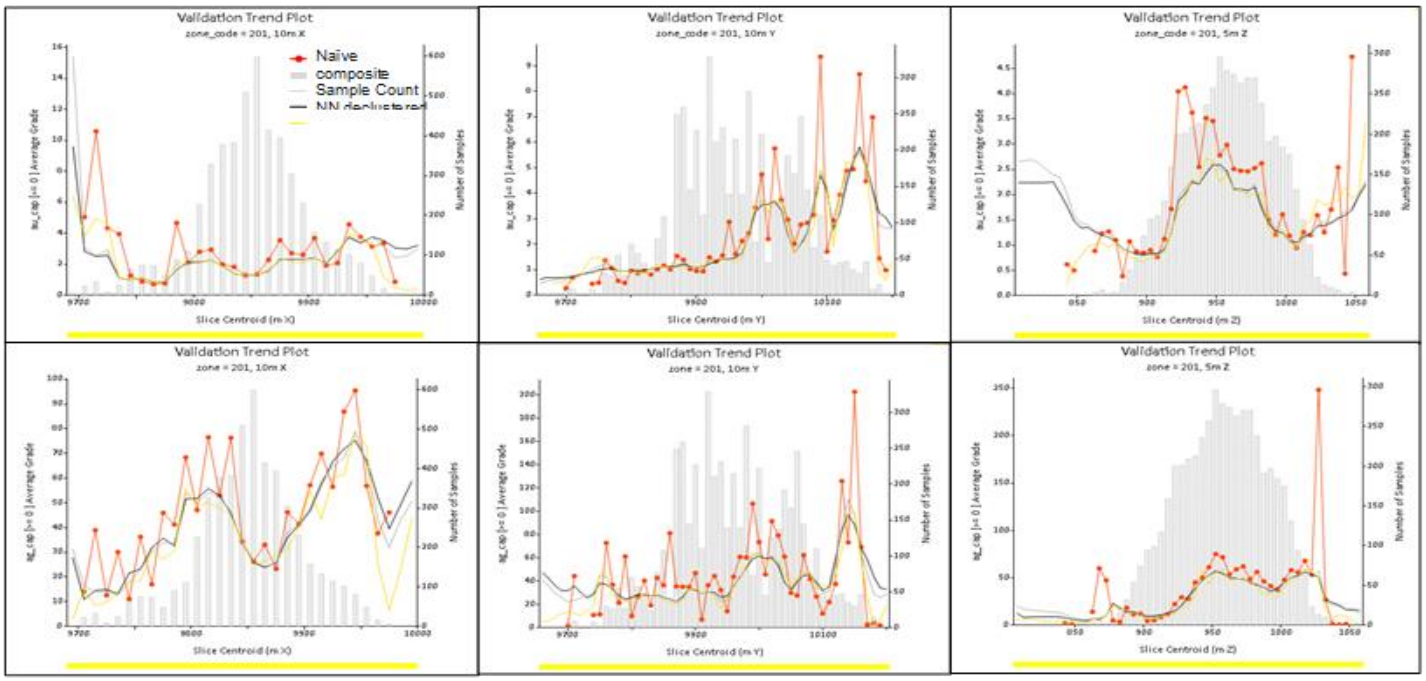
Table 14-10: Global Bias Check for Silver by Zone

Zone	NN_Declus	AG_OK	AG_ID2	OK / NN Declus	OK / ID2
90	10.5	10.5	10.4	0%	0%
91	14.9	15.1	13.5	2%	-11%
92	8.2	7.9	7.9	-4%	1%
93	7.9	7.5	7.2	-4%	-5%
94	4.3	3.8	3.7	-12%	-1%
95	78.1	74.8	74.0	-4%	-1%
99	11.9	11.7	11.7	-2%	1%
101	48.6	44.8	46.3	-8%	3%
201	35.2	36.1	35.7	3%	-1%
202	113.9	109.4	113.1	-4%	3%
301	28.9	28.9	29.3	0%	1%
302	78.6	82.6	83.8	5%	1%
303	115.0	117.3	120.1	2%	2%
401	84.5	79.8	81.7	-6%	2%
402	813.1	799.7	806.0	-2%	1%
501	201.6	201.3	195.4	0%	-3%
502	522.7	469.4	485.3	-10%	3%
504	115.3	116.8	95.5	1%	-18%
601	50.7	50.6	50.9	0%	1%
602	41.4	41.5	39.3	0%	-5%
603	52.8	55.8	57.2	6%	2%
604	231.1	239.7	233.8	4%	-2%
703	144.7	143.1	146.7	-1%	3%
801	55.0	54.3	54.3	-1%	0%
802	208.5	184.8	185.9	-11%	1%
811	13.9	14.7	14.0	5%	-5%
			Overall % Differences	**-2%**	**-1%**

14.11.1.4 Open Pit Model – Swath Plots

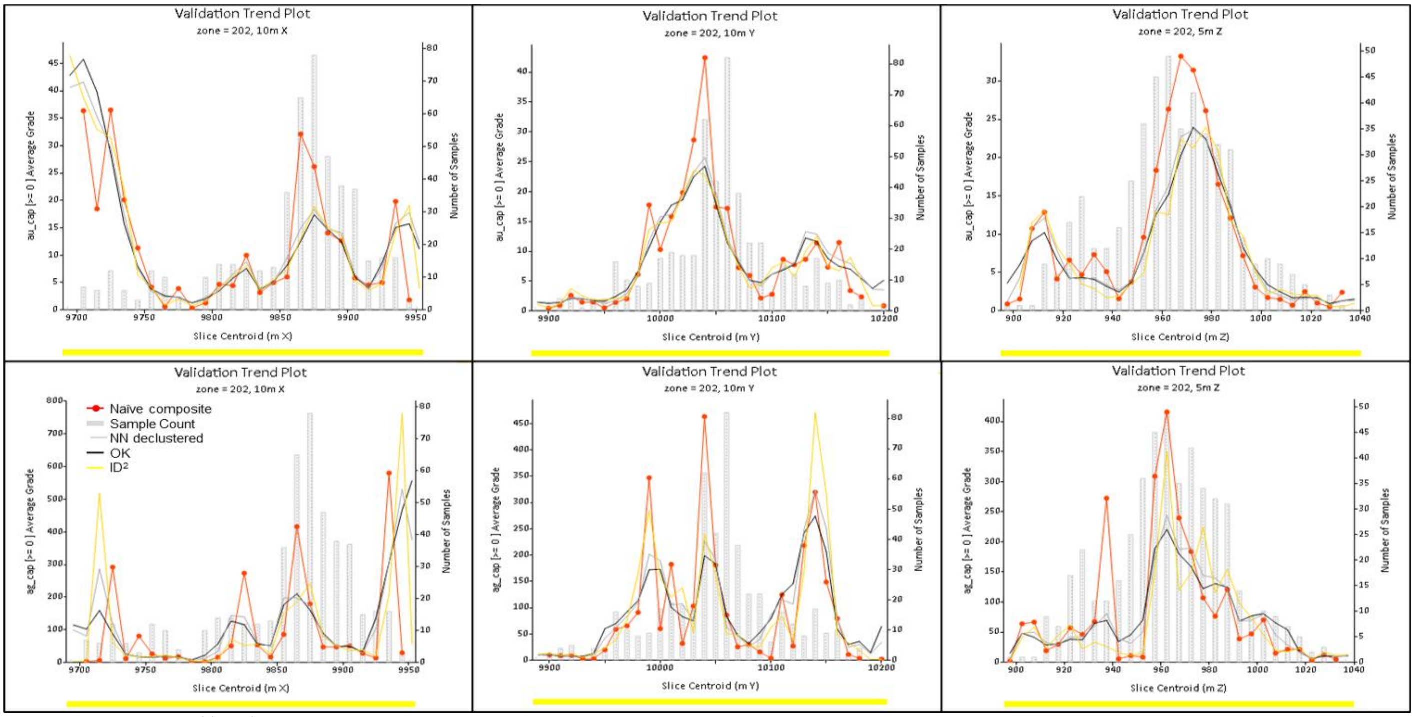
The model was checked for local trends in the grade estimate using swath plots within each zone. This was done by plotting the mean values from the naïve, declustered NN, and ID2 and against the OK estimate along north-south, east-west, and horizontal swaths. The ID2, declustered NN and OK models show 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 14-12 and Figure 14-13, respectively.

Figure 14-12: Swath Plot for Gold (top) and Silver (bottom) in Zone 201 - 21A Rhyolite, (left) Northing, (middle) Easting, (right) Elevation



Note: Figure prepared by Skeena, 2022.

Figure 14-13: Swath Plot for Gold (top) and Silver (bottom) in Zone 201 – 21A Mudstone, (left) Northing, (middle) Easting, (right) Elevation



Note: Figure prepared by Skeena, 2022

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 14-11.

Table 14-11: Details of Block Model Dimensions and Block Size for the Underground Model

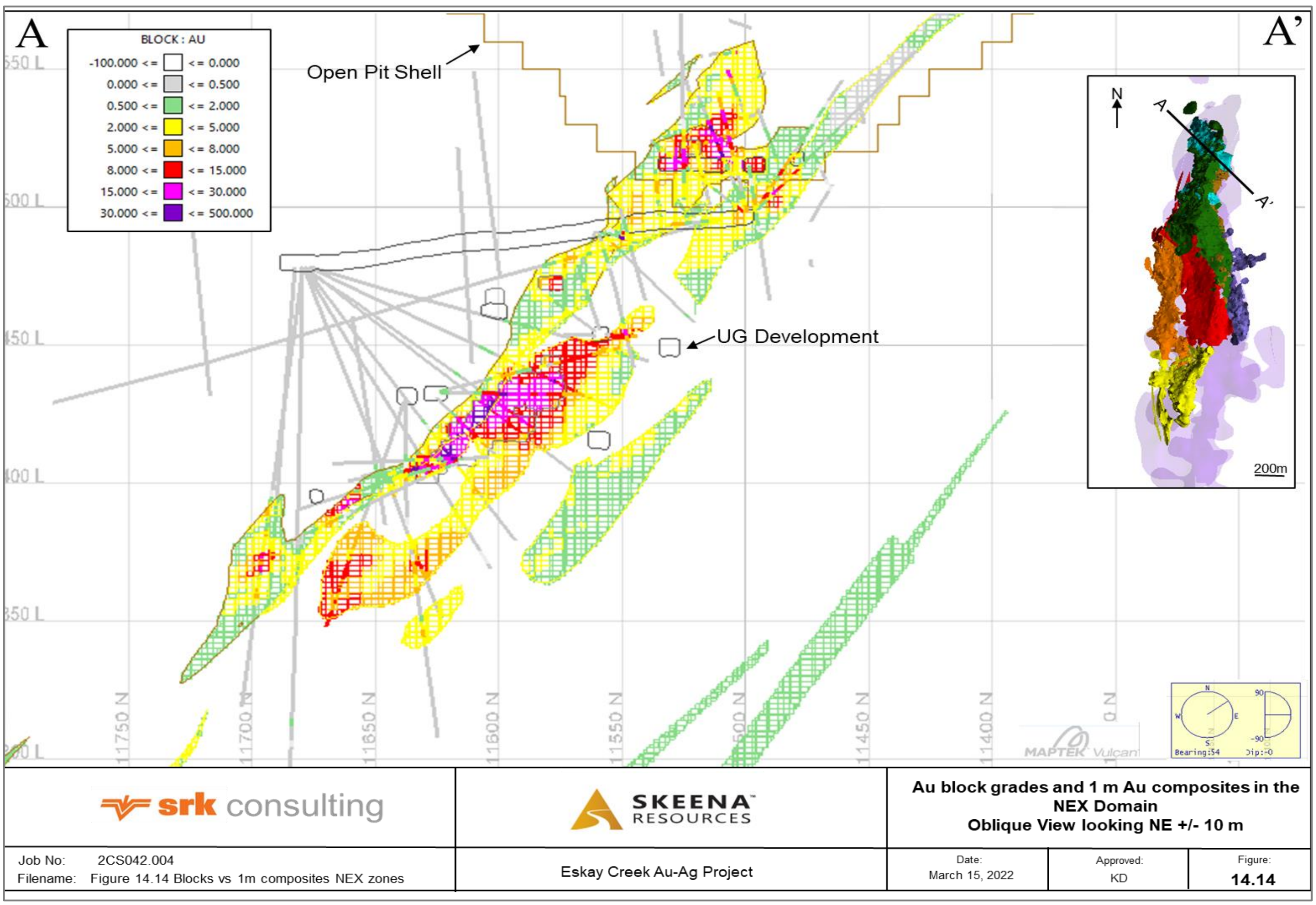
	Bearing	Plunge	Dip	Origin			End Offset			Block Size		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8508	-50	1188	3654	1500	3	3	2
Sub-block	90	0	0	9300	8508	-50	1188	3654	1500	1	1	1

Five domains were captured within the underground model: 22, HW, NEX, WT, and LP. OK was used to estimate gold and silver in all domain except the Even Lower Mudstone and Footwall Andesite in the LP domain. 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 two thirds of the variogram range, pass 2 equalled the variogram range and pass 3 equalled two and a half times the variogram range. Hard boundaries during interpolation were honoured. Hard boundaries were used for composites within the 1 m restriction domain to limit the effect of high-grade smearing from mined-out intervals. For pass 1, a minimum of 8 and maximum of 10 composites were used per block. For pass 2, a minimum of 5 and maximum of 15 composites were used per block, and for pass 3, a minimum of 3 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 14-14).

Figure 14-14: 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 ID2 and NN declustered models to assess for global bias. Although variable between zones, the overall bias was less than 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 14-12 and Table 14-13, the results are within acceptable limits.

Table 14-12: Global Validation of Gold

Zone	NN_Declus	AU_OK	AU_ID2	OK/ NN declus	OK / ID2
101	1.25	1.30	1.29	4%	1%
703	3.39	3.42	3.45	1%	-1%
801	2.55	2.59	2.59	1%	0%
802	5.53	5.42	5.51	-2%	-2%
811	2.72	3.00	2.91	10%	3%
90	0.88	0.89	0.86	1%	3%
91	0.74	0.72	0.73	-3%	-2%
92	0.89	0.89	0.88	0%	2%
93	0.89	0.79	0.78	-12%	1%
94	0.91	0.90	0.90	-1%	1%
				3%	**0%**

Table 14-13: Global Validation of Silver

Zone	NN_Declus	AG_OK	Ag_ID2	OK/ NN declus	OK / ID2
101	48.8	44.8	46.3	-8%	-3%
703	156.6	160.6	165.4	3%	-3%
801	59.5	61.6	61.1	4%	1%
802	244.4	237.7	237.6	-3%	0%
811	13.7	14.5	14.2	5%	2%
90	11.0	11.0	11.1	0%	0%
91	15.3	15.3	13.8	0%	11%
92	8.2	7.7	7.8	-6%	-1%
93	8.7	7.9	7.6	-9%	4%
94	4.3	3.6	3.6	-16%	0%
				0%	**-1%**

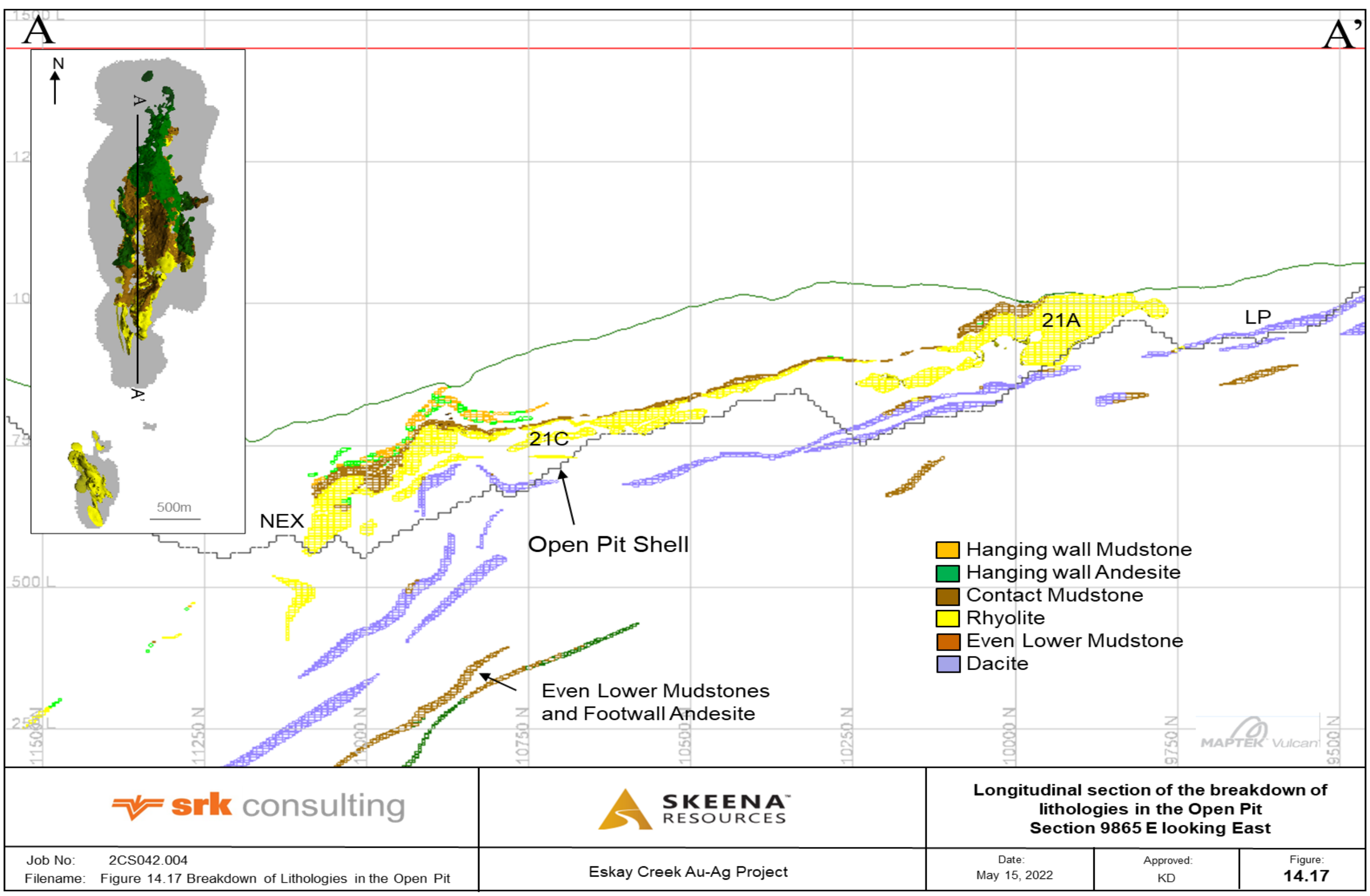
14.11.2.4 Underground Model – Swath Plots Summary

As part of the validation process, declustered composite samples (declustered NN model using 1 m blocks) and ID2 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.

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 2022 pit-constrained resource estimate indicates that on a tonnage weighted basis, 64% of the resource is hosted within the rhyolite facies with only 23% hosted in the remaining unmined mudstones/hanging wall andesite (Figure 14-15). Ten percent (10%) is hosted in the footwall Dacite. On an ounce-weighted basis, 55% of the pit-constrained resource estimate is contained within the rhyolite with the remaining 42% hosted within the unmined mudstones/hanging wall Andesite and 4% in the footwall Dacite.

Figure 14-15: Breakdown of Lithologies in the 21C, 21A, NEX and LP Domains (looking east)



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.

SRK 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, SRK considers that blocks estimated during the first estimation pass using a minimum of 4 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.

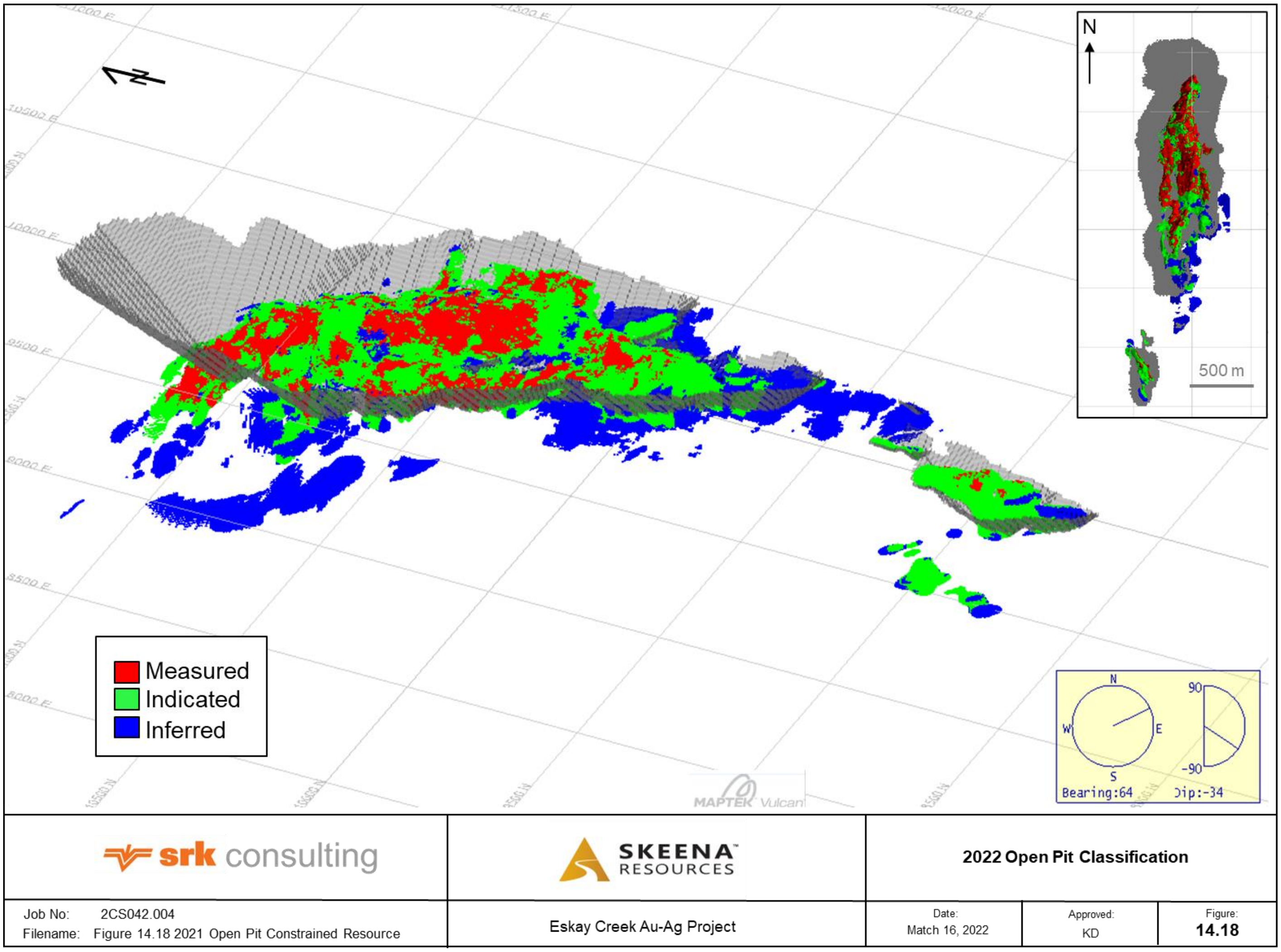
Mineralization in domains exhibiting good geological continuity estimated during pass 1 and 2 with a minimum of 3 holes were classified as Indicated. For the LP domain, an average distance of less 50 m was required as an additional constraint.

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 pass 3, using search distances of 2.5 times the variogram range, a KV of less than 0.8 and an average distance of less than 100 m 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.

Figure 14-16 shows the distribution of the measured, indicated, and inferred mineral resources in the pit-constrained model.

Figure 14-16: Long Section View of the Mineral Resource Classification in Blocks 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, PGeo (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 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 14-14). 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 14-14: 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
Gold Price	1,700	US Dollars Per Ounce
Silver Price	23	US Dollars Per Ounce
Selling Cost	25	US Dollars Per Ounce AuEq
Strip Ratio	7.55 : 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 and drift-and-fill underground mining scenarios. The parameters are summarized in Table 14-15.

Table 14-15: Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Mining costs	80	US$/t mined
Process cost	25	US$/t milled
General and Administrative	12	US$/t milled
All In Costs	117	US$/t milled

Parameter	Value	Unit
Process recovery Gold	90	Percent
Process recovery Silver	80	Percent
Sell Price Gold	1,700	US$/oz (95% Payable)
Sell Price Silver	23	US$/oz (95% Payable)
Transportation/Refining Costs	25	US$/oz 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 14-16 was determined to be 0.66 g/t AuEq; however, a pit constrained cut-off of 0.7 AuEq was selected for reporting the estimate. 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 long-hole mining 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 represented in Table 14-16 and the mineral resource amenable to underground mining are presented in Table 14-17. The mineral resource considered potentially amenable to underground mining are reported exclusive of Mineral Resources potentially amenable to open pit mining. mineral resources that are not mineral reserves do not have demonstrated economic viability. 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 14-16 presents the open pit-constrained resources at a 0.7 g/t AuEq cut-off outside of the 0.2 m exclusion zone and is shown in Figure 14-17. Table 14 25 shows the resources potentially amenable to underground mining methods above the 2.4 g/t AuEq cut-off, for long-hole mining, and 2.8 g/t AuEq cut-off, for drift-and-fill mining, outside the 1 m exclusion zone. The underground resource in shown in Figure 14-18. A full table showing the resources in the open pit model and underground model by domain is provided in Appendix A and B respectively.

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.

Table 14-16: Open Pit Constrained Mineral Resource Statement* Reported at 0.7 g/t AuEq Cut-Off Grade by Domain

Classification	Tonnes (kt)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq Oz(000)	Au (koz)	Ag (koz)
Measured	21,784	4.8	3.5	92.4	3,355	2,481	64,679
Indicated	24,724	2.3	1.8	37.6	1,804	1,400	29,896
Total M + I	**46,508**	**3.5**	**2.6**	**63.2**	**5,159**	**3,881**	**94,575**
Inferred	3,420	1.5	1.3	20.2	170	140	2,222

Table 14-17: 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

Classification	Tonnes (kt)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (koz)
Measured	737	6.1	4.6	112.7	145	109	2,671
Indicated	550	5.1	4.4	62.6	91	77	1,107
Total M + I	**1,287**	**5.7**	**4.5**	**91.3**	**236**	**186**	**3,778**
Inferred	330	4.1	3.5	42.6	43	37	452

Notes: To accompany the Mineral Resource estimate statement:

- 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 January 18, 2022
- 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 resources are exclusive of the 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 and zone 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 within the stope optimized shapes using a cut-off of 2.4 g/t AuEq for the long-hole mining scenario and 2.8 g/t AuEq for drift-and-fill mining scenario.
- Cut-off grades are based on a price of US$1,700 per ounce of gold, US$23 per ounce silver, 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 45 degrees
  - A reference mining cost of US$3.00 per tonne mined
  - A processing cost of US$15.50 per tonne processed
  - General and administrative costs of US$6.00 per tonne processed

- Mining dilution of 5%
- Mining recovery of 95%
- Transportation and refining costs of US$25 per ounce AuEq.

- Underground key assumptions for reasonable prospects for eventual economic extraction are as follows:
  - A reference mining cost of US$80 per tonne mined
  - A processing cost of US$25 per tonne milled
  - General and administrative costs of US$12 er tonne milled
  - All in costs of US$117 per tonne milled
  - Transportation and refining costs of US$25 per ounce 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.

Figure 14-17: Oblique view of Open Pit Mineral Resources at a 0.7 g/t AuEq Cut-off Grade

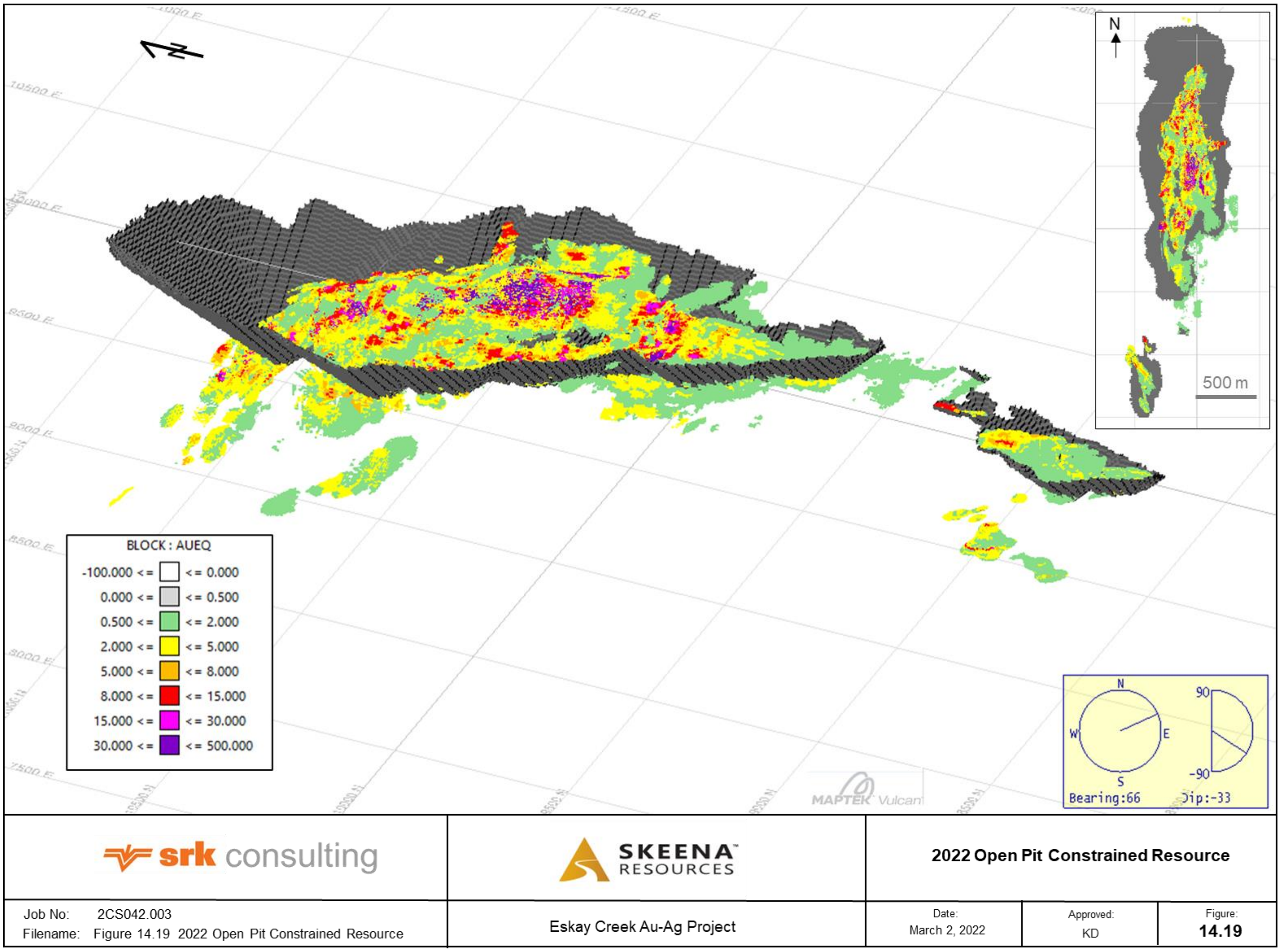
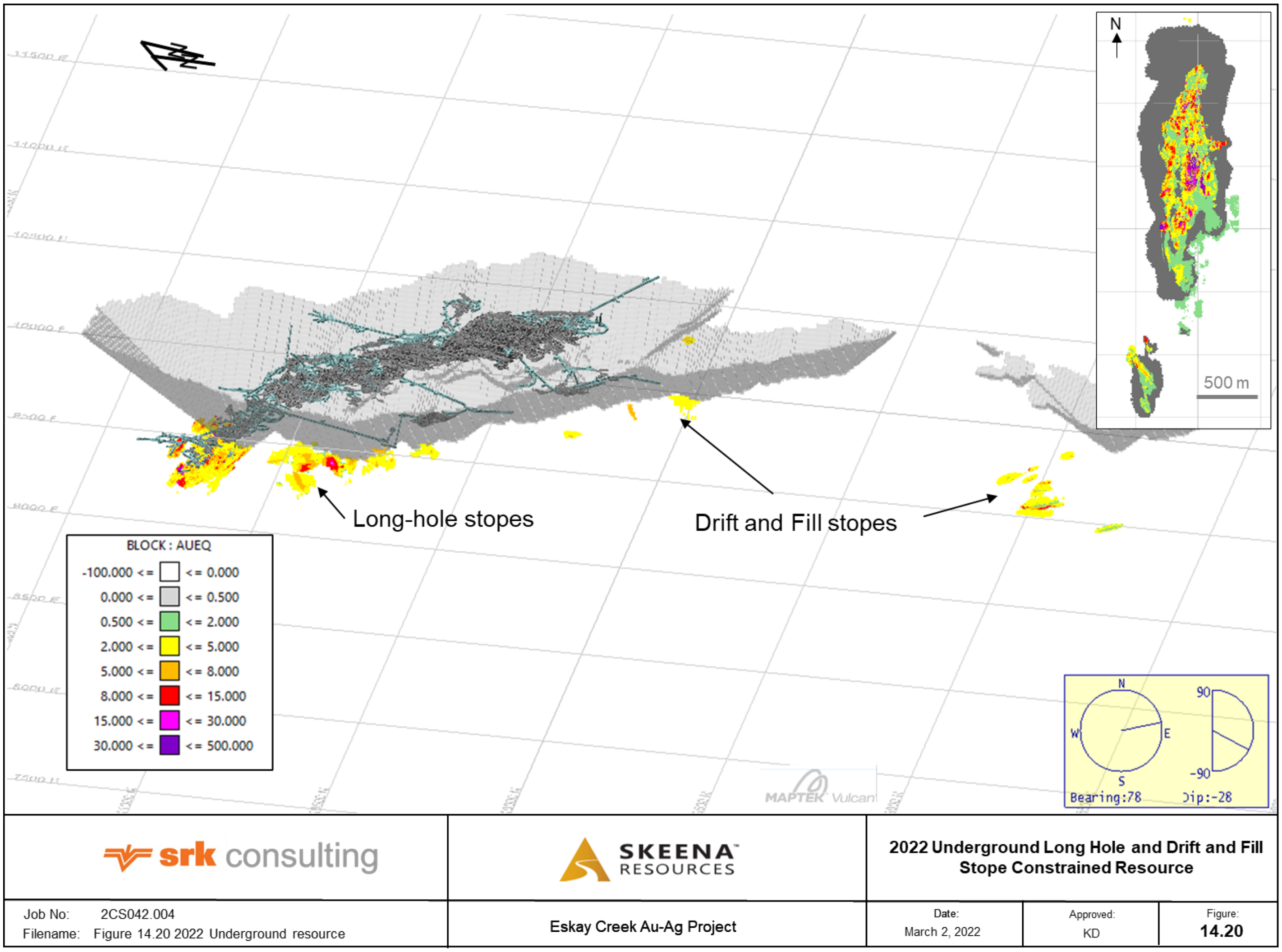


Figure 14-18: **Oblique View of Underground Mineral Resources Remaining at a 2.4 g/t AuEq Cut-off Grade for Long-hole Mining and 2.8 g/t AuEq Cut-off Grade for Drift-and-Fill Mining**

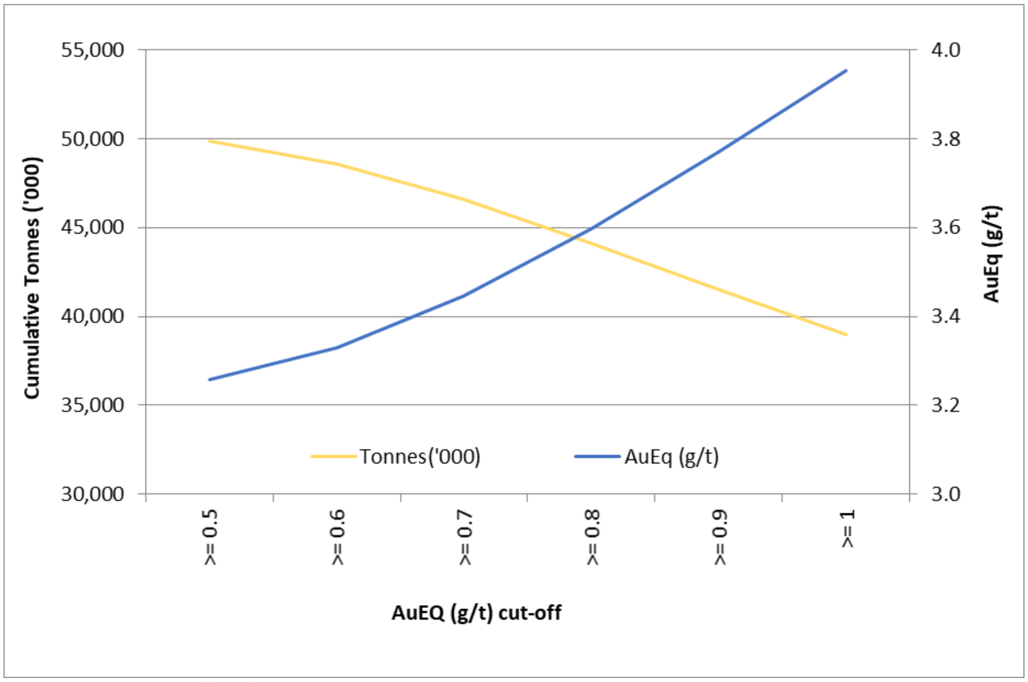


14.15 Grade Sensitivity Analysis

The Eskay Creek mineral resources were assessed in terms of cut-off grade selection by means of sensitivity analyses.

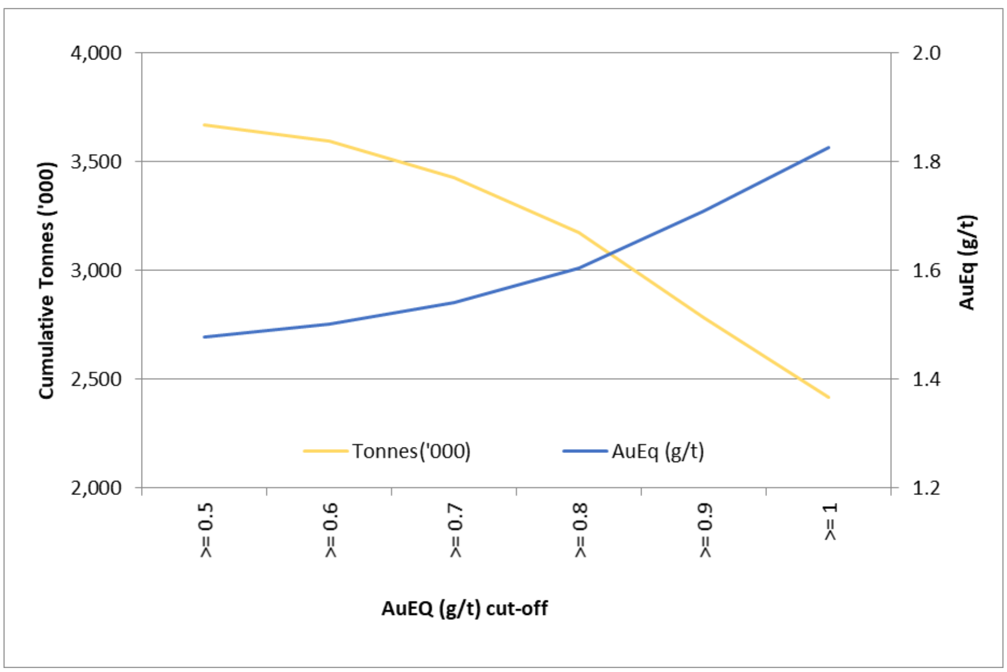
To illustrate this sensitivity, the global block model quantities and grade estimates are displayed at different cut-off grades in the open pit model as grade-tonnage curves in Figure 14-19 and Figure 14-20. The figures show that the resource is not sensitive to minor adjustments in cut-off grade selection as the average grade of the zones 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 14-19: Open Pit Model Measured + Indicated Grade-Tonnage Sensitivity Curve



Note: Figure prepared by Skeena, 2022

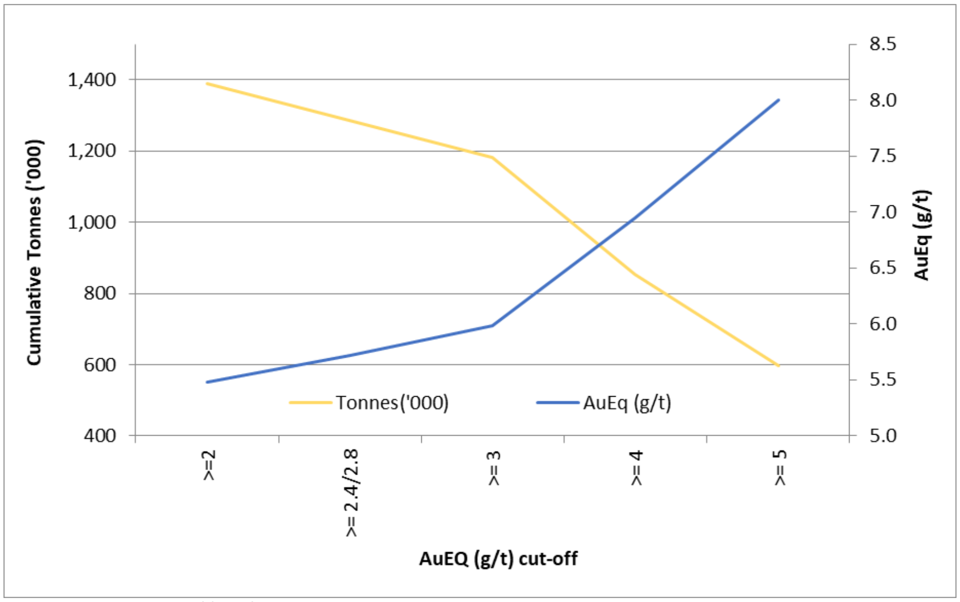
Figure 14-20: Open Pit Model Inferred Grade-Tonnage Sensitivity Curve



Note: Figure prepared by Skeena, 2022

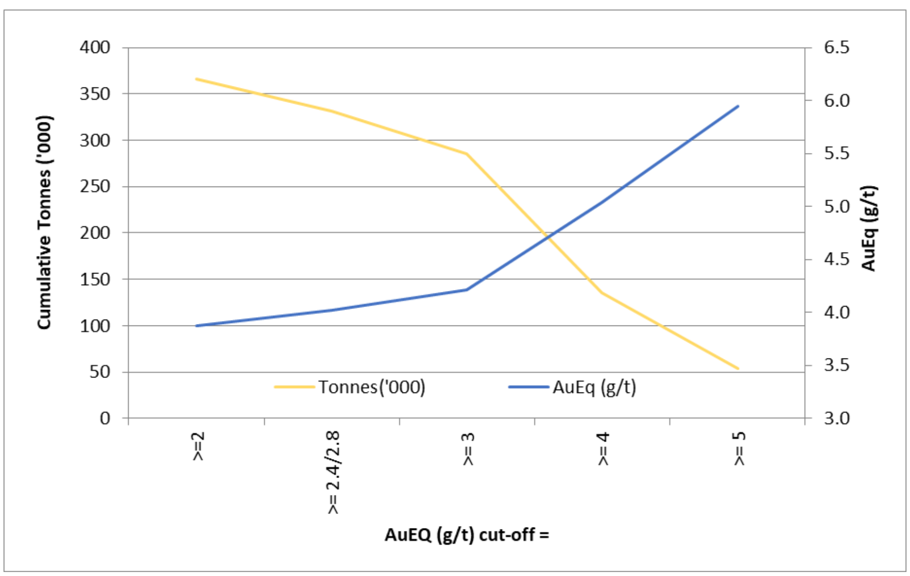
Figure 14-21 and Figure 14-22 presents global block model quantities and grade estimates within the underground model at different cut-off grades. The underground scenario is more sensitive to adjustments in cut-off grade selection due to the higher cut-off grades and selectivity of the mining methods. 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 14-21: Underground Model Measured + Indicated Grade-Tonnage Sensitivity Curve



Note: Figure prepared by Skeena, 2022

Figure 14-22: Underground Model Inferred Grade-Tonnage Sensitivity Curve



Note: Figure prepared by Skeena, 2022

14.16 Comparison to Previous Mineral Resource Model

The 2022 FS mineral resource represents a number of changes as compared to the 2021 PFS mineral resources due to the following key factors:

- The geological model and mineralization domains were updated
- The new MRE includes an additional 75 drill holes from the 22, 21A, 21E, HW, NEX, PMP and LP Domains;
- The open pit composites were increased from 2.0 m to 2.5 m
- Geostatistical methods (composites, variography) were all updated
- For the open pit block model, the parent block sizes were increased to 10 m x 10 m x 5 m from 9 m x 9 m x 4 m due to the change in bench heights from 8 m to 10 m
- The low-grade envelope surrounding the mineralization domains was removed in the 2022 model. Instead, all assays that showed continuity were included into new mineralization domains.
- The restriction domain to reduce the smearing of high-grade into neighbouring blocks was reduced to 1 m
- The 2022 estimate used specific gravity values based on zone and lithology mean values, whereas the 2021 specific gravity used the mean of the lithology.

Comparisons between the 2022 open pit, underground and combined mineral resources compared to the previous 2021 Mineral Resources are shown in Table 14-18, Table 14-19 and Table 14-20, respectively.

Overall, the measured and indicated categories in the 2022 open pit resource contains an increase of 24% tonnes and a decrease in grades of 17% AuEq, 16% Au, and 23% Ag. Overall, the AuEq and Au ounces increased by 1% and 3% respectively, while the contained silver ounces were reduced by 6%

The reason for the increase in tonnage and decrease in grade may be attributed to the following factors;

- Conversion of Inferred to Indicated material involved drilling into lower-grade areas (i.e., 22 Zone)
- Areas in the previous low-grade envelope > 0.7 g/t have been incorporated into mineralized wireframes where there is continuity
- The 2022 pit shell does not go as deep into the Au- and Ag-rich NEX Zone in the north.

The 2022 pit shell is much deeper and wider to the south where it incorporates lower grades of the LP domain,which was not included in the previous pit shell. It also includes the upper part of the Water Tower Zone. A comparison between the 2021 PFS Pit Shell and the 2022 FS Pit Shell is shown in Figure 14-23.

The underground measured and indicated resources show an increase of 51% tonnes and a change in grade of 0% AuEq, -10% Au, and +88% Ag. This resulted in an increase of 52% more AuEq ounces, 36% more Au ounces and 184% Ag ounces.

The increase in tonnes and ounces can be attributed to:

- the inclusion of more of the silver-rich NEX and HW Zones into the underground model, which had previously been included in the open pit model.

Table 14-18: Comparison of the 2021 vs 2022 Open Pit Constrained Mineral Resources

Resource Classification	Tonnes	Grade			Contained Ounces		
		AuEq	Au	Ag	AuEq	Au	Ag
Measured	26%	-17%	-17%	-22%	4%	7%	-2%
Indicated	22%	-21%	-18%	-28%	-5%	-3%	-13%
M & I	**24%**	**-17%**	**-16%**	**-23%**	**1%**	**3%**	**-6%**
Inferred	-35%	7%	30%	-19%	-26%	-20%	-47%

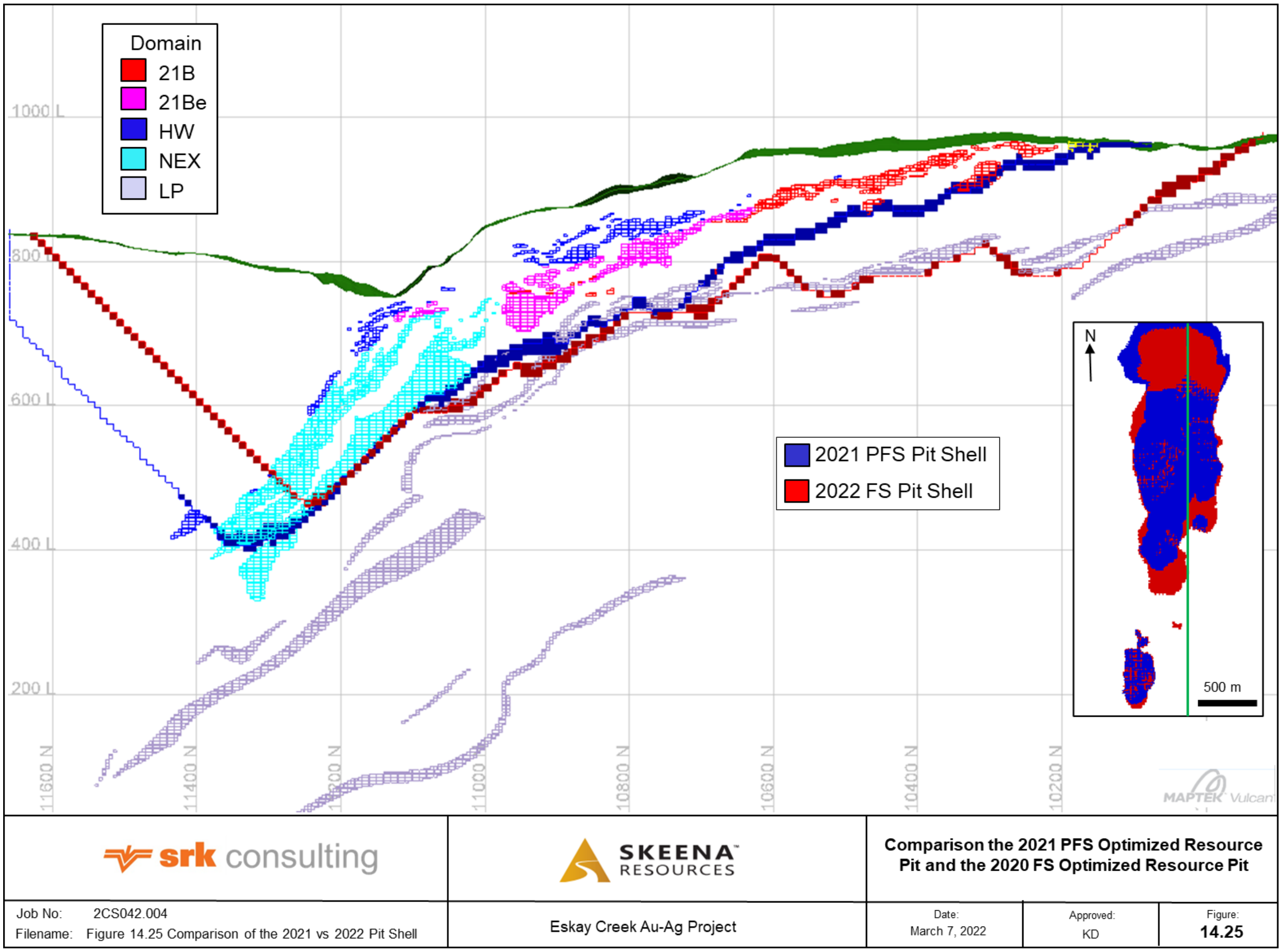
Table 14-19: Comparison of the 2021 vs 2022 Underground Mineral Resources

Resource Classification	Tonnes	Grade			Contained Ounces		
		AuEq	Au	Ag	AuEq	Au	Ag
Measured	114%	0%	-12%	67%	113%	88%	258%
Indicated	9%	-3%	-11%	75%	5%	-3%	90%
M & I	**51%**	**0%**	**-10%**	**88%**	**52%**	**36%**	**184%**
Inferred	-23%	-17%	-15%	-25%	-36%	-35%	-43%

Table 14-20: Comparison of the Total 2021 vs 2022 Mineral Resources (Open Pit and Underground Combined)

Resource Classification	Tonnes	Grade			Contained Ounces		
		AuEq	Au	Ag	AuEq	Au	Ag
Measured	28%	-16%	-15%	-21%	7%	9%	1%
Indicated	22%	-21%	-20%	-27%	-4%	-3%	-11%
M & I	**24%**	**-18%**	**-16%**	**-22%**	**3%**	**5%**	**-3%**
Inferred	-33%	8%	16%	-18%	-28%	-23%	-45%

Figure 14-23: Comparison between the 2021 PFS Resource Pit Shell and the 2022 FS Resource Pit Shell



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).

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.

14.17.1 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 14-21.

Table 14-21: Epithermal, Base Metal, and Metallurgical Concentrations Remaining in Each of the Domains Within the Open Pit Shell at 0.7 g/t AuEq

DOMAIN	AS_PPM	HG_PPM	SB_PPM	Lead %	Copper %	Zinc %	S_PPM	FE_%
22	688	6	240	0.097	0.015	0.162	9,211	1.2
21A	1,395	95	1,163	0.096	0.017	0.167	15,297	1.5
21C	337	13	455	0.182	0.044	0.326	16,016	1.9
21B	468	90	1,894	0.367	0.070	0.611	17,514	1.8
21Be	1,331	68	1,548	0.804	0.139	1.311	19,371	2.3
21E	409	14	2,389	0.071	0.022	0.153	15,676	2.3
HW	610	24	1,312	1.348	0.203	2.013	33,188	4.2
NEX	414	18	655	1.020	0.125	1.543	29,881	3.4
WT	780	9	339	0.086	0.023	0.166	15,989	2.3
LP	696	15	219	0.437	0.018	0.654	67,768	6.8
PMP	484	17	1,096	0.096	0.030	0.175	10,447	1.2
109	648	14	252	1.333	0.027	2.063	41,320	5.0
Average	**704**	**41**	**917**	**0.443**	**0.062**	**0.696**	**26,302**	**2.9**

15 MINERAL RESERVE ESTIMATES

15.1 Overview

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 was received on November 10, 2021 and 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. The conversion of Mineral Resources to Mineral Reserves included the use of more detailed pit slope geotechnical parameters, waste contact dilution, and NSR calculations based on lower metal prices and recoveries by mill feed type.

The initial step for reserve conversion was to develop a Net Smelter Return (NSR) block unit value for Measured and Indicated open pit resource blocks. The NSR calculation was based on the parameters in Table 15-1 and is applicable for a gold concentrate. Geotechnical parameters for seven slope design sectors were then coded into the reserve model based on a geotechnical assessment of preliminary pit designs. Economic pit shells were then generated using the NSR block values, pit sector slope criteria, and associated mining, process and G&A costs. Detailed open pit phases designs were created and re-assessed for geotechnical stability. In order to access the northern end of the deposit, a water diversion tunnel was developed as a sustaining capital expense. A detailed mine production schedule was then developed using diluted reserves and costed to support the financial evaluation of the project.

15.2 Geotechnical Considerations

Based on the available geotechnical and hydrogeological data, the recommended open pit slope design criteria are based on kinematic sectors and are summarised Sections 16.3 and 16.8. Recommended inter-ramp slope angles are illustrated on the feasibility study pit in Figure 16-1 and range from 26° to 51°. Maximum inter-ramp stack heights should be limited to approximately 80 m in toppling-controlled sectors and 120 m in other sectors. Inter-ramp stacks should be separated by geotechnical berms or ramps that are a minimum of 30 m wide. 20 m high double benches are likely achievable in all sectors, with recommended catch bench widths ranging from 12.7 m to 37.5 m. The slope design criteria assume that controlled blasting will be implemented. Scaling bench faces and cleaning accumulated material from bench toes is recommended. Slope depressurisation will be required in the north, east and south walls of the North pit to meet the design acceptance criteria in these slopes.

15.3 Economic Pit Shell Development

The final pit designs are based on pit shells using the Lerchs–Grossmann (LG) procedure in Hexagon MinePlan software. The parameters for the pit shells are shown in Table 15-1.

Table 15-1: Pit Optimization Parameters

Description	Units	Value	Gold Value	Silver Value
Exchange rates				
C$	US$ =	1.26		
Resource Model				
Block classification used		M+I		
Block Model height	m	10		
Mining Bench height	m	10		
Metal Prices				
Price	$/oz		1550	**20**
Royalty	%		2%	2%
Smelting, Refining, Transportation Terms				
Concentrate grades	g/t Au		20 - 70	
Payable	%		69 - 92.5	75 - 90
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	148.00		
Metallurgical Information - Rhyolite and Contact Mudstone				
Recovery	%	88.88 * (1-exp ([Au Feed g/t] * (-1.5669)))		
Mass pull				
Au feed < 0.5 g/t		0.5		
0.5 < Au feed < 3.5 g/t		1.1867*(Au Feed [g/t]) + 1.6069		
3.5 < Au feed < 8 g/t		4.275*(Au Feed [g/t]) – 9.204		
Au feed > 8 g/t		25		
Au con	g/t	Recovery*(Au Feed [g/t])/(Mass Pull)		
Metallurgical Information - HW, Mudstone, default				
Recovery	%	62.74 * (1-exp ([Au Feed g/t] * (-1.0706)))		
Au con	g/t	4.9359* (Au Feed [g/t]) + 1.7253		
Mass pull		Recovery*(Au Feed [g/t])/(Au Con)		
Power Cost				
Cost of power	C$/kWhr	$0.05		
Fuel Cost				
Diesel fuel cost to site	C$/L	$1.31		
Mining Cost *			NAG	PAG

Description	Units	Value	Gold Value	Silver Value
Waste base rate - 880 elevation	C$/t		3.02	3.71
Incremental rate - above	C$/t/10m bench		-0.018	-0.018
Incremental rate - below	C$/t/10m bench		0.041	0.041
Mill feed base rate - 880 elevation	C$/t		2.43	2.43
Incremental rate - above	C$/t/10 m bench		0.020	0.020
Incremental rate - below	C$/t/10 m bench		0.034	0.034
Processing **				
Processing cost	C$/t mill feed	14.18		
Maintenance (incl. road and bridges)	C$/t mill feed	4.04		
Total processing cost	C$/t mill feed	18.22		
General and Administrative Cost				
G&A cost	C$/t mill feed	6.23		
Total Process and G&A				
Process + G&A	C$/t mill feed	24.45		

Note: * mining costs based on using 144 t haul trucks. ** process costs based on 3 Mt/a dry throughput

Ultimate pits were generated using a revenue factor of 0.9 or metal price of $1,395 /oz. These were used as the basis for the design.

15.4 NSR Cut-off

For the statement of open pit reserves for the Eskay Creek Project, an NSR value per tonne of C$24.45/t was used as the mill feed cut-off. NSR calculations are inclusive of all revenues and royalties for the gold concentrate. Revenues are based on contributions of both gold and silver metals.

No underground reserves are stated in this report but remain an opportunity for future development.

15.5 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 high-grade is restricted to within a 1 metre buffer to reduce grade smearing into neighbouring blocks. 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. A low-grade envelope was included in the resource estimate so that neighbour block grades could be used in dilution calculations.

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 10 x 10 m in plan view, and 10 m high. Mining would be completed on 10 m lifts for waste and 5 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 mineralisation, proposed grade control methods, GPS-assisted digging accuracy, and blast heave.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 19.7% more tonnes and 15.6% 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 0.19 g/t Au and 3.71 g/t Ag. AGP considers these dilution percentages to be reasonable considering the expected seasonal working conditions as well as mining through underground workings.

15.6 Pit Design

Pit designs were developed for the north and south pit areas. The initial phases (Technical Sample, Quarry 1 and Quarry 2) were designed for the purpose of obtaining a technical sample and necessary NAG waste material to create supporting infrastructure. The north pit will consist of an additional three main phases, while the south pit will only contain a single small phase. The pit optimisation shells used to determine the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. Mining occurs on 10 metre benches with catch benches spaced 20 metres vertically. The haul roads are 30.2 m in width with a road grade of 10%. The south pit is significantly smaller than the north pit and is likely to be mined near the end of the mine schedule. The south pit generally has harder rock and lower gold grades. Rhyolite is the dominant rock type that will remain in the mined-out pit walls before reclamation. A summary of the phase tonnes and grades is displayed in Table 15-2.

Table 15-2: Phase Tonnes and Grade Summary

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
Quarry1	0.00	0.00	0	0.00	0.00	0.00	0	0	0	0.0	0.0	0.37	0.4	0.0
Quarry2	0.00	0.00	0	0.00	0.00	0.00	0	0	0	0.0	0.0	3.69	3.7	0.0
TS	0.02	6.48	57	0.03	0.08	0.02	5,796	147	603	1.8	1.9	1.04	1.1	48.0
Phase 1	4.65	3.51	61	0.09	0.16	0.02	1,924	135	1,619	1.6	1.5	22.51	27.2	4.8
Phase 2	9.64	3.70	118	0.70	1.11	0.10	659	74	1,900	2.1	2.9	96.75	106.4	10.0
Phase 3	13.42	2.49	59	0.46	0.74	0.07	382	15	569	1.7	2.3	93.80	107.2	7.0
South Phase 1	2.18	1.74	59	0.07	0.10	0.01	744	5	238	0.6	1.1	5.34	7.5	2.4
Total	29.91	2.99	78.5	0.45	0.72	0.07	741	52	1,137	1.7	2.3	223.5	253.4	7.5

Mine planning indicates that the northern end of the open pit will intersect Tom MacKay Creek requiring the provision of a water diversion tunnel to re-route flowing water around the open pit before re-entering the existing creek downstream. AGP has worked with Swiftwater Consulting Ltd. (Swiftwater) to develop the initial technical design for the water diversion. Minimum tunnel dimensions have been selected as 4.7 m wide by 4.7 m high in order to accommodate the expected water flows. The full length of the tunnel is 1214 metres. Starting from the tunnel inlet, 802 metres are at -2% gradient, 362 metres at -18.5% gradient, and 50 metres at -2% nearest the outlet.

The tunnel will be constructed using drill-and-blast methodology with a mean overbreak thickness of 200 mm occurring outside of the nominal tunnel dimensions. It has been determined that the tunnel must include a liner covering 100% of the tunnel wall to mitigate against water contact with potentially acid generating (PAG) rock, as well as provide additional structural support. Further discussion is included in Section 16.10.

15.7 Mine Schedule

The mine schedule plans to deliver 29.91 Mt of mill feed grading 2.99 g/t gold and 78.5 g/t silver over a mine life of eight years. Mill feed will continue in Year 9 with material reclaimed from stockpiles. Waste tonnage from the pits totalling 223 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is 7.5:1.

The mine schedule initially assumes a maximum of 3.0 Mt/a of feed will be sent to the process facility using a suitable ramp-up in Year 1. The mill throughput is increased to 3.7 Mt/a m in Year 6 and continues until Year 9. To maintain these mill throughput tonnages, a minimum proportion of 55% rhyolite tonnes was targeted in the mill feed to improve material flow in mill circuits. 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 proposed mine life includes three years of pre-stripping and eight years of mining. While there will be no mining in Year 9, the process plant will continue to operate with feed being reclaimed from ore stockpiles. Mill feed will be stockpiled during the pre-production years. A technical sample and two small quarries will be mined in pre-production so that process performance of the mill can be evaluated with a large representative feed sample of approximately 10 kt.

15.8 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 reserves for the Eskay Creek Project, an NSR value per tonne of C$24.45/t was used as the mill feed cut-off. NSR calculations are inclusive of all revenues and royalties for the gold concentrate. Revenues are based on contributions of both gold and silver metals. Revenues are based on contributions of both gold and silver metals. A marginal NSR cut-off of C$24.45/t was used to flag initial feed and waste blocks prior to dilution and represents the preliminary process and site G&A costs.

This estimate has an effective date of 30 June 2022. 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	17.3	3.64	99	4.92	2.02	55.1	2.73
Probable	12.6	2.10	50	2.75	0.85	20.5	1.12
Total	**29.9**	**2.99**	**79**	**4.00**	**2.87**	**75.5**	**3.85**

*Note: This mineral reserve estimate has an effective date of June 30, 2022 and is based on the mineral resource estimate dated January 18, 2022 for Skeena Resources by SRK Consulting (which has been updated since the PFS). 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$1,550/oz gold price and US$20.00/oz silver price. An NSR cut-off of C$24.45/t was used to define reserves based on preliminary processing costs of $18.22/t ore and G&A costs of C$6.23/t ore. The metallurgical recoveries varied according to gold head grade and concentrate grades. Gold and silver recoveries were approximately 83% overall during the LOM scheduling. Final operating costs within the pit design were C$3.72/t mined, with associated process costs of C$16.91/t ore and G&A costs of C$4.20/t ore.

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

Risks that could materially affect the reserve include mining selectivity near the ore contacts, NAG/PAG delineation during mining and assumed process recoveries for given rock types. These are considered manageable risks which will be mitigated as more test work and operating experience is obtained.

16 MINING METHODS

16.1 Overview

Open pit mining was selected for the FS, 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 FS. There is still potential for the inclusion of underground mining in future mining studies.

The mine plan is based on Proven and Probable Reserves. Inferred Mineral Resources are too speculative geologically to have economic considerations applied to them, so are treated as waste in this FS.

16.2 Geological Model Importation

The 2021 resource estimates were created using Leapfrog software for mineralization domains and Vulcan software for FS block modelling. SRK provided Skeena with support and review of the updated resource model, together with a resource estimate completed in accordance 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 FS, using their Lerchs Grossmann (LG) shell generation, pit and WRSF design and mine scheduling tools.

Table 16-1: Open Pit Model Framework

Framework Description	Skeena Resource Open Pit Model (Value)	Final FS Open Pit Model (Value)
MinePlan file 10 (control file)	opt10.dat	ecfs10.dat
MinePlan file 15 (model file)	opt15.dat	ecfs15.m02
X origin (m)	9,300	9,300
Y origin (m)	8,500	8,500
Z origin (m) (max)	1450	1450
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	120	120
Number of blocks in Y direction	370	370
Number of blocks in Z direction	150	150
X block size (m)	10	10
Y block size (m)	10	10
Z block size (m)	10	10

Table 16-2: Resource Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
AUOK	0	420	0.001	g/t	Gold grade
AGOK	0	16000	0.1	g/t	Silver grade
PBID	0	16	0.0001	%	Lead grade
ZNID	0	23	0.0001	%	Zinc grade
CUID	0	5	0.0001	%	Copper grade
ASID	0	270000	1	ppm	Arsenic grade
HGID	0	15,000	0.1	ppm	Mercury grade
SBID	0	230,000	1	ppm	Antimony grade
SID	0	300,000	1	ppm	Sulphur grade
FEID	0	25	0.0001	%	Iron grade
SG	0	4	0.0001	g/cm^3	Density
ZONE	0	820	1	-	Domain divided by rock type (1=rhyolite; 2=mudstone; no Lower Package)
ESTZN	0	980000	1	-	Estimation zone by domain, rock type and orientation of limb
DOMAN	0	99	1	-	Grouped domain that fits the historical mined areas (approximately)
ROCK	0	9	1	-	Rock type
MINED	0	1	1	-	MINED (1=mined out, 0=remaining), for reference only
RESAT	0	3	1	-	Resource category where 1=Measured; 2=Indicated; 3=Inferred
AUEOK	0	550	0.001	g/t	Gold equivalent grade
AUFES	0	120	0.001	-	Au/(Fe+S) ratio
SPCT	0	27	0.0001	%	Sulphur grade

Field Name	Min	Max	Precision	Units	Comments
TRIZN	0	71000	1	-	Broken down according to Domain, rock type and orientation of limb and within individual triangulation
ORE%	0	100	0.01	%	Percentage of ore in the block, adjusted for underground workings.
PAG%	0	100	0.01	%	Total percent within PAG solids
PAGNG	0	2	1	-	PAG value (1=PAG,2=NAG), PAG coded using a minimum tolerance of 45%

Table 16-3: Open Pit Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
AU	0	420	0.001	g/t	Gold grade
AG	0	16000	0.1	g/t	Silver grade
PB	0	16	0.0001	%	Lead grade
ZN	0	23	0.0001	%	Zinc grade
CU	0	5	0.0001	%	Copper grade
AS	0	270000	1	ppm	Arsenic grade
HG	0	15000	0.1	ppm	Mercury grade
SB	0	230000	1	ppm	Antimony grade
S	0	300000	1	ppm	Sulphur grade
FE	0	25	0.0001	%	Iron grade
SG	0	4	0.0001	g/cm^3	Density
DOMAN	0	99	1	-	Grouped domain that fits the historical mined areas (approximately)
ROCK	0	9	1	-	Rock type
MINED	0	1	1	-	MINED (1=mined out, 0=remaining), for reference only
RESAT	0	9	1	-	Resource category where 1=Measured; 2=Indicated; 3=Inferred
AUEQ	0	550	0.001	g/t	Gold equivalent grade
AUFES	0	120	0.001	-	Au/(Fe+S) ratio
SPCT	0	27	0.0001	%	Sulphur grade
ORE%	0	100	0.01	%	Percentage of ore in the block, adjusted for underground workings.
PAG%	0	100	0.01	%	Total percent within PAG solids
ARD	0	2	1	-	Acid rock drainage value (1=PAG,2=NAG), PAG coded using a minimum tolerance of 45%
TOPO%	0	100	0.01	%	Percentage of block below topography
NSR1	0	100000	0.01	C$/t	Net Smelter Return - PFS parameters except 1550Au
TMP1	0	100000	0.01		Temporary item for debugging python 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)
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)

Field Name	Min	Max	Precision	Units	Comments
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/m^3	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
SLOPE	0	10	1	-	Slope domain: 1= weak slope, 2=competent slope
NSR2	0	100000	0.01	C$/t	Net Smelter Return - PFS parameters
CON2	0	99	1	g/t	Gold concentrate grade for NSR2 (20-45 g/t)
SLOPN	0	10	1	-	Slope domain: 1= weak, 2=competent, 3=weak north ext, 4=competent north ext
VLT1	-1000	20000	0.01	C$/t	Value per tonne for run 1 pit shells using NSR1
VLB1	-30000	9000000	1	C$	Value per block for run 1 pit shells using NSR1
RSCOD	0	1	1	-	Restricted mining area (0=south of Tom Mackay Creek, 1=north of Tom Mackay Creek)
VLT2	-1000	20000	0.01	C$/t	Value per tonne for run 2 pit shells using NSR1 (river restriction)
VLB2	-30000	9000000	1	C$	Value per block for run 2 pit shells using NSR1 (river restriction)
PFS	0	100	0.01	%	Percentage of block within final PFS pit solids
BERM	0	99	0.01	m	Berm width for pit design
NSR3	0	100000	0.01	C$/t	Net Smelter Return - MI, Dec.2021 recoveries, June2021 OM terms
SLP	0	99	1	-	Slope domains coded using Dec. 2021 BGC solids
VLT3	-1000	20000	0.01	C$/t	Value per tonne for run 3 pit shells using NSR3 (Dec recoveries, slopes, and OM terms)
VLB3	-30000	9000000	1	C$	Value per block for run 3 pit shells using NSR3 (Dec recoveries, slopes, and OM terms)
NSR4	0	100000	0.01	C$/t	Net Smelter Return - MI, Dec.2021 recoveries, June2021 OM terms (no penalties)
VLT4	-1000	20000	0.01	C$/t	Value per tonne for run 4 pit shells using NSR4
VLB4	-30000	9000000	1	C$	Value per block for run 4 pit shells using NSR4
VLT5	-1000	20000	0.01	C$/t	Value per tonne for run 5 pit shells using NSR4 (PFS slopes)
VLB5	-30000	9000000	1	C$	Value per block for run 5 pit shells using NSR4 (PFS slopes)
NSR6	0	100000	0.01	C$/t	Net Smelter Return - MI, Dec.2021 recoveries (CM same as rhyolite), June2021 OM terms
VLT6	-1000	20000	0.01	C$/t	Value per tonne for run 6 pit shells using NSR6

Field Name	Min	Max	Precision	Units	Comments
VLB6	-30000	9000000	1	C$	Value per block for run 6 pitshells using NSR6
AUEQR	0	99000	0.01	oz	Recovered gold equivalent ounces (using diluted grades)

16.3 Open Pit Slope Design Criteria and Diversion Tunnel Geotechnical Assessments

16.3.1 Introduction

A geotechnical assessment has been undertaken for the proposed feasibility-level open pit slopes and diversion tunnel designs for the Eskay Creek mining project being explored by Skeena Resources Limited (Skeena). The project targets a deposit that will be mined via a 260 m deep North pit and 80 m deep South pit. A diversion tunnel is proposed to divert flows from the Tom MacKay Creek around the north pit boundary. BGC Engineering Inc. (BGC) has undertaken this work at the request of AGP Mining Consultants Inc. (AGP) to support this economic Feasibility Study (FS) of the Eskay Creek project. This section provides feasibility-level slope design criteria for the open pit. This section also summarizes geotechnical assessments carried out for the proposed diversion tunnel. The diversion tunnel assessments are not considered at a feasibility-level as they are included in sustaining capital expenditures.

16.3.2 2021 Qualified Person Inspections

Mr. Ian Stilwell, P.Eng. and Ms. Catherine Schmid, P.Eng. of BGC conducted Qualified Person (QP) inspections of the project site from September 1 to 3, 2021. Inspections supported the current study and involved site reconnaissance and an audit of the 2021 geotechnical drilling investigation, which was being carried out by Ausenco personnel.

16.3.3 Sources of Data

BGC relied on the following information sources to inform the open pit and diversion tunnel geotechnical assessments:

- A geotechnical drilling investigation completed by Ausenco Engineering Canada Inc. (Ausenco) in 2021 to support the current study. The program consisted of 14 inclined boreholes at depths between approximately 71 m and 222 m. Holes were distributed within the perimeter of the proposed North pit walls and along the diversion tunnel alignment at a variety of different azimuths, with inclinations ranging from approximately 60° to 90°. Drilling activities included geomechanical and oriented core logging, packer testing, and point load index testing and were carried out by Ausenco. Laboratory sample selection was carried out by Ausenco with guidance provided by BGC. Geotechnical laboratory testing was completed by Queen's University, Saskatchewan Research Council and BGC.

- Historical geotechnical and exploration drilling data, including 12 inclined geotechnical drillholes from 2020 located in the North and South pit walls with geomechanical and oriented core logging, point load index testing and laboratory testing, infrastructure foundation drillholes from 2020 with laboratory testing data, and exploration drillholes drilled between 2018 and 2021 with partial geomechanical logging data collected.

- The 2021 three dimensional (3D) Eskay Creek geology model developed by Skeena.

- Technical reports prepared by others documenting regional geology, proposed open pit drill-and-blast parameters and site-specific seismic hazard analysis.

BGC performed quality assurance and quality control checks on geomechanical and oriented core logging data prior to their use. Checks included a desktop review of logged rock mass rating parameters, spot checks of core box and run photographs, and a review of discontinuity orientation measurements and calculations.

The prefeasibility-level pit plan, dated 8 June 2021, formed the basis for BGC's provisional slope design recommendations that AGP used to produce the FS-level pit plan, dated January 4, 2022. BGC carried out inter-ramp and overall slope stability analyses on the FS-level pit plan. Diversion tunnel geometry and geotechnical assessments were based on the February 8, 2022, proposed tunnel alignment prepared by AGP.

16.3.4 Geotechnical Model

BGC developed a geotechnical model that characterizes the rock mass conditions, structural geology, hydrogeology, and seismicity of the open pit and diversion tunnel areas. This model was used as a basis for the open pit and diversion tunnel geotechnical assessments.

The rock mass model is based on data from drillhole logging, laboratory testing, the Eskay Creek geology model and relevant background reports. Ten geotechnical units of similar rock mass properties were identified. Rock mass quality ranges from 'fair' in the bedded mudstone units in the upper walls of the north pit and diversion tunnel, to 'good' in volcanic units, including the Hanging Wall Andesite unit that is interlayered with mudstone in the upper north pit walls, and the Rhyolite and Footwall Dacite units in the lower and south walls of the north pit and all walls of the south pit. Fault Zones are associated with decreased rock mass quality and are interpreted to be generally less than 5 m wide. A layer of soil overburden up to 10 m thick is interpreted to overlie the open pit and diversion tunnel area.

The structural geology model includes 3D modeled faults and discontinuity fabric, which consists of bedding joints, fault-parallel discontinuities and joints. Sources of structural fabric data included oriented drill core, 3D model surface orientations and surface mapping. Structural domains were defined to represent areas with similar structural conditions (i.e., discontinuity set orientations). Faults of the 3D model and geologic contacts were defined as structural domain boundaries. A total of 11 structural domains were identified in the open pit and diversion tunnel areas.

The conceptual hydrogeological model carried forward for the open pit and diversion tunnel area is based on BGC's understanding of groundwater conditions in the open pit area.

A site-specific seismic hazard analysis conducted by Ausenco in 2021 identified that seismicity in the Eskay Creek area is related to a complex offshore fault system resulting from strains on the Queen Charlotte Fault. Based on the results of the probabilistic seismic hazard analysis, BGC considered a design peak ground acceleration of 0.079g, corresponding to a 1 in 2,475-year return period, to represent the design value for pseudo-static analyses of the open pit and diversion tunnel portals.

16.3.5 Geotechnical Model Limitations

The data available at this stage of study vary in reliability. Where data gaps exist, the geotechnical model has been inferred from available data. Estimates of engineering properties are provided with ranges where possible and sensitivity analyses are encouraged for mine design based on these data. The geotechnical model interpretations require additional validation and testing with higher data density before they can be used for detailed design.

16.3.6 Open Pit Mining Geotechnical Assessment

To inform the geotechnical design recommendations for the open pit slopes, BGC conducted geotechnical analyses of inter-ramp scale, bench scale and overall scale slopes. BGC considers the following design acceptance criteria to apply to the open pit slopes, based on industry standards:

- Inter-ramp scale slopes should achieve a minimum factor of safety of 1.2 under static conditions and 1.0 under pseudo-static conditions
- Bench scale slopes should achieve a catch bench width that meets minimum requirements for rock fall mitigation and provincial regulations. A minimum reliability of 50%, based on potential failure modes having a factor of safety of 1.0 or less, is considered acceptable.
- Overall scale slopes should achieve a minimum factor of safety of 1.3 under static conditions and 1.05 under pseudo-static conditions
- If slope instabilities have the potential to impact the diversion tunnel, they should achieve a minimum factor of safety of 1.5.

Inter-ramp scale kinematic analyses were performed in each structural domain to identify plausible planar, wedge, and toppling instability modes formed by the combination of discontinuities and the pit wall orientation. Based on the results, the structural domains in the pit wall were subdivided into "kinematic sectors" with similar kinematic controls. Toppling was found to be the primary kinematic control in the majority of the east-dipping walls of the North and South pits, as well as in several of the west-dipping walls of the north pit. Sliding along bedding-parallel joints was identified as the critical kinematic control in northwest- to northeast-dipping walls in hanging wall structural domains within the North pit, and in west-dipping walls of the South pit. The remaining pit walls are either controlled by wedge instabilities formed by the intersection of multiple discontinuity sets or are not kinematically controlled.

Bench scale kinematic analyses were completed to estimate the effective bench face angles that can be expected during mining. Back-break of the bench crest was estimated from a probabilistic analytical model that incorporated the discontinuity sets developed for each structural domain and their characteristics, a double bench height of 20 m, and a pre-split (design) bench face angle of 80°. The predicted back-break widths in the double benched open pit walls range from 3.6 to 8.7 m, which correspond to effective bench face angles between 59° and 70°.

Based on the results of the bench scale and inter-ramp kinematic analyses, BGC prepared provisional recommended slope design criteria, which were then incorporated into the FS mine plan by AGP. BGC then carried out limit equilibrium inter-ramp and overall slope stability analyses on representative cross sections through the FS-level pit plan. Stability analyses indicate that the slopes of the FS pit meet the design acceptance criteria with horizontal depressurization 40 m behind the pit face in the east walls of the North pit, and 20 m behind the pit face in the north and south walls of the North pit. No depressurization was required in the South pit.

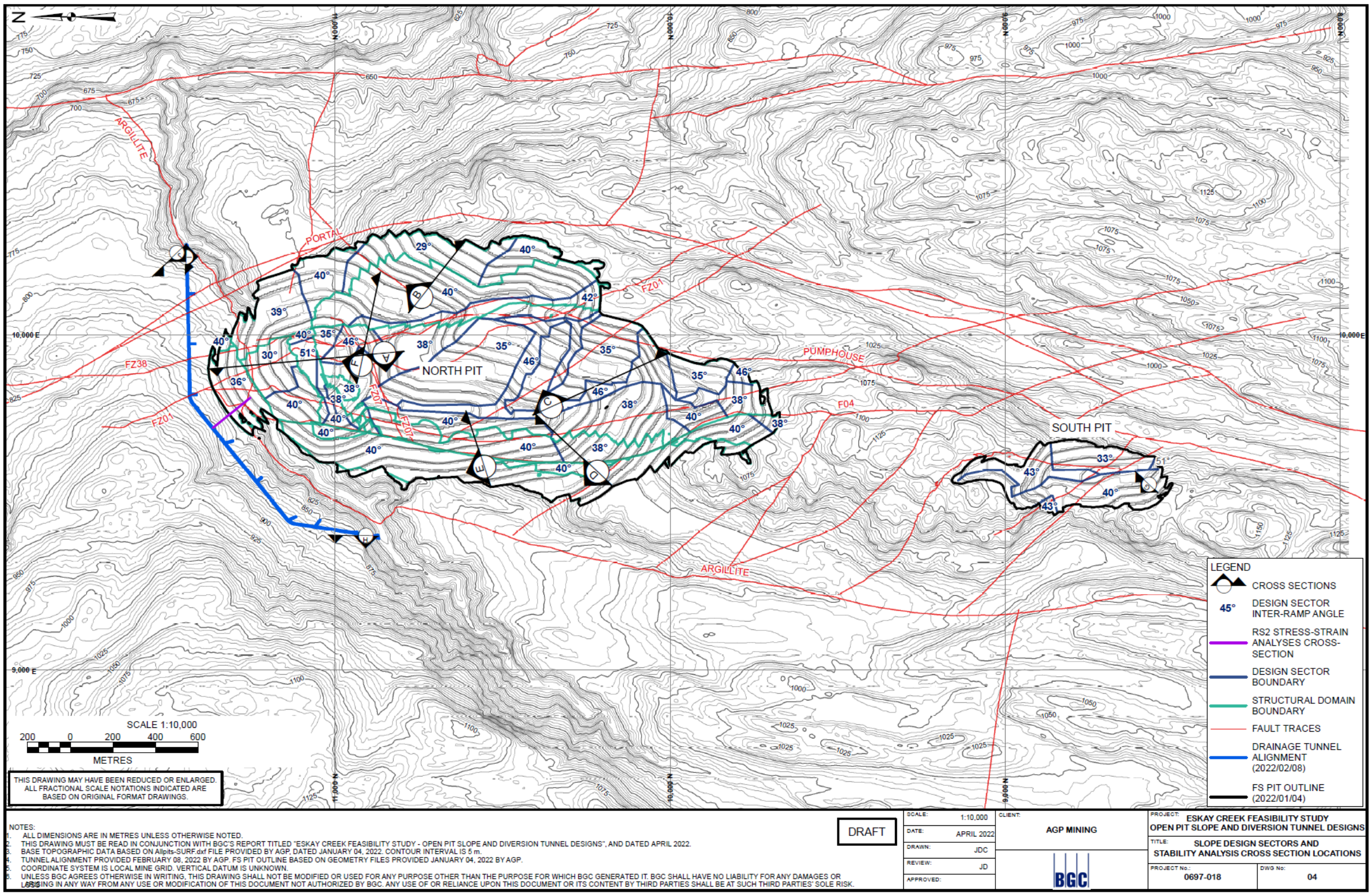
16.3.7 Slope Design Criteria

The recommended open pit slope design criteria are based on kinematic sectors and are summarized in Table 16-4. Recommended inter-ramp slope angles are illustrated on the FS pit in Figure 16-1 and range from 26° to 51°. Maximum inter-ramp stack heights should be limited to approximately 80 m in toppling-controlled sectors and 120 m in other sectors. Inter-ramp stacks should be separated by geotechnical berms or ramps that are a minimum of 30 m wide. 20 m high double benches are likely achievable in all sectors, with recommended catch bench widths ranging from 12.7 m to 37.5 m, depending on the sector. The slope design criteria assume that controlled blasting will be implemented. Scaling bench faces and cleaning accumulated material from bench toes is recommended. Slope depressurization will be required in the north, east and south walls of the North pit to meet the design acceptance criteria in these slopes.

Table 16-4: Open Pit Slope Design Parameters

Kinematic Sector	Slope Azimuth		Bench Geometry					Inter-Ramp Geometry	
			Design Height	Design Angle	Design Width	Effective Angle	Minimum Width	Maximum Height	Angle
	Start (°)	End (°)	Bh (m)	Da (°)	Dw (m)	Ba (°)	Bw (m)	Ih (m)	Ia (°)
WW-HW-258	215	305	20	80	20.3	65	14.3	80	40
WW-FW-200	185	215	20	80	22.1	68	17.7	80	38
WW-FW-258	215	305	20	80	20.3	70	16.7	80	40
CN-HW-045	030	060	20	80	18.7	65	12.7	80	42
CN-HW-278	240	315	20	80	20.3	65	14.3	80	40
CN-HW-353	315	030	20	80	31.1	65	25.1	80	30
CN-FW-093	060	125	20	80	20.3	70	16.7	80	40
CN-FW-225	210	240	20	80	22.1	68	17.5	120	38
CN-FW-268	240	295	20	80	20.3	70	16.7	80	40
CN-FW-358	295	060	20	80	12.7	70	9.1	120	51
CS-HW-045	025	065	20	80	23.0	65	17.0	80	37
CS-HW-138	090	185	20	80	37.5	59	28.8	80	26
CS-HW-215	185	245	20	80	22.1	65	16.1	80	38
CS-HW-268	245	290	20	80	20.3	65	14.3	80	40
CS-HW-338	290	025	20	80	25.0	65	19.0	80	35
CS-FW-045	025	065	20	80	23.0	70	19.4	120	37
CS-FW-095	065	125	20	80	25.0	70	21.4	120	35
CS-FW-148	125	170	20	80	15.8	62	8.5	120	46
CS-FW-205	170	240	20	80	22.1	70	18.5	120	38
CS-FW-265	240	290	20	80	20.3	70	16.7	80	40
CS-FW-338	290	025	20	80	25.0	70	21.4	120	35
EE-HW-035	000	070	20	80	21.2	65	15.2	80	39
EE-HW-088	070	105	20	80	20.3	65	14.3	80	40
EE-HW-130	105	155	20	80	32.6	65	26.6	80	29
EE-FW-108	070	145	20	80	20.3	70	16.7	80	40
EE-FW-195	145	245	20	80	18.7	70	14.8	80	42
SP-093	060	125	20	80	27.3	67	22.3	80	33
SP-165	125	205	20	80	12.7	70	9.1	80	51
SP-273	205	340	20	80	20.3	70	16.7	80	40
SP-020	340	060	20	80	17.9	70	14.3	80	43
BW-333	280	025	20	80	24.0	65	18.0	80	36
BE-353	310	035	20	80	20.3	65	14.3	80	40

Figure 16-1: Slope Design Sectors



Note: Figure prepared by BGC,2022

16.3.8 Diversion Tunnel Geotechnical Assessment and Recommendations

Ground support recommendations for the proposed diversion tunnel are based on empirical and kinematic analyses. The recommended ground support for the diversion tunnel is 2.4 m long resin-grouted rebar on 1.2 m spacing with 7 cm of mesh-reinforced or fibre-reinforced shotcrete. Approximately 10% of the ground intersected by the proposed tunnel alignment is estimated to be faulted, including smaller-scale discrete faults identified in the geotechnical logs and larger scale regional faults identified in Skeena's geology model (between approximate chainages 0+740 m to 1+060 m; see Figure 16-2). The recommended ground support through faulted ground in the diversion tunnel is 2.4 m long resin-grouted #7 rebar rock bolts on 1.1 m spacing, 12 to 15 cm of fibre-reinforced shotcrete (or mesh-reinforced shotcrete), and reinforced ribs spaced 2.9 m to 4.0 m apart, consisting of a single layer of six resin-grouted rebar (16 to 20 mm diameter) with 30 to 35 cm of shotcrete.

Two-dimensional finite element numerical modelling was used to estimate the stress-strain condition along the tunnel alignment during operation. The results of the numerical analyses indicate that the proposed ground support does not yield due to excessive deformations or loading during the tunnel life cycle, including during excavation of the open pit, but that some yielding and cracking of the shotcrete or concrete in the sill (floor) may occur. Importantly, the results of the modelling indicate no interaction between the diversion tunnel and the open pit slopes. The results also indicate that the tunnel stability is sensitive to the pore pressures, and the hydrogeological conditions along the tunnel alignment are not well understood.

The stability of the upstream and downstream portal slopes was evaluated using two-dimensional limit equilibrium slope stability analyses. The results indicate that the portal slopes stability are sensitive to the pore pressure conditions. The hydrogeological conditions in the portal areas are not well understood, and with the assumed fully saturated conditions, the natural slopes around the proposed portals do not meet the design factor of safety of 1.5. With horizontal depressurization 30 to 40 m behind the slope face, two of the three cross sections meet the design criteria. One of the analysed cross sections at the downstream portal does not meet the design criteria even with depressurization. Due to the steep topography, reduction of the overall slope angle is not possible without excavating impractical amounts of material in the natural slopes above. Depressurization, ground support, rock fall mitigation, and monitoring should be utilized to reduce the likelihood of slope instabilities impacting the tunnel infrastructure and personnel. It is recommended that Skeena establish the tunnel portals with a benched rock cut comprising bench heights of 5 m, bench widths of 3.5 m, and a bench face angles of 75°. Ground support consisting of 2.4 m long resin-grouted #7 rebar rock bolts on 1.5 m spacing with welded wire mesh and 50 mm of shotcrete is recommended for the design cut slopes at both portal locations. It is expected that raveling rock fall will accumulate on the benches, and that regular maintenance/clearing of debris will be required to maintain catchment.

Rock fall mitigation is recommended at both the upstream and downstream portals to mitigate the potential for rock fall debris from the natural slopes above the portal to impact the tunnel infrastructure and personnel. Proposed mitigation of 2 m high rock fall barriers (fences) is recommended for the upper benches at each portal. The results of preliminary rock fall analyses indicate a maximum total kinetic energy (unfactored) of 600 to 900 kJ at the proposed locations of the fences. This is a preliminary recommendation pending additional data collection and calibration of the rock fall analyses.

16.3.9 Data Gap Analysis

Further data collection and interpretation tasks are recommended to fill in data gaps to support future stages of design. These include:

- Improvements to geotechnical core logging methodology, including the addition of Joint Roughness Coefficient (JRC), degree of alteration/weathering, fracture spacing, number of discontinuity sets, identification of faults/shears, logging of both “worst-case” and “representative” discontinuities (if not feasible to log all discontinuities), logging of joint roughness number (Jr) and joint alteration number (Ja) for every discontinuity, and the use of geotechnical intervals instead of runs for the main delineation of logging units.

- The collection of supplementary structural data in areas of the open pit and diversion tunnel where existing data is sparse or where additional data it is required to validate design inputs. Surface mapping is recommended to obtain information on discontinuity persistence and waviness across the open pit area and at the diversion tunnel portal locations. Additional characterization of the location, orientation and geotechnical characteristics of major structures (i.e., fault and shear zones) is also recommended.

- Supplementary laboratory strength data, particularly in the Hanging Wall Mudstone, Contact Mudstone, Footwall Sediments and Bowser Sediments units where existing laboratory data is limited. Additionally, discontinuities were not systematically tagged by structure type during the 2020 and 2021 drilling programs, so it was not possible to develop relationships between discontinuity type and shear strength. This is recommended for future studies.

- Supplementary hydrogeological data and assessments. Additional hydrogeological testing including packer testing, piezometer installations, pumping well construction and long-term aquifer testing is recommended. A numerical groundwater flow model should be calibrated and developed under transient conditions to inform subsequent geotechnical evaluations and depressurization assumptions.
- Calibration of rock fall analyses at portal locations based on ongoing observations of rock fall activity along the Tom Mackay Creek valley.
- An in-situ stress study in the diversion tunnel area. An analysis of borehole breakouts from televiewer surveys may provide information on in-situ stresses.

16.4 Hydrogeological Considerations

16.4.1 Overview

The 2021 geotechnical drilling program provided groundwater information in the expanded northern area of the North Pit and confirmed the effect of pumping from the underground mine workings in lowering groundwater levels above these areas (Figure 16-2).

16.4.2 Hydraulic Conductivity

Packer testing of six (6) of the 11 geotechnical boreholes indicated generally lower K for Bowser Group compared to 2020 results and approximately an order of magnitude less than volcanic rocks at equivalent depths. The recent data illustrated an average reduction of almost 100% in K per 50 m increase in depth (Table 16-5).

Table 16-5: Bedrock Hydraulic Conductivities vs Depths

Lithology	Depth (mbgs)	Geomean K (m/s)
Andesite	0 - 50	9.1E-07
	50 - 100	4.94E-07
	>100	2.7E-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 16-2) and three-dimensional numerical groundwater model (Groundwater Vistas with add-on module MODFLOW-SURFACT) to predict potential inflows to the open pits.

16.4.3 Groundwater Modelling

The PFS three-dimensional numerical model was updated with data collected in 2021and incorporating the FS mine layout. The conceptual model in the pit area was verified by the data collected in 2021 with that indicated in the PFS model, i.e., a cone of depression associated with the former underground workings attributed to the managed water level around 765 masl (Figure 16-18).

The recalibrated numerical model was used to predict pit inflows in mining years one, three, six and nine, and to support permitting of the technical sample. The numerical model was run for passive and active dewatering scenarios with passive dewatering assuming gravity drainage into the pit and active dewatering using wells assigned with drain elevations set at the required water level drawdown targets.

Annual groundwater recharge in the calibrated model was estimated as ranging from 13% - 21% of mean annual precipitation (MAP) based on topographic elevation but with increased recharge (37% of MAP) in the area overlying the historic underground mine workings.

The hydraulic conductivity of the lithologies in the calibrated model are moderate to highly conductive for bedrock (2.5E-07 m/s to 5.0E-06 m/s), which is illustrative of the well jointed / fractured nature of the materials. The Andesite Creek is reportedly highly conductive (1.0E-05 m/s) whereas other mapped faults may behave as aquitards.

The baseline model calibration targets included the summer and annual low flows in Tom McKay Creek (at the creek outlet), measured pumping rates from the historical underground mine workings, and average groundwater elevations collected to date.

Criteria for water level drawdown behind the pit wall were provided by mining geotechnical consultants BGC based on 0 m, 20 m, and 40 m targets in the various pit sectors. Four steady-state models were developed using the PFS pit shells and dewatering wells spaced on a 200 m grid in the North Pit. Observation wells were set between the dewatering wells to check that the drawdown targets were met. The numbers of modelled wells and predicted inflows for the modelled years are:

- Year 1 – 3 wells; <0.5 L/s (i.e., negligible)
- Year 3 – 9 wells (5 perimeter and 4 in-pit wells); 7 L/s
- Year 6 – 16 wells (8 perimeter and 8 in-pit wells); 31 L/s
- Year 9 – 14 wells (perimeter and 6 in-pit wells); 62 L/s

The predicted inflow to dewatering wells and residual pit inflow (not captured by the dewatering wells) as well as seepage to the Tom McKay diversion tunnel and the historic underground workings over the life of mine are shown graphically in Figure 16-3.

A sensitivity analysis was carried using wet and dry scenarios (30% more and 30% less recharge) and higher and lower K (5x) than the calibrated baseline model. The higher recharge scenario indicated that in Year 9, an additional 4 L/s would need to be collected in 3 wells to meet the drawdown criteria.

Figure 16-2: Cross Section of Deposit Geology and Groundwater Levels

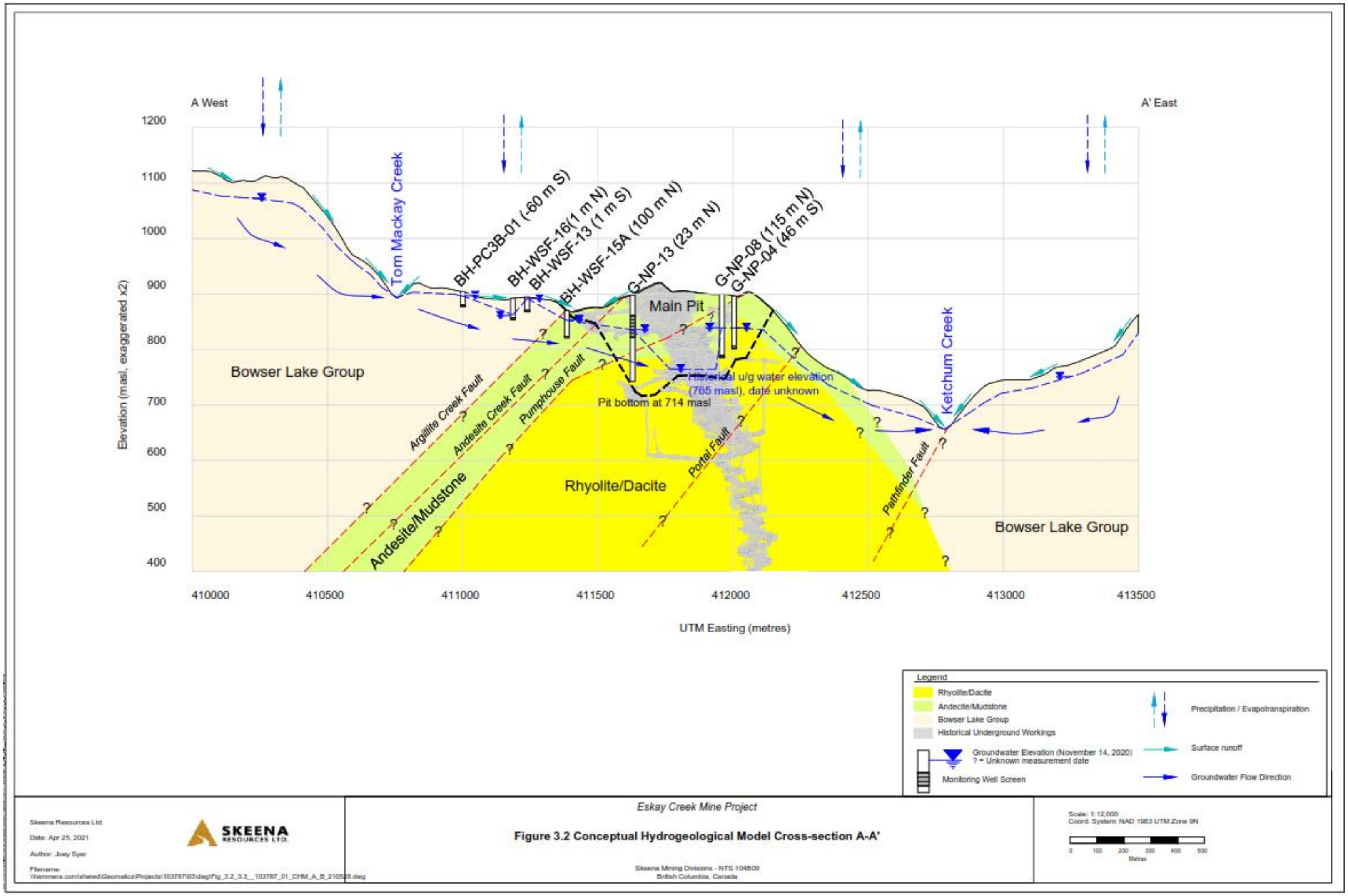


Figure 16-3: Dewatering flow rates and mine tunnel inflow estimates

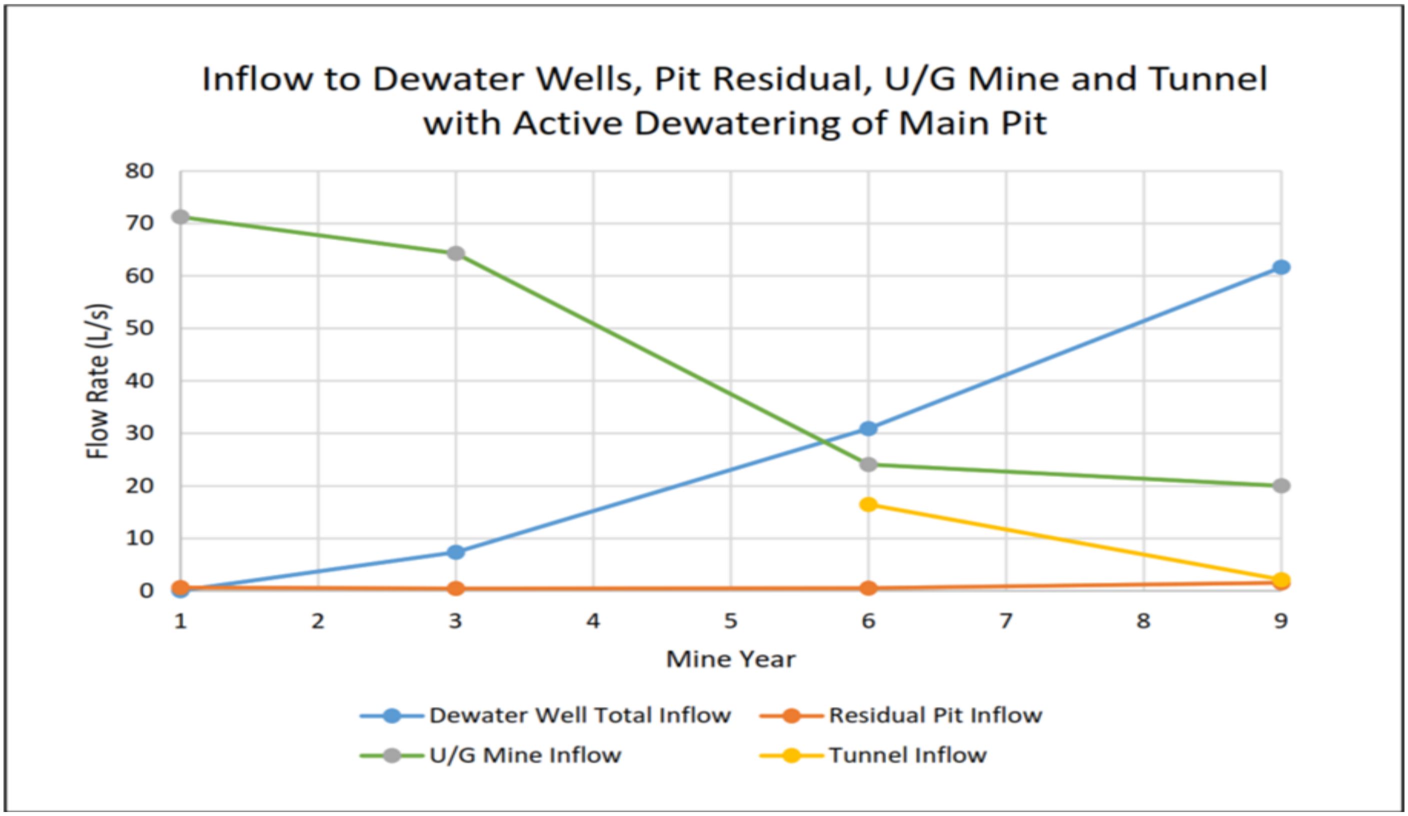
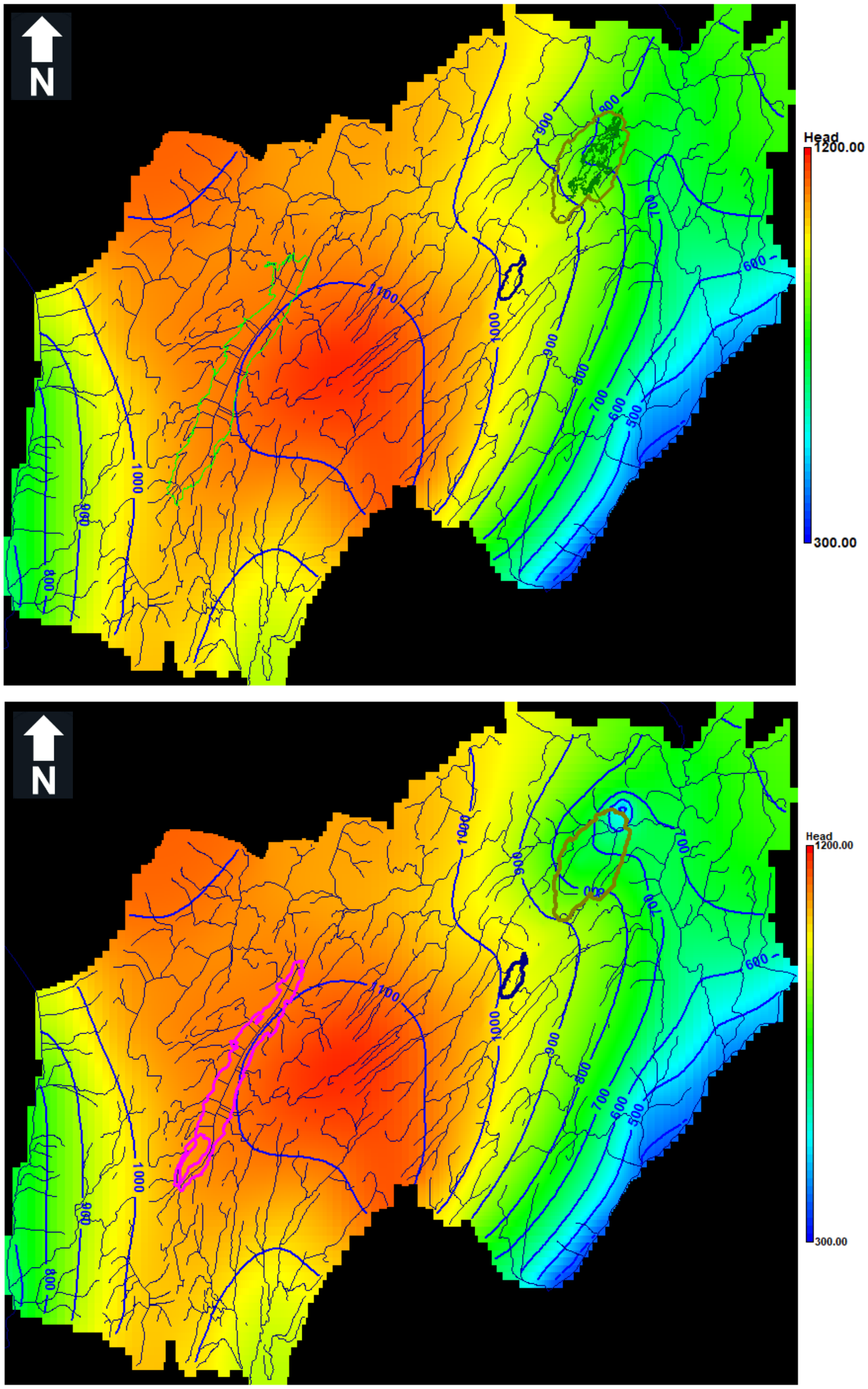


Figure prepared by Ausenco, 2022.

Groundwater contour maps for baseline (current) and end of mining are shown in Figure 16-4. The effect of pit dewatering can be seen in the northern (deepest) pit area, where the drawdown is in the order of 100 – 200 m from current conditions. The baseline model estimates that approximately 90% of groundwater flow from the historic underground workings goes to Ketchum Creek with the remainder to Tom McKay Creek. A relatively small volume of groundwater is pumped out of the historic underground workings during spring freshet to prevent discharge to creeks. This water is treated and released to Ketchum Creek and this arrangement will continue until the pit is significantly progressed when water stored in the underground workings is expected to seep to the North Pit.

Figure 16-4: Groundwater Contour Maps (current in upper image; end-of-mine life in lower image)



Note: Figure prepared by Ausenco, 2022.

16.4.4 Potential Groundwater Risks and Opportunities Based on Current Mine Plan

16.4.4.1 Mining Pits

Groundwater modeling indicates that pit dewatering can be achieved with less than 10 wells in the early years of pit development, and up to 16 (perimeter and in-pit wells) in later years. Vertical dewatering wells may be replaced by in-pit, horizontal boreholes, particularly in lower permeability materials.

Dewatering the historic underground workings ahead of pit advancement may prove to be an effective way to dewater the country rock surrounding the pit and potentially reduce the risk of unplanned releases of stored water during mining. The existing underground water management system may be extended to maintain the water level below the active pit bottom.

Pumping tests in wells intersecting the larger mapped faults (e.g., Andesite Creek Fault) and valley bottoms along the perimeter of the pit will provide valuable information on bulk hydraulic parameters of these potential aquifers and the dewatering response that can be realized in the surrounding rock mass from perimeter dewatering wells.

The South Pit will be developed at higher elevation than the North Pit, and advanced predominantly above the water table. A dewatering program is therefore not anticipated, and seasonally perched groundwater is best dealt with by in-pit horizontal boreholes.

16.4.4.2 Tailings Storage Facility

The tailings storage facility (TSF) is underlain by tailings, lake sediments and Bowser Group (predominantly fine-grained sedimentary rocks). Hydraulic containment that is typical in basin storage (I.e., the groundwater level in surrounding areas is higher than the lake surface) is operative in central parts of the lake basin but less evident in the south and north of the TSF. Flow vectors show seepage to the west and south of the tailings depositional area; however, the tailings and underlying fine-grained sediments, as well as low permeability dam materials are expected to limit seepage from the TSF. Leakage via the dam abutments is possible but unlikely to be significant. The dams will be founded on bedrock and are designed with geosynthetic membranes to limit seepage through the dams whereas as the south dam will be built using clay core.

Sub-aqueous storage of PAG materials in the TSF is expected to limit ML/ARD generation; however elevated solutes (primarily sulphate) can be expected to change downgradient groundwater quality. Seepage from the south end of the TSF is predicted to mostly flow westward to Harrymel Creek, and minimally to Tom McKay in the north. The estimated seepage to Harrymel is less than 10% of the Annual 7-day Low Flow and therefore unlikely to have a significant effect on surface water quality. Discharge to Tom McKay will be maintained through a penstock so groundwater seepage to the creek is of less consequence to surface water quality.

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 16-6. 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 - 70 g/t gold bulk concentrates.

Table 16-6: Pit Shell Parameter Assumptions

Description	Units	Value	Gold Value	Silver Value
Exchange rates				
C$	US$ =	1.26		
Resource Model				
Block classification used		M+I		
Block Model height	m	10		
Mining Bench height	m	10		
Metal Prices				
Price	$/oz		1550	20
Royalty	%		2%	2%
Smelting, Refining, Transportation Terms				
Concentrate grades	g/t Au		20 - 70	
Payable	%		69 - 92.5	75 - 90
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	148.00		
Metallurgical Information - Rhyolite and Contact Mudstone				
Recovery	%	88.88 * (1-exp([Au Feed g/t] * (-1.5669)))		
Mass pull				
Au feed < 0.5g/t		0.5		
0.5 < Au feed < 3.5g/t		1.1867*(Au Feed [g/t]) + 1.6069		
3.5 < Au feed < 8g/t		4.275*(Au Feed [g/t]) – 9.204		
Au feed > 8g/t		25		
Au con	g/t	Recovery*(Au Feed [g/t])/(Mass Pull)		
Metallurgical Information - HW, Mudstone, default				
Recovery	%	62.74 * (1-exp([Au Feed g/t] * (-1.0706)))		
Au con	g/t	4.9359* (Au Feed [g/t]) + 1.7253		
Mass pull		Recovery*(Au Feed [g/t])/(Au Con)		
Cost of power	C$/Kwhr	$0.05		
Diesel fuel cost to site	C$/ l	$1.31		
Mining Cost *			NAG	PAG
Waste base rate - 880 elevation	C$/t		3.02	3.71
Incremental rate - above	C$/t/10m bench		-0.018	-0.018
Incremental rate - below	C$/t/10m bench		0.041	0.041
Mill feed base rate - 880 elevation	C$/t		2.43	2.43
Incremental rate - above	C$/t/10m bench		0.020	0.020
Incremental rate - below	C$/t/10m bench		0.034	0.034
Processing **				
Processing cost	C$/t mill feed	$14.18		
Maintenance (incl road and bridges)	C$/t mill feed	$4.04		

Description	Units	Value	Gold Value	Silver Value
Total processing cost	C$/t mill feed	$18.22		
General and Administrative Cost				
G&A cost	C$/t mill feed	$6.23		
Total Process and G&A				
Process + G&A	C$/t mill feed	$24.45		

Note: * mining costs based on using 144 t haul trucks. ** process costs based on 3 Mt/a dry throughput

Wall slopes for pit optimization were based on review of available historical underground data and analysis as outlined in Section 16.3 from the 2021 field program. A design sector map was created which was defined by structural domains and dominant geotechnical units. Solids were used to code the model SLP item, then overall slopes were applied by azimuth as shown in Table 16-7.

Table 16-7: Pit Shell Slopes

Structural Domain	Dominant Geotechnical Unit	SLP Code	Azimuths Start (°)	Azimuths End (°)	Angle (°)
WW (west wall)	HW/Contact MS	1	185	215	38
			225	175	40
	Rhyolite/FW	2	185	215	38
			225	175	40
CN (center north)	HW/Contact MS	3	040	050	42
			060	105	40
			115	150	29
			160	240	38
			250	305	40
			315	030	30
	Rhyolite/FW	4	060	125	40
			135	200	51
			210	240	38
			250	295	40
			305	050	51
CS (center south)	HW/Contact MS	5	035	055	37
			065	080	28
			090	185	26
			195	245	38
			255	280	40
			290	025	35
	Rhyolite/FW	6	035	055	37
			065	125	35
			135	160	46
			170	240	38
			250	280	40
			290	025	35
EE (east wall)	HW/Contact MS	7	000	070	39
			080	095	40
			105	145	29
			155	220	26
			230	250	30

Structural Domain	Dominant Geotechnical Unit	SLP Code	Azimuths		Angle
			Start (°)	End (°)	(°)
			260	350	40
	Rhyolite/FW	8	000	070	39
			080	145	40
			155	235	42
			245	350	40
SP (south pit)	Rhyolite/FW	9	060	125	33
			135	195	51
			205	340	40
			350	050	43
BW (bowser west)	Bowser	10	280		36
BE (bowser east)	Bowser	11	310		40
	Default	12	0		40

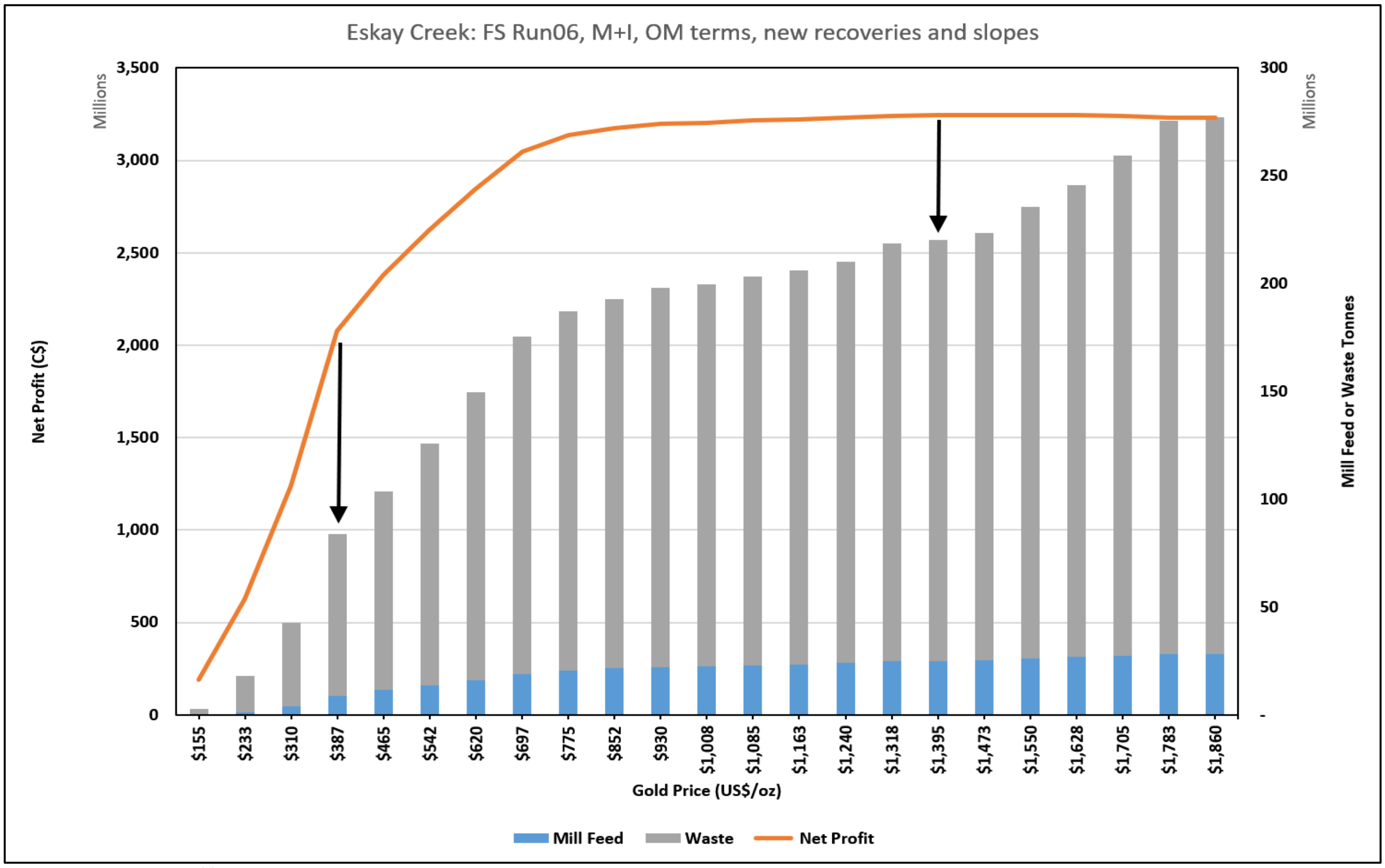
Nested L–G pit shells were generated to examine sensitivity to the gold and silver prices with a target of US$1,550/oz Au and US$20.00/oz Ag. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Undiluted Measured and Indicated resources were 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$1,550/oz up to US$1,860/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$1,550/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 16-5.

Figure 16-5 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$387/oz Au, the cumulative waste tonnage is 75 Mt, with a corresponding mill feed tonnage of 8.7 Mt or a strip ratio of 8.7:1. 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 64% of the net value of a $1,550/oz pit but with only 36% of the waste of the larger pit shell. This break point contains two distinct areas in the north pit. The southern portion of this pit shell contains a high-grade zone with no historic underground mining. The northern portion of this shell extends down the Tom MacKay Creek and leaves a reasonable final pushback to the ultimate pit limits.

The second and final pit shell selected represented the ultimate pit at US$1,395/oz Au. This resulted in a substantial jump in the waste tonnage from the first break point by 120 Mt with a gain of 16.4 Mt of feed material for an incremental strip ratio of 7.3:1. The net profit continues to increase beyond this break point, although at a flatter rate than the first breakpoint. The cumulative value of the two break points was 100% of the US$1,550/oz Au pit shell with 93% 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$ 620/oz Au or US$697/oz Au. However, access would become more difficult as shells ran the length of the deposit and backfill areas would likely be even more restricted 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.

Figure 16-5: Eskay Creek Potential Profit vs. Price by Pit Shell



Note: Figure prepared by AGP, 2022.

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 high-grade is restricted to within a 1 metre buffer to reduce grade smearing into neighbouring blocks. 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. A low-grade envelope based on 0.7 g/t AuEq was included in the resource estimate so that neighbour block grades could be used in dilution calculations.

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 10 x 10 m in plan view, and 10 m high. Mining would be completed on 10 m lifts for waste and 5 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 12.5% (1.25 m/10 m) by volume would result. If two sides are contacting, it would rise to 25%. Three sides would be 37.5%, and four sides 50%. Four waste contact sides represent an isolated block of mill feed. These isolated ore blocks were not very common, so they were left as highly diluted ore blocks rather than flagging them as waste.

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$24.45/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.

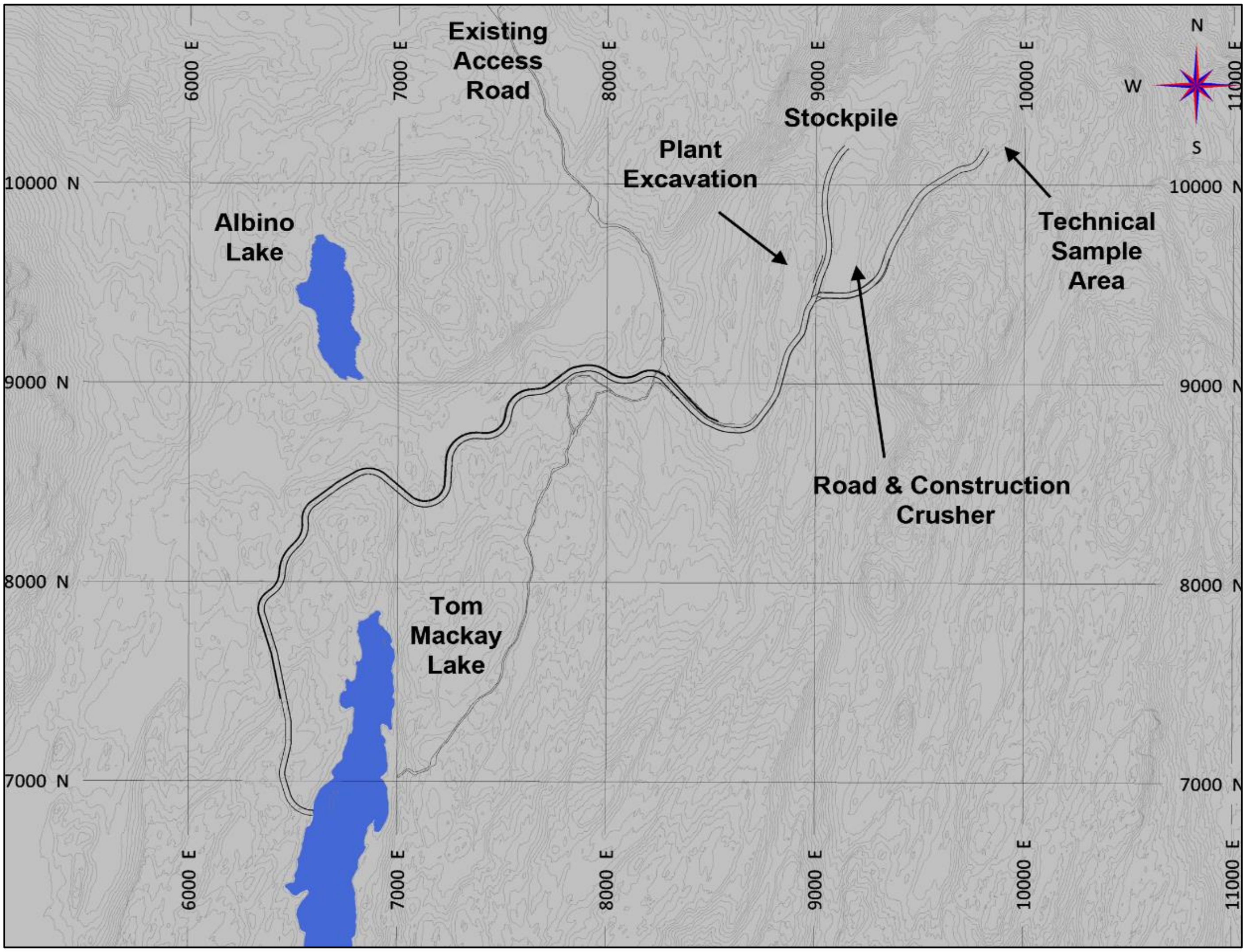
In this manner, the contact diluted blocks were included in the tonnage and grade calculation of mill feed tonnes. The mill feed tonnage report was then run with the block model DORE% item to report out the diluted tonnes and grade.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 19.7% more tonnes and 15.6% 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 0.19 g/t Au and 3.71 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 Initial Road Construction

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 and two quarries in the north pit area. 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 NAG waste sources. A road will also be required between the technical sample area and the stockpile location near the future crusher. The approximate initial road locations are shown in Figure 16-6. The initial roads will be established in year -3 of the mine schedule so infrastructure is established for obtaining a mineralized technical sample of 10 kt. 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 3.0 Mtpa by the end of Year 1.

Figure 16-6: Initial Roads for Pre-Production



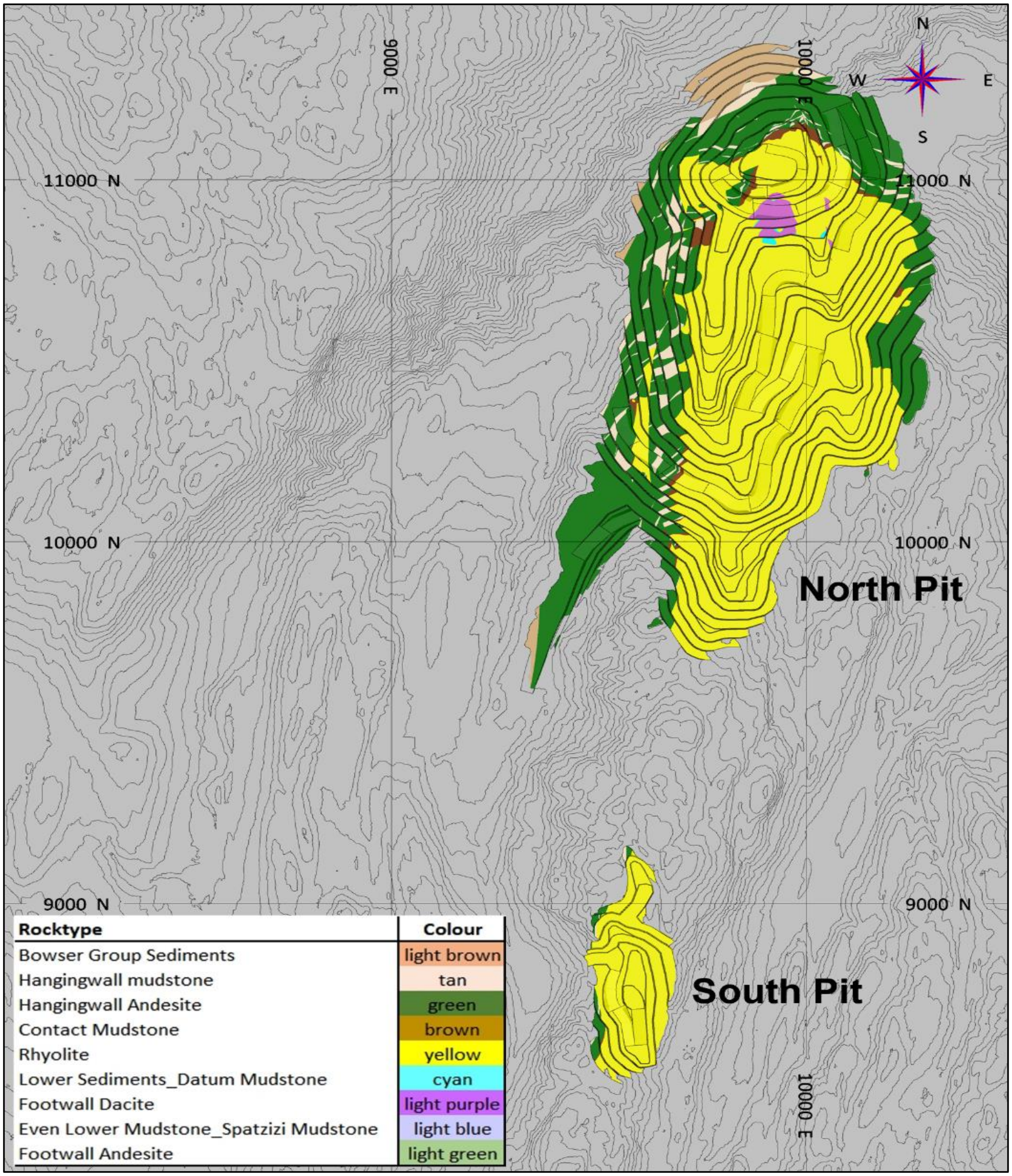
Note: Figure prepared by AGP,2022

16.8 Pit Designs

Pit designs were developed for the north and south pit areas. The initial phases were designed for the purpose of obtaining a technical sample and necessary NAG waste material to create supporting infrastructure. The north pit will consist of an additional three 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.

The north and south pits are displayed in Figure 16-7. The south pit is significantly smaller than the north pit and is likely to be mined near the end of the mine schedule. The south pit generally has harder rock and lower gold grades. Rhyolite is the dominant rock type that will remain in the mined-out pit walls before reclamation.

Figure 16-7: Rock types in Ultimate North and South Pits



Note: Figure prepared by AGP, 2022.

Geotechnical parameters outlined in Table 16-4 were applied to pit designs. A summarized version of these parameters is also displayed in Table 16-8 Pit slope sectors from Table 16-1 were developed as solids and used to code slope domains in the mine planning model. All areas of the pit were designed with 80° bench face angles and 10 metre bench heights. 20 metres between catch berms was determined to be acceptable in all areas of the pits. The catch berm widths were varied to target design inter-ramp angles in each slope sector. In addition to these criteria, adjustments were made to consider maximum inter-ramp stack heights of 80 metres in HW/Contact MS, Bowser, and toppling controlled sectors, and 120 m in all other sectors. A minimum 30 m wide geotechnical berm or ramp was to be included between inter-ramp stacks.

Table 16-8: Pit Slope Design Parameter Summary

Structural Domain	Dominant Geotechnical Unit	Domain Code SLP	Bench Face Angle	Height Between Berms	Range of values within Rosette	
					Inter-Ramp Angle	Catch Bench Width
			(degrees)	(m)	(degrees)	(m)
WW (West)	HW/Contact MS	1	80	20	38-40	20.3-22.1
	Rhyolite/FW	2	80	20	38-40	20.3-22.1
CN (Centre N)	HW/Contact MS	3	80	20	29-42	18.7-32.6
	Rhyolite/FW	4	80	20	38-51	12.7-22.1
CS (Centre S)	HW/Contact MS	5	80	20	26-40	20.3-37.5
	Rhyolite/FW	6	80	20	15.7-25	35-46
EE (East)	HW/Contact MS	7	80	20	26-40	20.3-37.5
	Rhyolite/FW	8	80	20	39-42	18.7-21.2
SP (South Pit)	Rhyolite/FW	9	80	20	33-51	12.7-27.3
BW (Bowser W)	Bowser	10	80	20	36	24
BE (Bowser E)	Bowser	11	80	20	40	20.3

Note: 10m bench heights

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.

Tonnes and grade for the final pit designs are reported in Table 16-9 using the diluted tonnes and grade from the model and a mining recovery of 98% to account for additional mineralized material losses. Only Measured and Indicated Mineral Resources were included in the mill feed summary. An NSR cut-off grade of C$24.45/t was used to determine mill feed material blocks. Opportunities should be further considered for different storage, blending and processing strategies of lower-grade mineralized ore.

Table 16-9: Final Design – Phases, Tonnages, and Grades

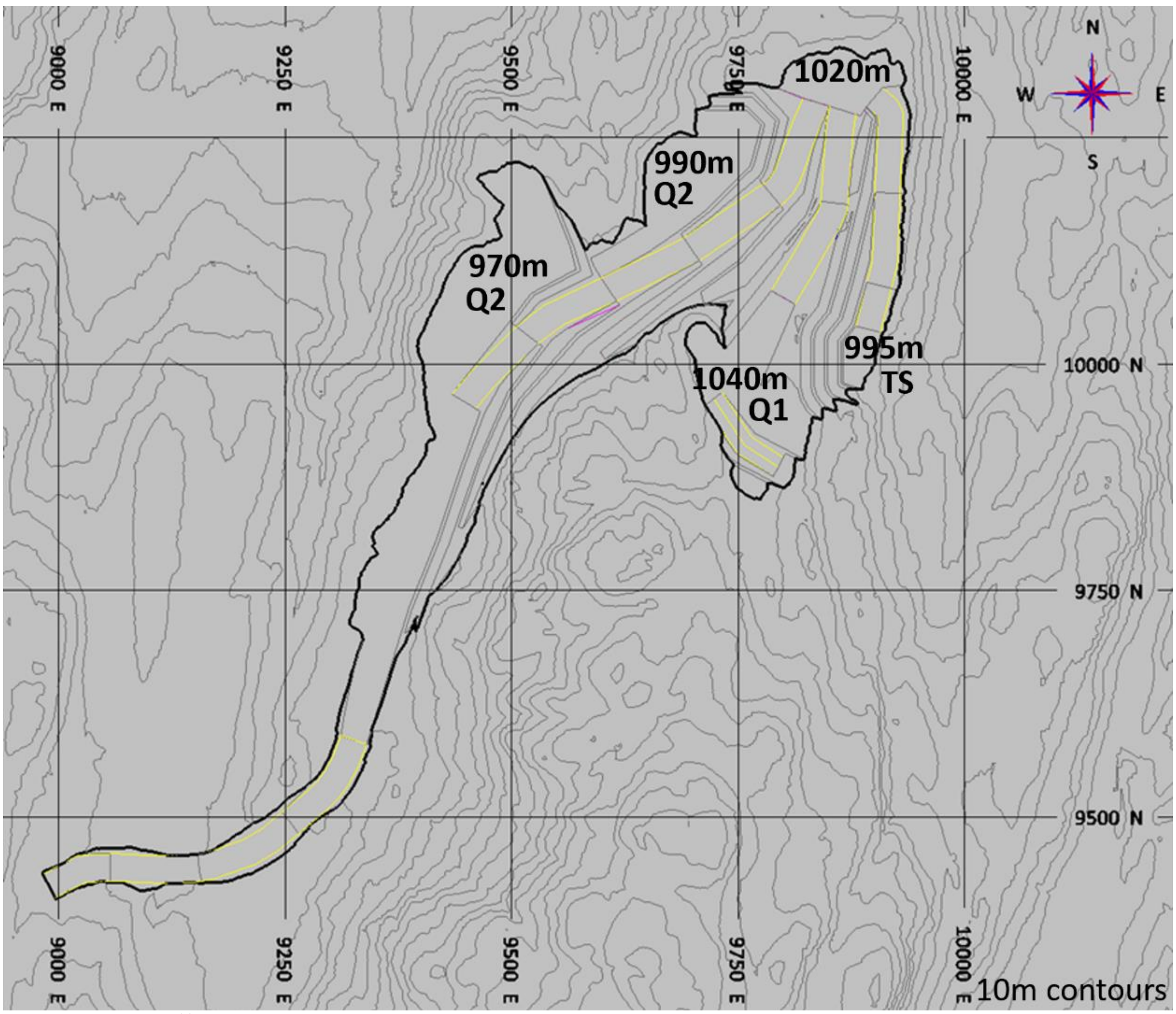
Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	As (ppm)	Hg (ppm)	Sb (ppm)	S (%)	Fe (%)	Waste (Mt)	Total (Mt)	Strip Ratio
Quarry1	0.00	0.00	0	0.00	0.00	0.00	0	0	0	0.0	0.0	0.37	0.4	0.0
Quarry2	0.00	0.00	0	0.00	0.00	0.00	0	0	0	0.0	0.0	3.69	3.7	0.0
TS	0.02	6.48	57	0.03	0.08	0.02	5,796	147	603	1.8	1.9	1.04	1.1	48.0
Phase 1	4.65	3.51	61	0.09	0.16	0.02	1,924	135	1,619	1.6	1.5	22.51	27.2	4.8
Phase 2	9.64	3.70	118	0.70	1.11	0.10	659	74	1,900	2.1	2.9	96.75	106.4	10.0
Phase 3	13.42	2.49	59	0.46	0.74	0.07	382	15	569	1.7	2.3	93.80	107.2	7.0
South Phase 1	2.18	1.74	59	0.07	0.10	0.01	744	5	238	0.6	1.1	5.34	7.5	2.4
Total	29.91	2.99	78.5	0.45	0.72	0.07	741	52	1,137	1.7	2.3	223.5	253.4	7.5

The phase designs are described in further detail in the following sub-sections.

16.8.1 North Phases TS, Q1 and Q2

A technical sample phase (TS) will be mined in Years -3 and -2 so that process performance of the mill can be evaluated with a larger sample than drill hole samples. Quarry 1 (Q1) and Quarry 2 (Q2) will be mined along with TS so that adequate NAG waste is available for construction of required roads and site infrastructure. These phases are located near the top of the north pit ridge as displayed in Figure 16-8.

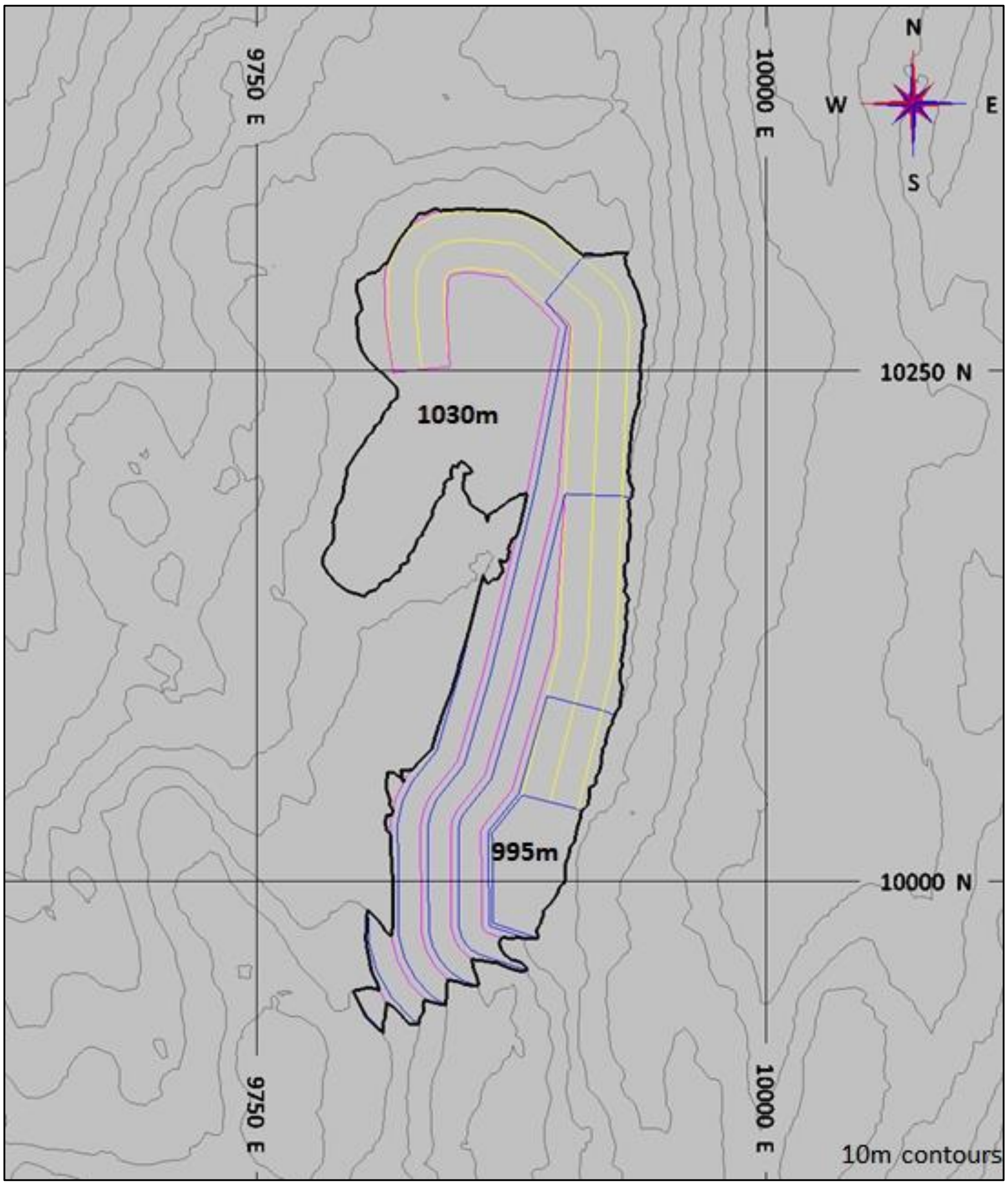
Figure 16-8: Combination of TS, Q1 and Q2 Phases



Note: Figure prepared by AGP, 2022.

Very few shallow targets were available due to the plunging nature of the orebody away from topography. Phase TS bench elevations range from 1050 m down to 995 m. All benches are 10 metre high, with the exception of a 5 metre bottom bench. The mill feed sample is available on the bottom two benches of this phase. The north phase TS design is shown in Figure 16-9.

Figure 16-9: Proposed North Phase TS

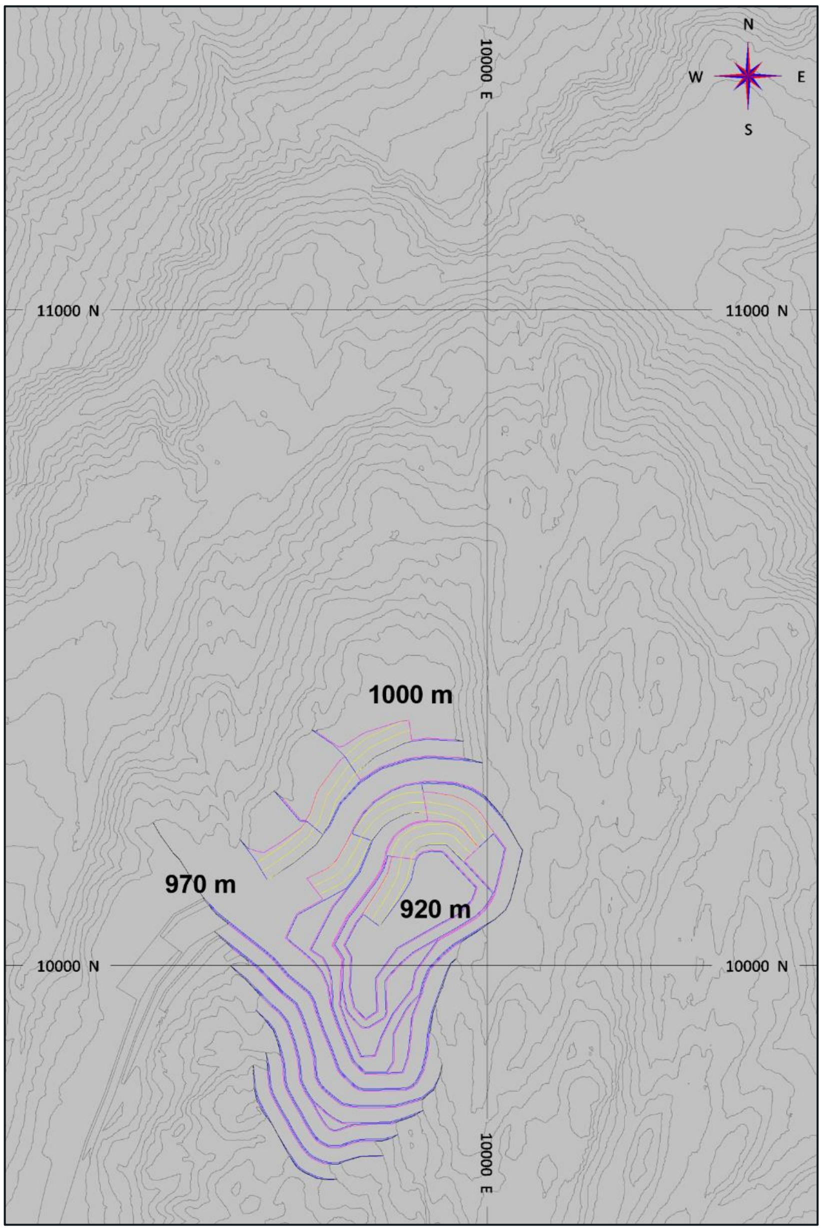


Note: Figure prepared by AGP, 2022.

16.8.2 North Phase 1

Phase 1 will start being mined in Year -2. 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 1090 masl down to 920 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 16-10.

Figure 16-10: Proposed North Phase 1

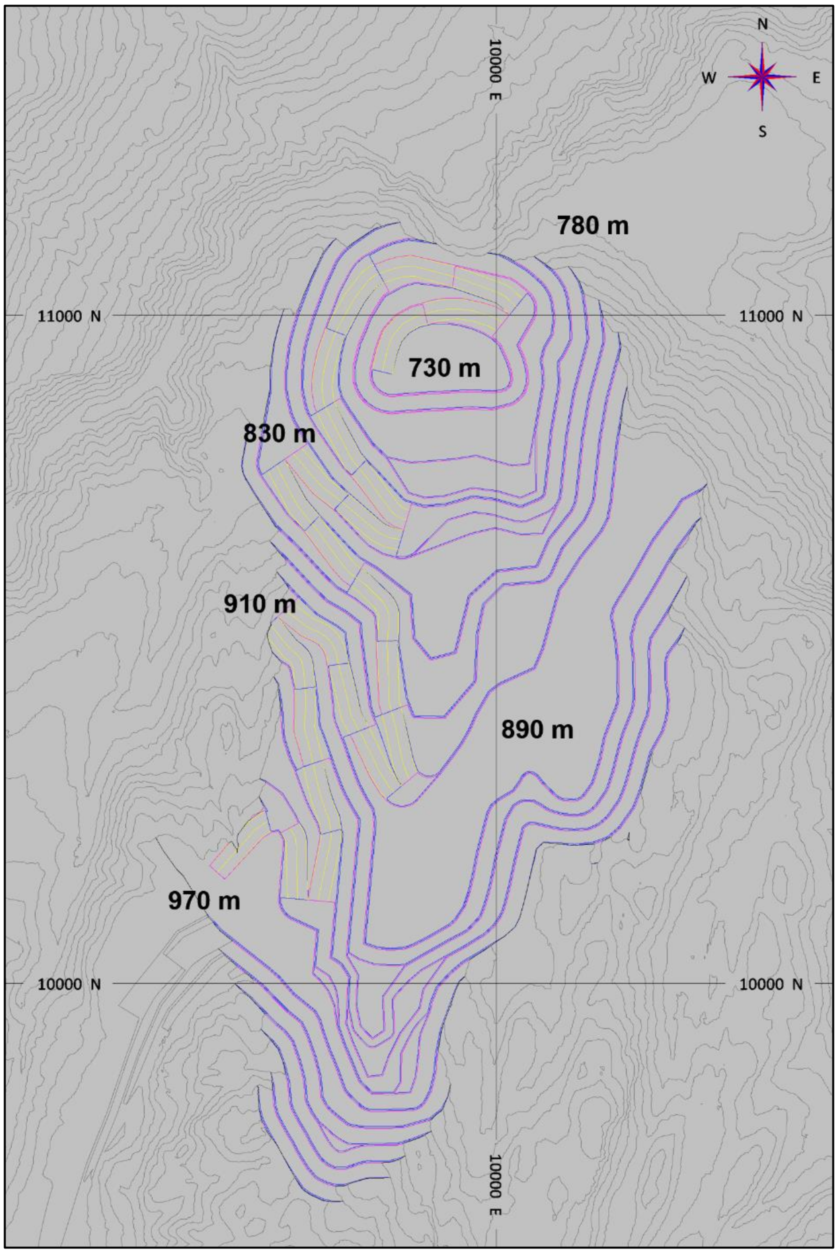


Note: Figure prepared by AGP, 2022.

16.8.3 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 phase 1, 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 990 masl down to 760 masl. The north phase 2 design is shown in Figure 16-11.

Figure 16-11: Proposed North Phase 2

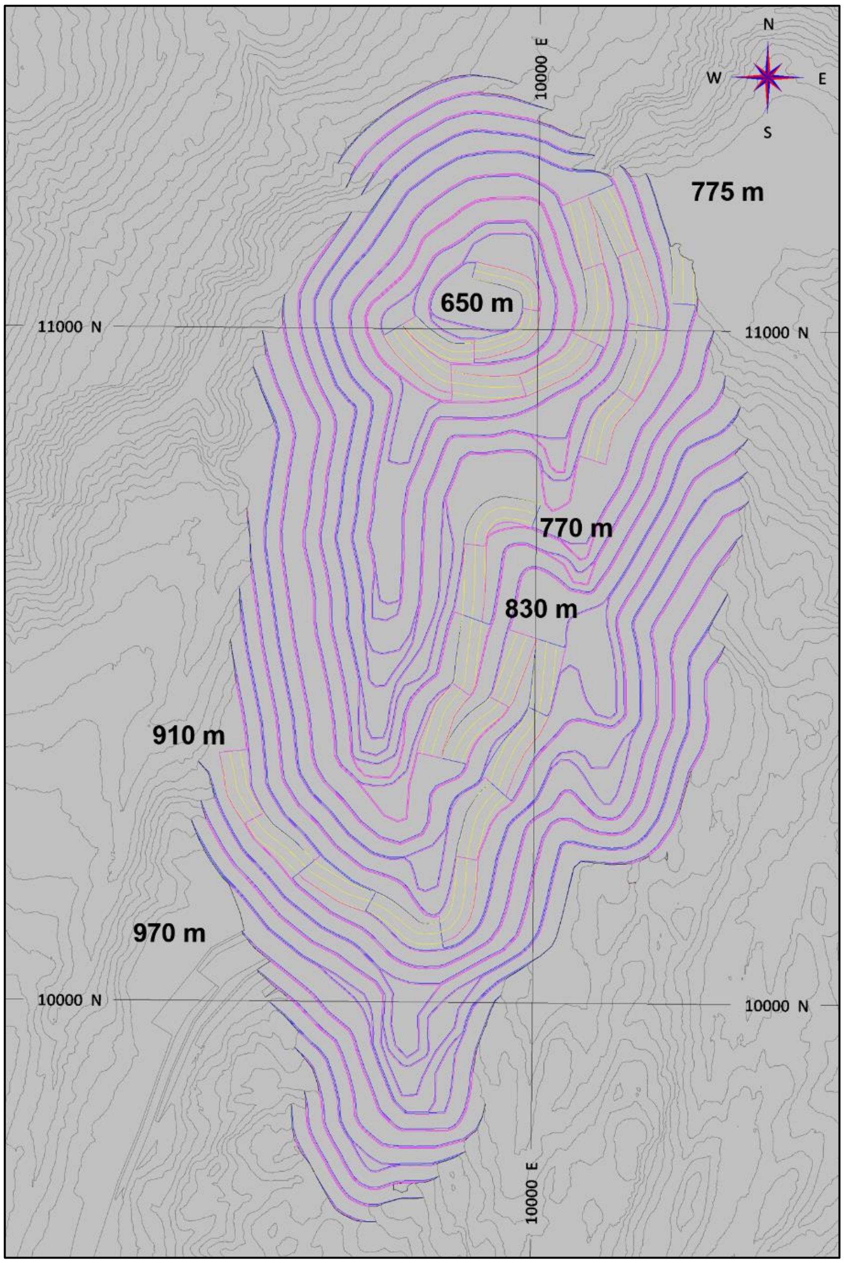


Note: Figure prepared by AGP, 2022.

16.8.4 North Phase 3

Phase 3 is the final north phase and extends across Tom Mackay Creek. The access to the north side of the creek is displayed later in the schedule figures. The pit exit at 910 m elevation is where the mined material will leave the pit near the crusher. Phase bench elevations will range from 960 masl down to 650 masl. The phase 3 design is shown in Figure 16-12.

Figure 16-12: Proposed North Phase 3

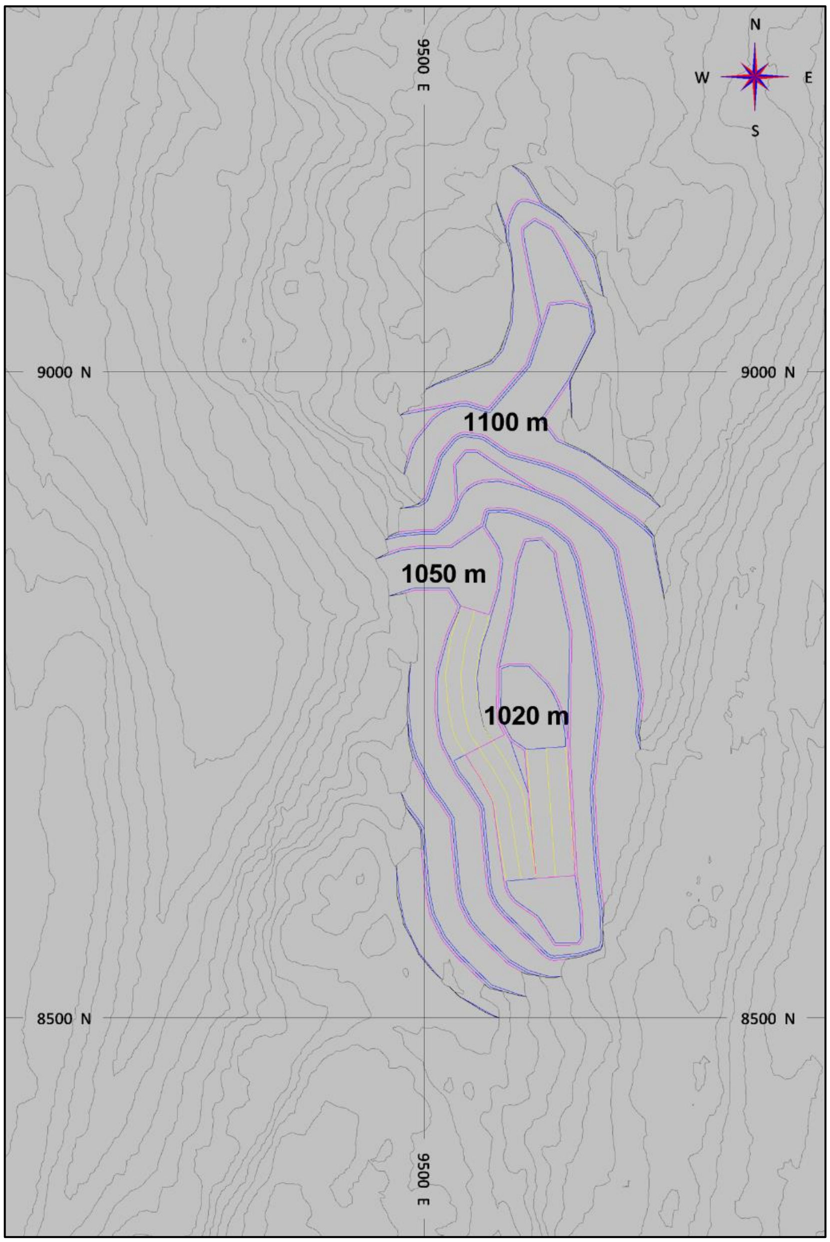


Note: Figure prepared by AGP, 2021.

16.8.5 South Phase 1

There will only be a single small phase in the south pit. Phase bench elevations will range from 1140 masl down to 1020 masl. This phase will be mined at the highest elevation of any of the phases, 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 16-13.

Figure 16-13: Proposed South Phase 1



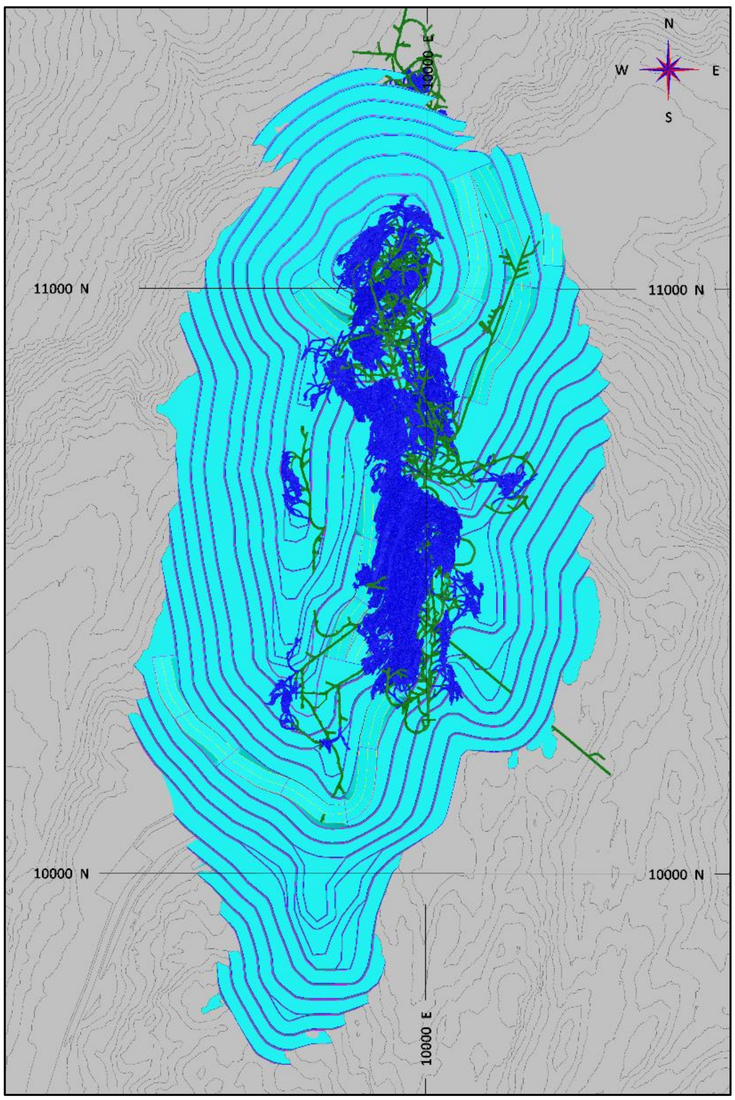
Note: Figure prepared by AGP, 2022.

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 only near the north pit and shown in dark blue in Figure 16-14.

Figure 16-14: Location of Historic Underground Workings



Note: Figure prepared by AGP, 2021.

16.10 Water Diversion

Mine planning indicates that the northern end of the open pit will intersect Tom MacKay creek requiring the provision of a water diversion tunnel to re-route flowing water around the open pit before re-entering the existing creek downstream. AGP has worked with Swiftwater Consulting Ltd. (Swiftwater) to develop the initial technical design for the water diversion.

The tunnel design flow will be linked to certain requirements of the British Columbia Dam Safety Regulation (B.C. Reg. 11/2021; the Regulation). The Regulation requires that the design of a dam and ancillary features be commensurate with the consequences that may occur should the dam breach. These consequences are determined through the completion of a Dam Failure Consequence Classification (DFCC), which is then used to select a flow scenario under which the dam and supporting infrastructure is to be designed, in accordance with the Canadian Dam Association (CDA) Dam Safety Guidelines (CDA, 2013).

Swiftwater has conducted only a preliminary assessment of potential failure modes and downstream consequences. Such an assessment is intended to conservatively estimate the potential effects of a dam failure, for the purpose of informing the preliminary sizing of various diversion infrastructure.

The preliminary DFCC for the Tom MacKay Creek diversion dam is SIGNIFICANT, and Swiftwater recommends that an Inflow Design Flood (IDF) of 72.6 m^3/s be adopted for the diversion dam and tunnel. Both the DFCC and IDF will need to be updated as the diversion concept develops and additional hydrological data is obtained. However, these values are considered appropriate for use in the PFS concept development.

The proposed diversion is anticipated to require the following infrastructure:

- A diversion tunnel (or tunnels) that allows water to safely bypass the future pit
- An upstream diversion dam to direct flow from Tom Mackay Creek into the tunnel
- A grizzly rack to prevent larger trash and debris from entering the tunnel
- A downstream cofferdam, if required, for the purpose of preventing water exiting the tunnel from back-watering into the pit
- An energy dissipation feature at the outlet of the tunnel to mitigate against excessive erosion and scour due to concentrated tunnel outflows
- Access roads, bridges, and related infrastructure
- Instrumentation as required to monitor water levels and flows relevant to the diversion works.

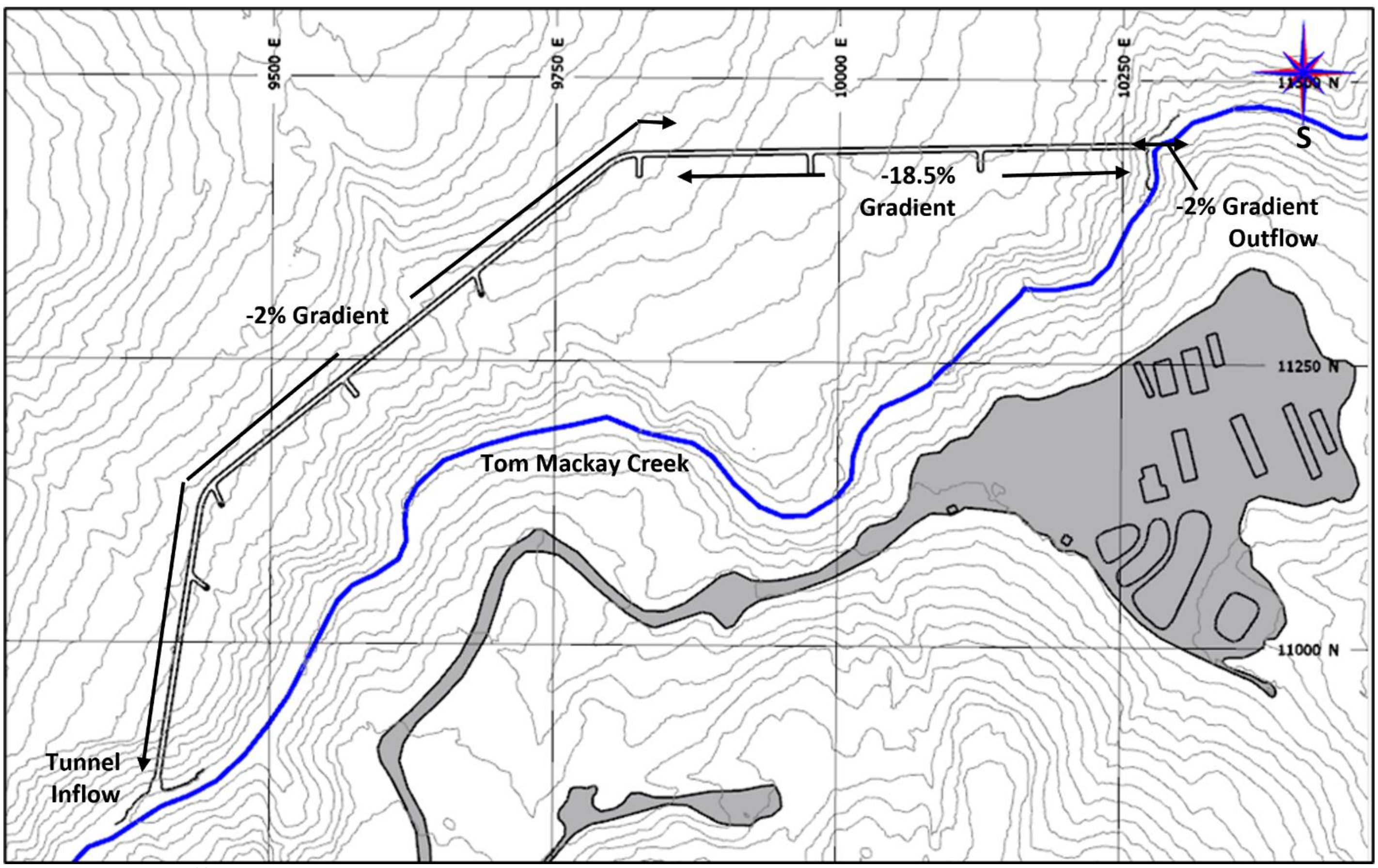
There are several additional features that may or may not be required including the following:

- An isolation gate, capable of isolating the tunnel for the purpose of inspection or maintenance
- Sediment management features, including rock traps
- Adits to collect water from tributaries that overlie the tunnel alignment.

The recommended water diversion layout is displayed in Figure 16-15. Minimum tunnel dimensions have been selected as 4.7 metres wide by 4.7 metres high in order to accommodate the expected water flows. The full length of the tunnel is 1214 metres. Starting from the tunnel inlet, 802 metres are at -2% gradient, 362 metres at -18.5% gradient, and 50 metres at -2% nearest the outlet.

The tunnel will be constructed using drill-and-blast methodology with a mean overbreak thickness of 200 mm occurring outside of the nominal tunnel dimensions. It has been determined that the tunnel must include a liner covering 100% of the tunnel wall to mitigate against water contact with Potentially Acid Generating (PAG) rock, as well as provide additional structural support.

Figure 16-15: Water Diversion Layout



Note: Figure prepared by AGP, 2021.

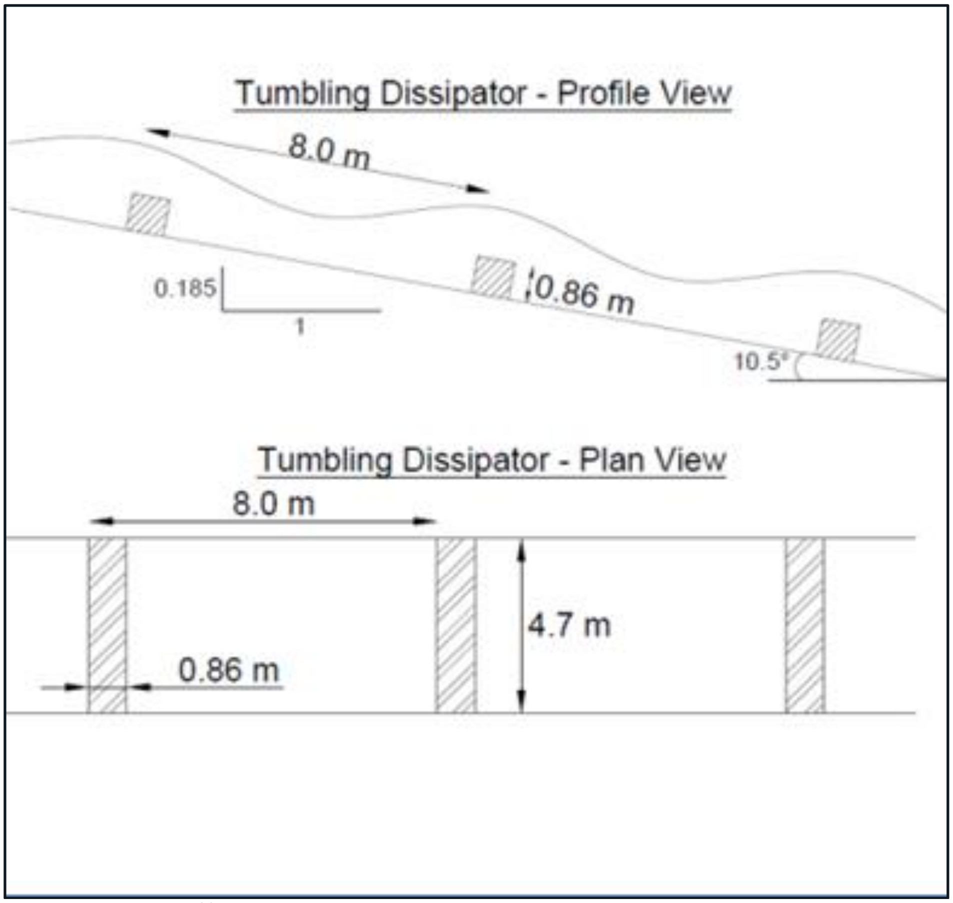
The transition from the shallow gradient tunnel into the steeper grade section should occur gradually, the invert should be constructed on a radius equivalent to 10 times the tunnel diameter, or 47 m, to allow for a smooth flow transition on to the steeper ramp.

The steeper grade section is to be constructed at a slope of 18.5%. Due to the high velocities on the slope, fibrecrete will be replaced in the tunnel invert and side walls by cast-in-place reinforced 40Mpa strength concrete with a dowelled rebar mat. A final decision regarding whether or not to provide energy dissipation on the slope will be based on a cost-benefit analysis of construction pricing versus anticipated maintenance effort. At this stage it is uncertain whether it will be necessary to provide energy dissipation on the -18.5% gradient slope. The proposed associated construction is shown in Figure 16-16.

At the base of the steeper grade section, a 5.0 m deep plunge pool should be provided to reduce energy as the tunnel transitions back into a short section of shallow grade tunnel that discharges to the environment.

As there is potential for additional surface water flows on the slopes above the diversion tunnel, three raises have been included in the design to allow for additional surface water collection. By diverting this water into the tunnel, this water will also be directed away from the mining areas.

Figure 16-16: Potential Tumbling Energy Dissipator on 18.5% gradient Section



Note: Figure prepared by AGP, 2021

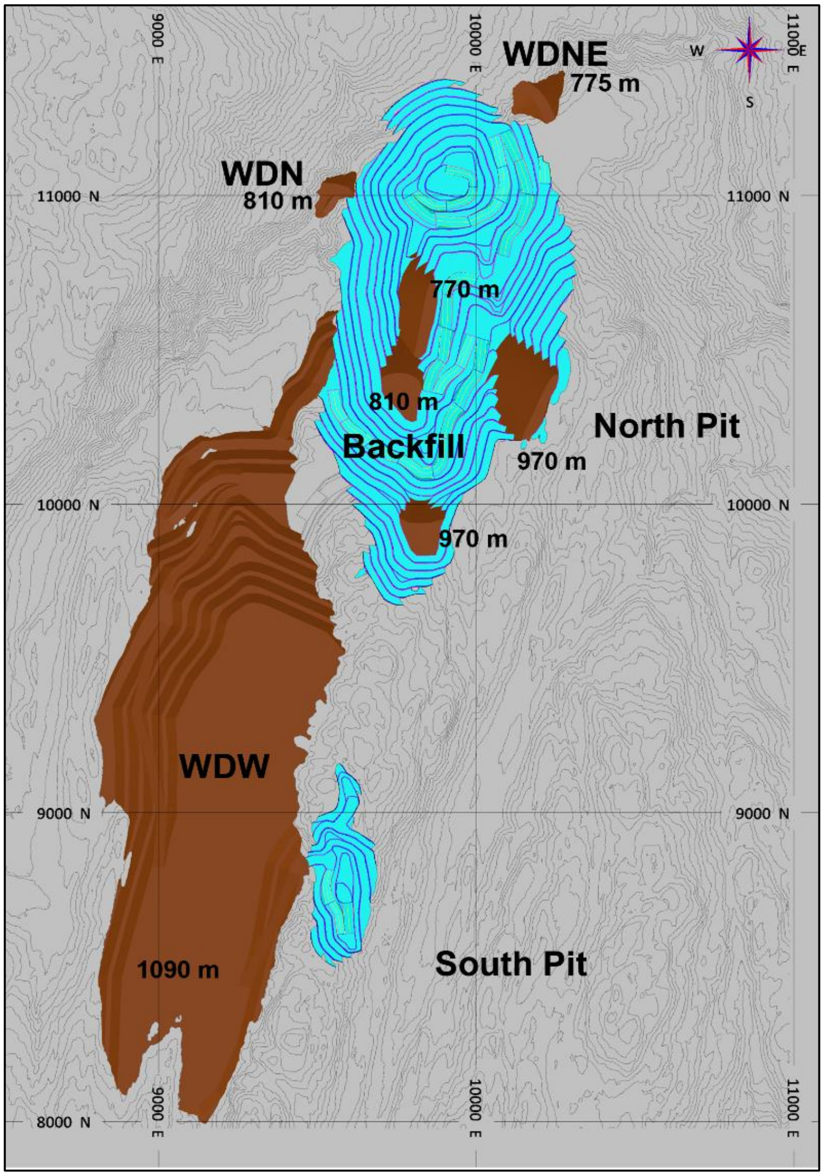
16.11 Waste Rock Storage Facility Design, WDW Stability Analysis, and WDW Water Management

16.11.1 Waste Rock Storage Facility Design

Various rock types are present in the material mined within the final pits. The key difference since the PFS study was revised segregation of PAG and NAG waste rock. Based on recent test work, the only lithologies considered as NAG were hanging wall andesite and upper members of the 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 142 Mt and 81.5 Mt, respectively. The total amount of waste within the pits in the mine plan is 223 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 16-17. The projected storage capacities are shown in Table 16-10.

Figure 16-17: Planned Waste Storage Areas



Note: Figure prepared by AGP, 2022.

Table 16-10: WRSF Parameters

	Units	WDW	WDN	WDNE	North Pit Backfill	TSF Embankment	TSF
Waste storage capacity	Mm^3	60	0.2	0.4	2.8	2.3	40
Maximum elevation	masl	1090	810	775	970	1122	1120

The WRSF design used a swell factor of 1.30. For the WDW facility, the lift height will be 20 m. Assuming a 33.7° face slope, the overall slope will be 24° with 15 m berm widths. A 33.7° 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 TMSF embankment will be constructed with NAG waste, while all PAG material will be sent to the TMSF and submersed below water. The intent is that PAG waste is dumped across the Tom MacKay facility as berms, followed by use of a dragline to retreat and transfer the portion of causeway material above water into the void between berms and 3 m 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.

Contact water from the WDW will be directed to the Contact Water Pond 5 (Pond 5) in both temporary and permanent collection ditch to the west of the pit and used for process water or pump along with the tailings to TMSF. Non-contact from the south of the WDW, including Argillite Creek, will flow under the WDW in a rock drain into the lower section of Argillite Creek.

16.11.2 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 BC's standard of practice guidelines.

16.11.3 WDW Water Management

Refer to Section 20.2.3 for WDW water management.

16.12 Mine Schedule

The mine schedule plans to deliver 29.9 Mt of mill feed grading 2.99 g/t gold and 79 g/t silver over a mine life of eight years. Waste tonnage from the pits totalling 223 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is 7.5:1. The detailed planned mine schedule is shown in Table 16-11 and Table 16-12, as well as by phase in Table 16-13 and Figure 16-18. Figure 16-19 and Figure 16-20 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 3.0 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. Processing will increase to 3.7 Mt/a in year 6 and continue at that rate until year 9. 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 eight years of mining. Mill feed is stockpiled during the pre-production years. Four stockpiles were used for this schedule where:

- VLG Rhyolite = rhyolite material with $24.45/t < NSR < $35/t
- VLG HW/MS = hangingwall/mudstone material with $24.45/t < NSR < $35/t
- LG Rhyolite = rhyolite material with $35/t < NSR < $70/t
- LG HW/MS = hangingwall/mudstone material with $35/t < NSR < $70/t
- MG = material with $70/t < NSR < $150/t, and
- HG = material with NSR > $150/t

HW/mudstone material has a significant effect on the plant material flowability or throughput, so a minimum proportion of 55 % rhyolite was targeted for the mill feed.

A total stockpile capacity of approximately 6.0 Mt was reached late in this schedule. If space is found to be too restrictive during operations, LG stockpiles may need to be placed on selected benches of the waste facilities or blended into the mill feed as better ore characteristics are determined during operation. 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 TMSF embankments were scheduled to be constructed in Years -1 to Year 6. A total of 2.3 million cubic metres were scheduled to be sourced from the mining areas.

Years -3 and -2 have the technical sample and quarries 1 and 2 being the primary mining areas. In Year -2, the upper benches of Phase 1 of the north pit are started. Roadbuilding and site infrastructure will be the predominant activities in these periods. In year -1, a 10kt mineralized sample will be processed to test the technical properties of the mill feed. The final bench elevations for the technical sample and phase 1 were 995 m and 1040 m, respectively. The Quarry 2 will be mined down to 980 m elevation.

Year -1 has ore and waste mined tonnages of 687 kt and 6.8 Mt, respectively. 365 kt of mill feed will be processed in this period as part of commissioning. At the end of pre-production, the crusher stockpile will contain 459 kt of total mill feed material in anticipation of additional plant ramp-up months. Phase 1 ends up on 1000 m bench while the WDW waste facility reaches the 970 m lift.

Table 16-11: Mine Schedule (mining summary and processed material)

Description		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total
Mining summary	NAG Waste (Mt)	0.5	2.4	4.5	22.0	26.9	26.1	21.7	14.0	12.9	10.1	0.9		142
	PAG Waste (Mt)	0.0	1.0	2.3	5.0	11.7	12.8	12.1	11.9	12.7	9.7	2.3		82
	Mined Waste (Mt)	0.5	3.4	6.8	27.0	38.7	38.9	33.8	25.8	25.7	19.8	3.2		223
	Mined Ore (Mt)	0.0	0.1	0.7	3.5	3.3	3.1	4.1	5.0	4.3	4.2	1.4		29.9
	Au (g/t)	0.00	1.58	1.55	3.71	3.06	4.10	3.24	3.00	2.24	2.15	2.61		2.99
	Ag (g/t)	0	13	56	68	78	112	111	82	57	47	88		79
	Sb (ppm)	0	309	333	2245	1809	1607	1432	753	456	396	710		1137
	Hg (ppm)	0.00	30.43	25.53	150.62	89.69	87.38	46.04	20.63	14.16	9.69	15.78		52.11
	As (ppm)	0	888	562	2342	605	694	476	608	415	349	399		741
	Pb (%)	0.00	0.00	0.08	0.10	0.18	0.30	0.52	0.62	0.41	0.62	1.10		0.45
	Zn (%)	0.00	0.01	0.14	0.17	0.33	0.53	0.84	0.96	0.65	0.95	1.70		0.72
	Cu (%)	0.00	0.00	0.01	0.02	0.04	0.07	0.09	0.09	0.06	0.07	0.14		0.07
	Fe (%)	0.00	1.43	1.45	1.63	1.97	2.54	2.55	2.21	2.02	2.21	3.25		2.29
	S (%)	0.00	1.08	1.20	1.64	1.46	1.84	1.73	1.63	1.53	1.69	2.49		1.73
	Mined Total (Mt)	0.5	3.5	7.5	30.6	42.0	42.0	37.9	30.8	30.0	24.0	4.6		253
Processed Material	Mill Feed (Mt)	0.0	0.0	0.4	2.7	3.0	3.0	3.0	3.0	3.7	3.7	3.7	3.7	29.9
	Au (g/t)	0.00	0.00	1.89	4.47	3.28	4.21	4.12	4.26	2.50	2.45	1.72	1.12	2.99
	Ag (g/t)	0	0	74	76	86	115	142	123	65	52	50	27	79
	Sb (ppm)	0	0	351	2492	1954	1695	1761	1037	522	452	532	580	1137
	Hg (ppm)	0.00	0.00	26.58	177.04	94.60	94.35	57.38	27.70	16.20	11.16	15.94	22.55	52.11
	As (ppm)	0	0	630	2943	641	686	552	755	440	382	393	404	741
	Pb (%)	0.00	0.00	0.07	0.09	0.19	0.30	0.59	0.82	0.45	0.67	0.62	0.32	0.45
	Zn (%)	0.00	0.00	0.15	0.16	0.33	0.53	0.96	1.27	0.71	1.03	0.96	0.49	0.72
	Cu (%)	0.00	0.00	0.01	0.02	0.04	0.07	0.11	0.12	0.07	0.08	0.08	0.04	0.07
	Fe (%)	0.00	0.00	1.47	1.61	1.89	2.33	2.55	2.36	2.07	2.43	2.67	2.58	2.29
	S (%)	0.00	0.00	1.21	1.70	1.46	1.79	1.79	1.79	1.59	1.85	1.94	1.65	1.73

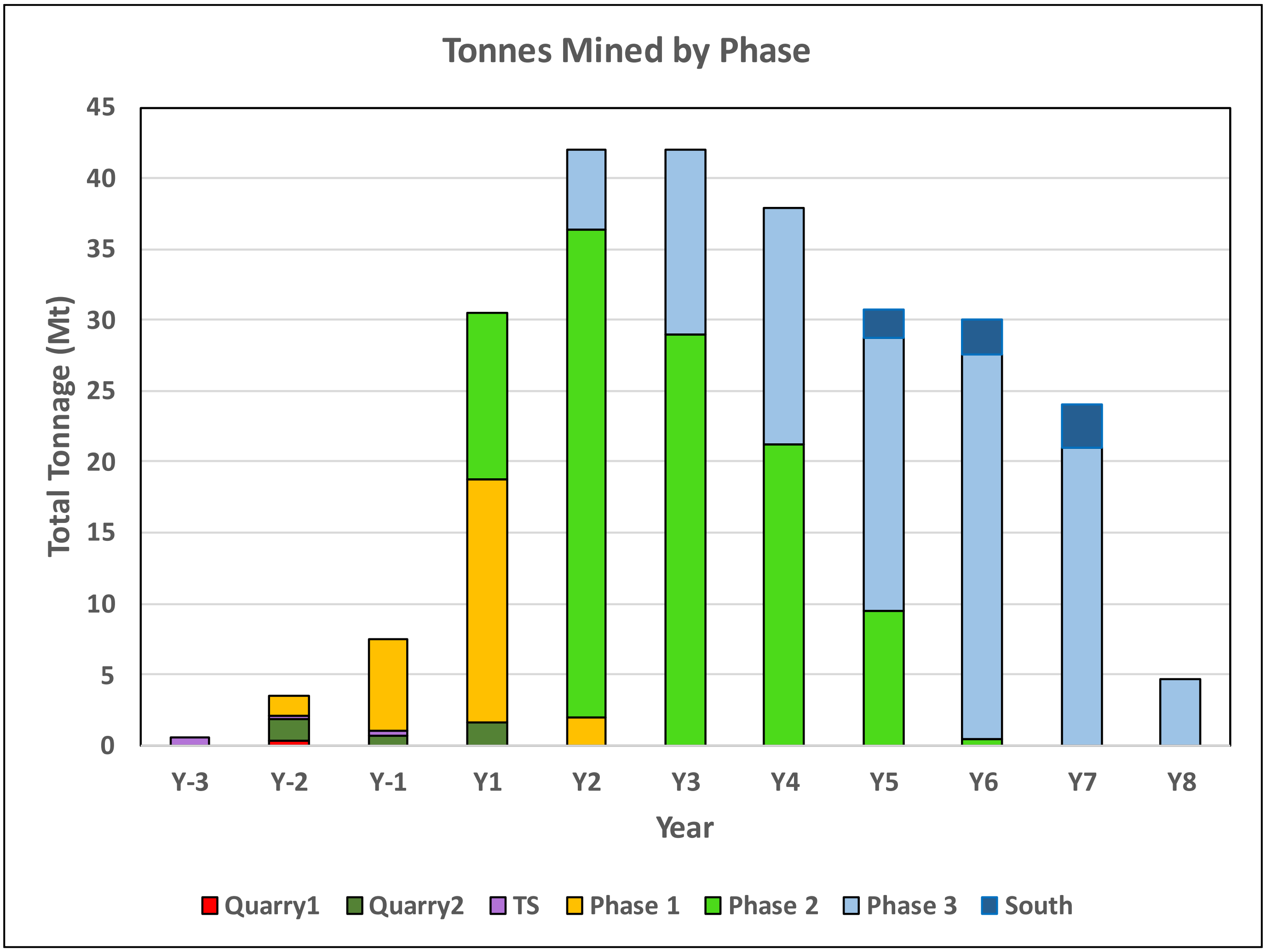
Table 16-12: Mine Schedule (mining summary and processed material)

Description		Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total
Stockpile Balance	Rhyolite VLG (kt), NSR=$24.45-$35/t	0	15	18	51	51	41	152	346	30	358	413	0	
	HW/MS VLG (kt), NSR=$24.45-$35/t	0	0	0	1	1	1	81	194	194	224	240	0	
	Rhyolite LG (kt), NSR=$35-$70/t	0	79	333	998	1338	1531	1843	2088	3037	3212	2759	0	
	HW/MS LG (kt), NSR=$35-$70/t	0	2	2	0	0	0	120	176	188	213	221	0	
	MG (kt), NSR=$70-$150/t	0	41	107	209	209	153	422	567	567	776	81	15	
	HG (kt), NSR> $150/t	0	1	0	0	0	0	226	1426	1426	1196	4	4	
Total Stockpile Balance	(Mt)	0.0	0.1	0.5	1.3	1.6	1.7	2.8	4.8	5.4	6.0	3.7	0.0	
Total Stockpile Reclaim	(Mt)	0.0	0.0	0.2	0.9	0.5	0.6	0.0	0.0	0.3	0.5	2.5	3.7	9.1
Total Material Movement	(Mt)	0.5	3.5	7.7	31.4	42.5	42.6	37.9	30.8	30.3	24.5	7.1	3.7	263

Table 16-13: Tonnes Mined by Phase

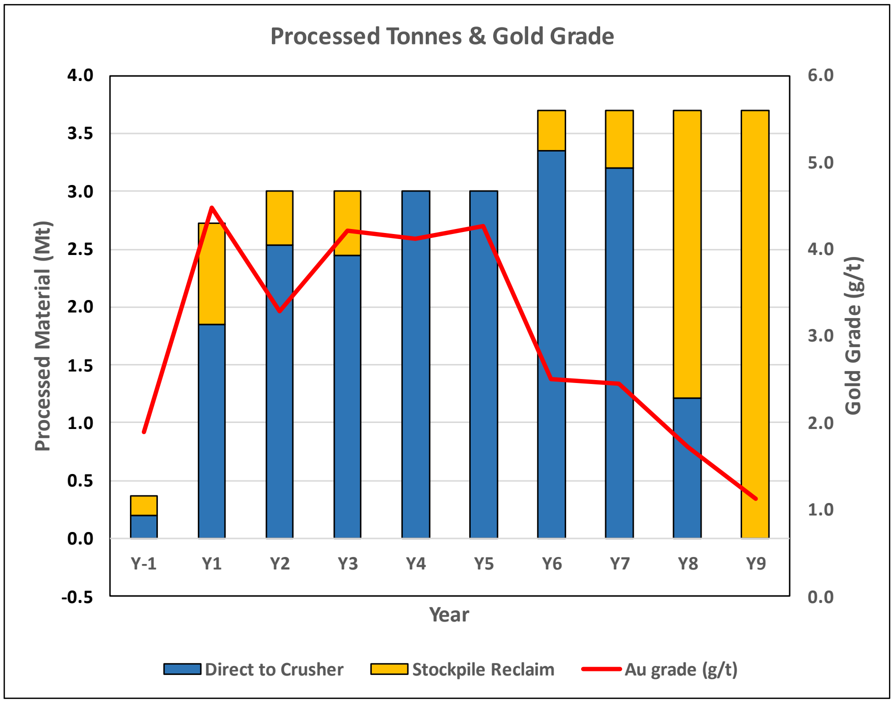
Phase	Total Tonnage (Mt)											Total (Mt)
	Y-3	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	
Quarry1	0.0	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4
Quarry2	0.0	1.5	0.6	1.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	3.7
TS	0.5	0.2	0.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.1
1	0.0	1.5	6.4	17.2	2.0	0.0	0.0	0.0	0.0	0.0	0.0	27.1
2	0.0	0.0	0.0	11.8	34.4	29.0	21.2	9.5	0.5	0.0	0.0	106.4
3	0.0	0.0	0.0	0.0	5.6	13.0	16.7	19.2	27.1	21.0	4.6	107.2
South	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.1	2.4	3.0	0.0	7.5
Total	**0.5**	**3.5**	**7.5**	**30.6**	**42.0**	**42.0**	**37.9**	**30.8**	**30.0**	**24.0**	**4.6**	**253.4**

Figure 16-18: Tonnes Mined by Phase



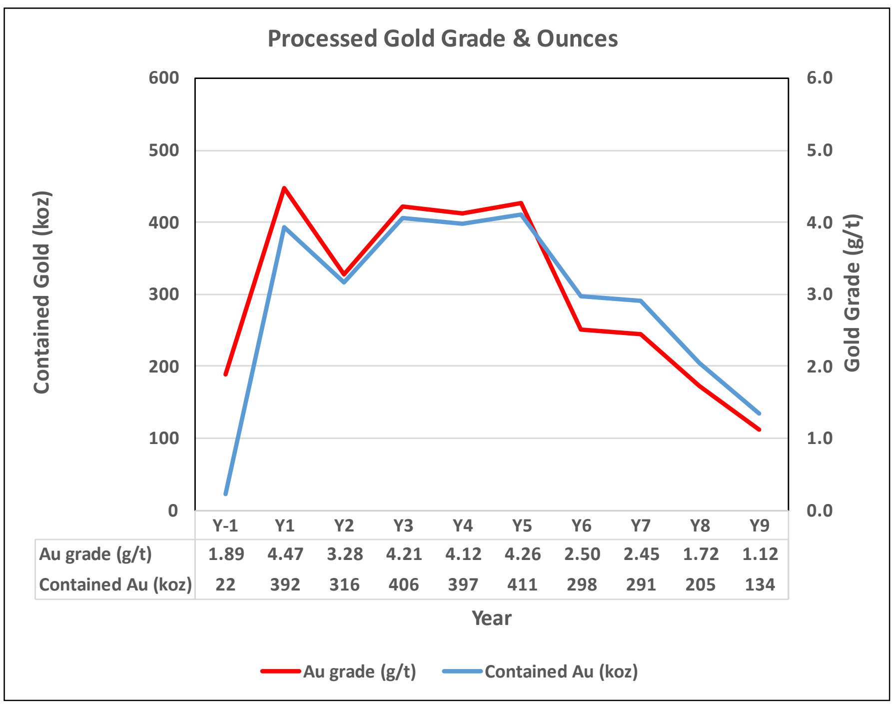
Note: Figure prepared by AGP, 2022.

Figure 16-19: Planned Life of Mine Mill Feed Tonnes and Ounces



Note: Figure prepared by AGP, 2022.

Figure 16-20: Process Grade and Contained Ounces of Gold



Note: Figure prepared by AGP, 2022.

Table 16-14 displays a summary of the reserve classifications for the mill feed.

Year 1 production assumes the plant will require twelve months to achieve a full production rate of 3 Mtpa. Ramp-up tonnages started in year -1 and steadily increased until full production was reached in year 1 month 9. 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 2.73 Mt. Mill feed will be from stockpile or direct feed from Phase 1 and Phase 2. All NAG waste will be directed to the WDW facility and the TMSF embankments.

Year 2 production will be at the full 3.0 Mt of mill feed. Phases 1 mining will be completed to a final level of 920 masl. Phases 2 continues mining while phase 3 is started. Phases 2 and 3 will be mined down to the levels of 900 masl, and 920 masl respectively. All NAG waste will be directed to the WDW facility and the TMSF embankments.

Year 3 production will see Phase 2 as the dominant phase of mining in this period, driving to a depth of 840 masl. Phase 3 is the only other active phase and advances down to level 870masl. All NAG waste will be directed to the WDW facility and the TMSF embankments. A small portion of NAG waste material will be directed back into the pit as backfill as space allows along the south wall of 970 m bench. Work will start in year 3 to prepare for the diversion tunnel, including initial access to the inlet and outlet locations.

Year 4 production will again have Phases 2 and 3 as the only active phases with their final levels being 790 masl and 840 masl respectively. NAG waste will be sent to the WDW facility up to 1070 masl lift. The diversion tunnel mining will need to be conducted in year 4. Sheet piles will need to be installed so that debris grates and sheet curtains can be installed in year 5 to divert the water channel into the tunnel.

Year 5 will see the Tom MacKay Creek being diverted into the diversion tunnel. This is the final year where the plant operates at 3.0 Mtpa due changing ore properties and increased hardness in the remaining years. Initial mining will begin in the south pit. Phases 2 and 3 are also active mining phases and will be advanced down to 740 masl and 800 masl levels respectively. Most of the NAG waste will be directed to the WDW facility to 1070 and 1090 masl elevations, but WDN and WDNE are both dumped in complete across Tom MacKay Creek as well. WDNE will be used for accessing upper benches on the north side of the creek.

Year 6 will have Phase 2 being completed it is advanced down to 730 masl. Phase 3 and the south pit continue to be mined and are mined down to 750 masl and 1060 masl respectively. NAG waste will be sent to the WDW facility up to the final elevation of 1090 masl and to the backfill at 970 masl elevation along the east wall of the north pit.

Year 7 will have the south pit being mined down to its lowest elevation of 1020 masl. Phase 3 continuing to be mined and advanced down to 700 masl. NAG waste material will be directed to the backfill dump at 810 masl, with the remainder being directed to the WDW facility.

Year 8 will be the final mining period with mining being completed in Phase 3. Phase 3 will have mining completed on the 650 masl level. The waste in this period will be sent as much as possible to backfill locations with the remainder being sent to WDW.

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 29.9 Mt of mill feed grading 2.99 g/t Au and 78.5 g/t Ag. NAG waste totalling 142 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 82 Mt will also be directed to the TMSF and submersed below water. The overall strip ratio is 7.5:1.

Table 16-14: Reserve Summary of Scheduled Material

Reserve Class	Mill Feed (Mt)	Grade		Contained Ounces	
		Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Proven	17.3	3.64	99	2.02	55.1
Probable	12.6	2.10	50	0.85	20.5
Total	**29.9**	**2.99**	**79**	**2.87**	**75.5**

16.13 Mine Plan Sequence

Anticipated end-of-year positions for the open pits are shown in are shown in Figure 16-21 to Figure 16-30.

Mining will be initiated in the north pit and will continue throughout the schedule, while the south pit will only be active in years 5 to 7.

16.14 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 with 165 mm bits. This will provide the capability to drill patterns for either 5 m or 10 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 small loaders loading in tandem. 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 LOM plan are included in Section 21.

16.15 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; and
- Location of previous underground infrastructure prior to blasthole rigs drilling.

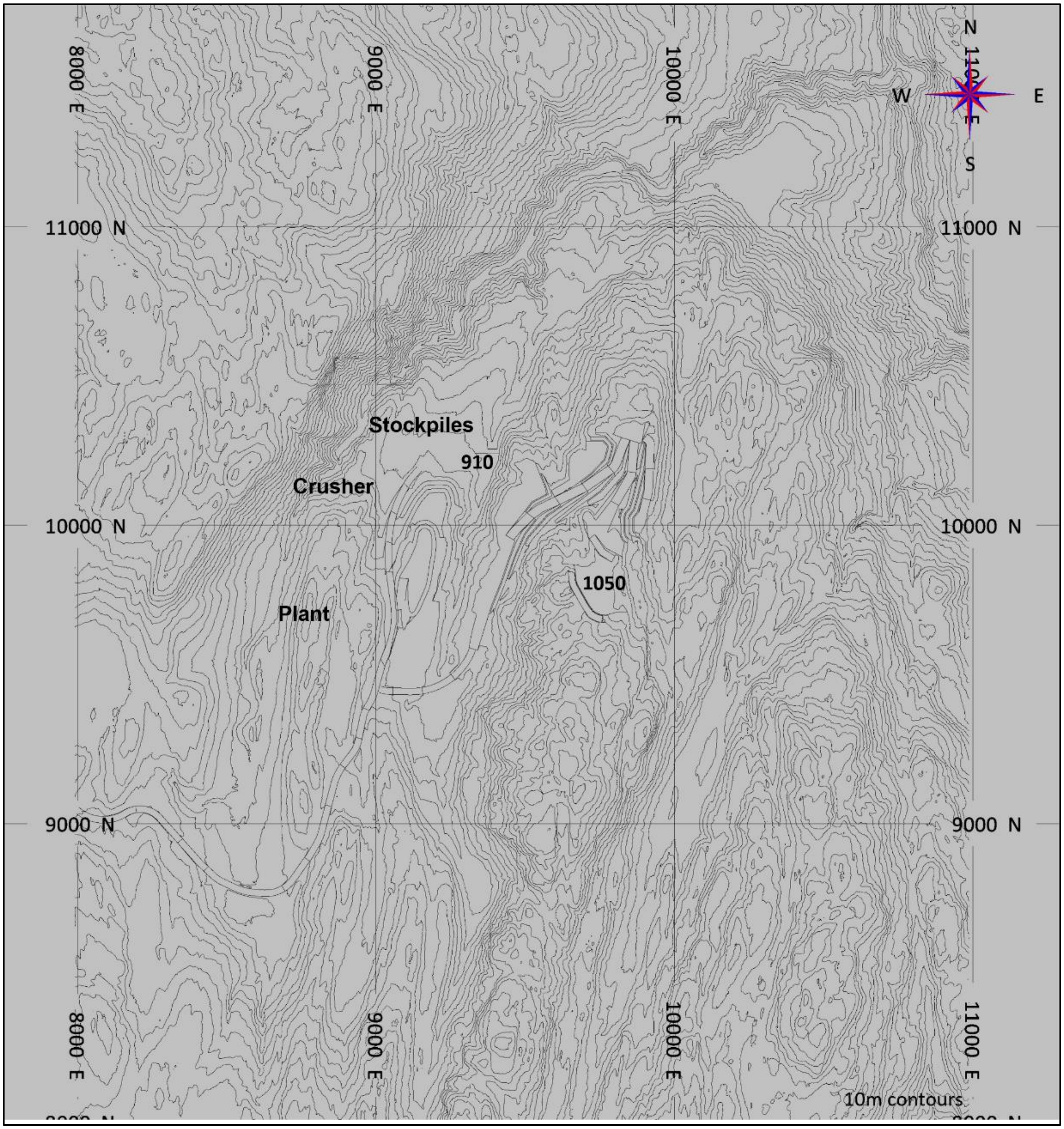
Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model. Blasthole sampling will also be part of the grade control program initially to determine the best method for Eskay Creek.

Blasthole samples will be used to assay for PAG waste qualities. 25% of the ore blast holes and 80% of the waste holes were assumed to be sampled for this purpose.

16.16 End of Period Plans

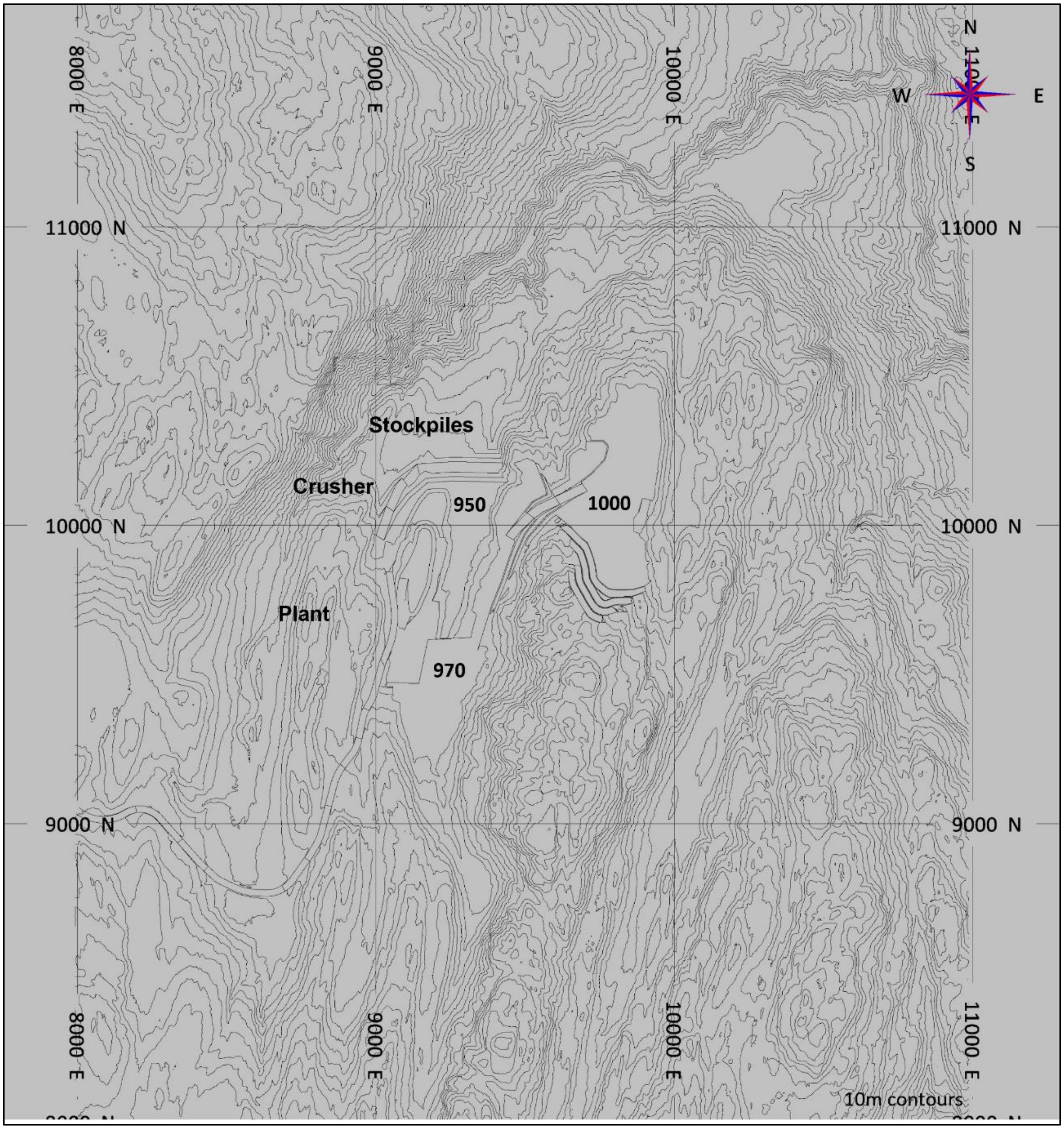
Images of the end of period positions in the pit of the waste dumps and pits are shown in Figure 16-21 to Figure 16-30.

Figure 16-21: End of Preproduction Period – Year-2



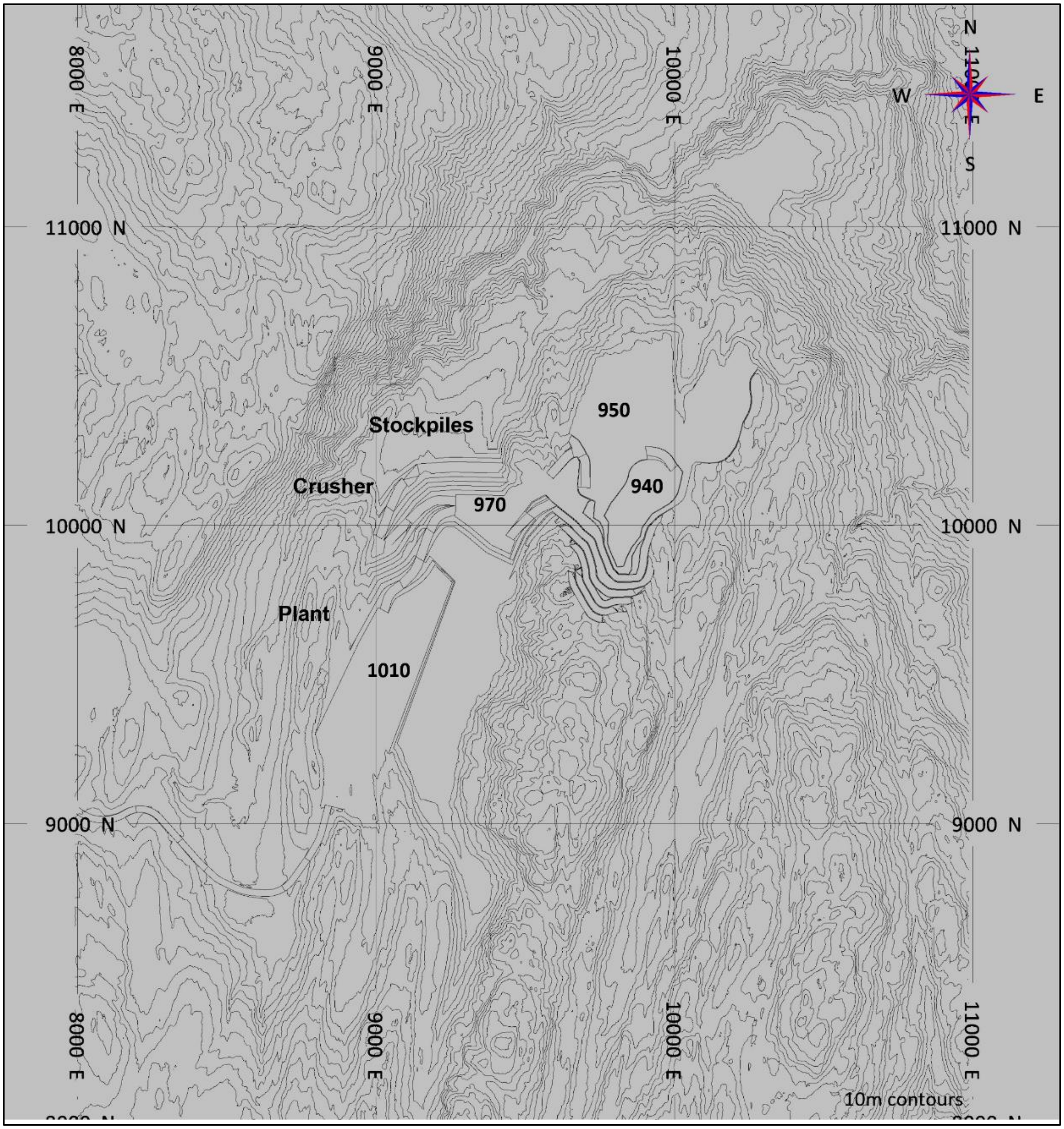
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-22: End of Preproduction period – Year-1



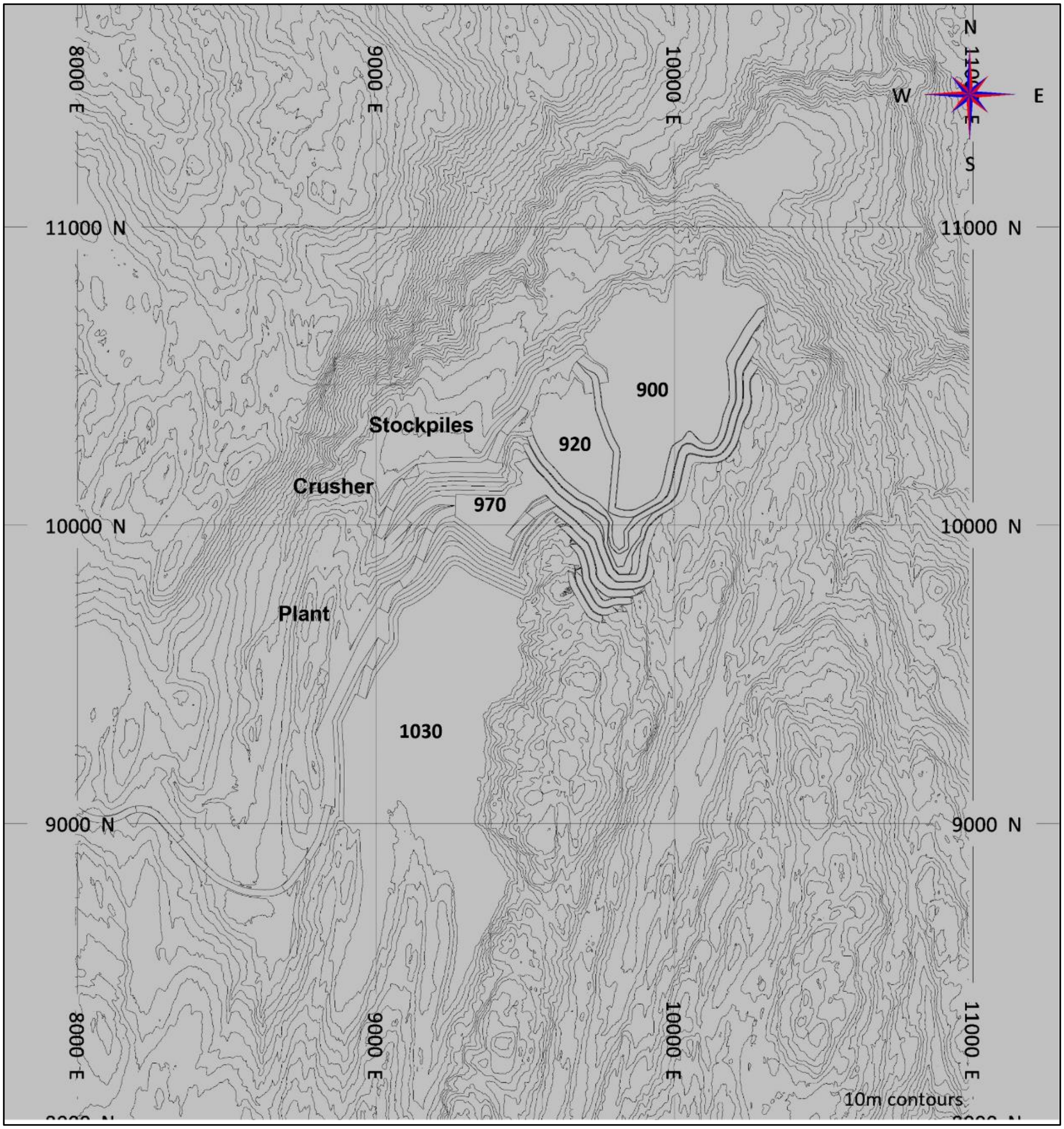
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-23: End of Year 1



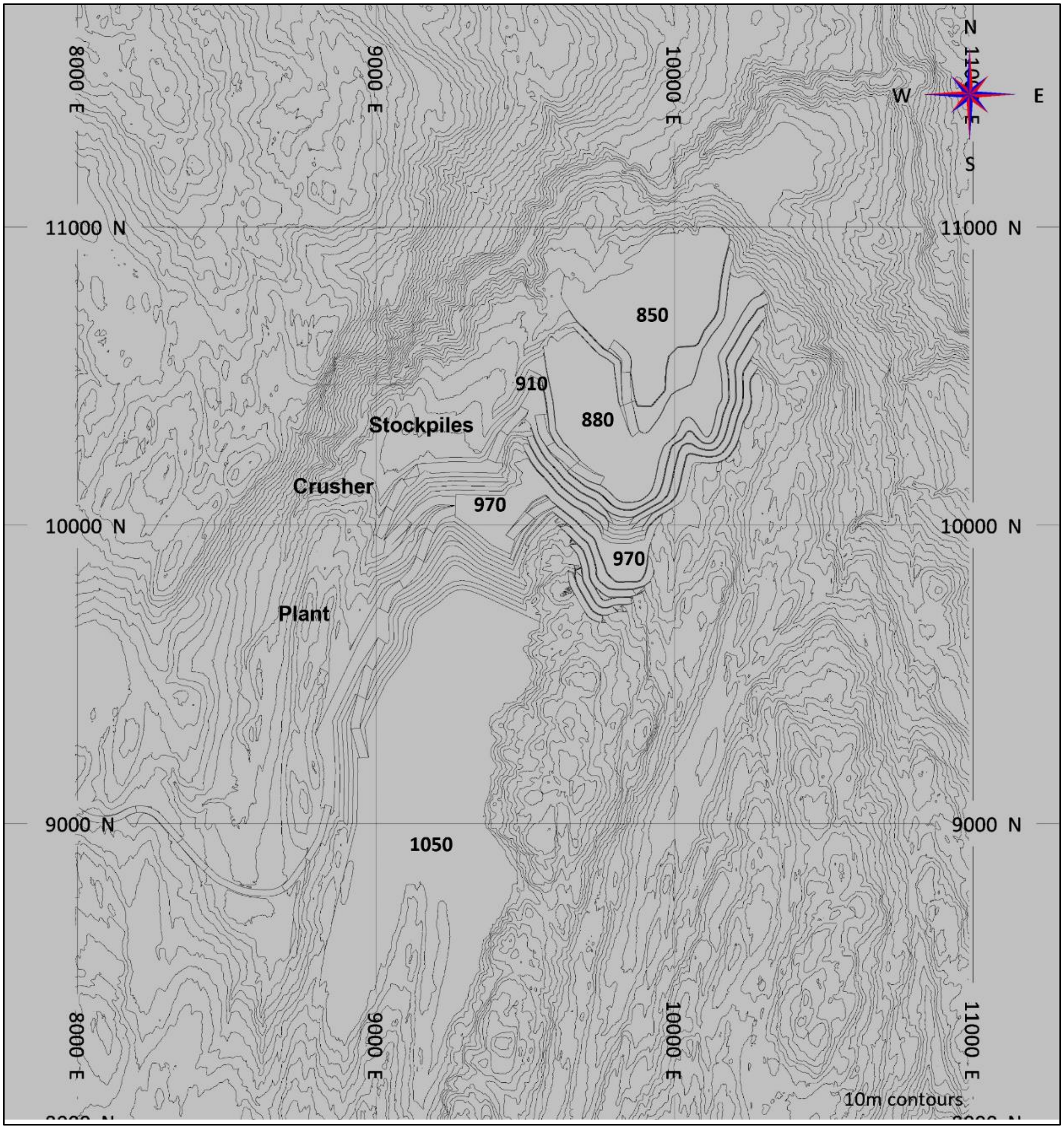
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-24: End of Year 2



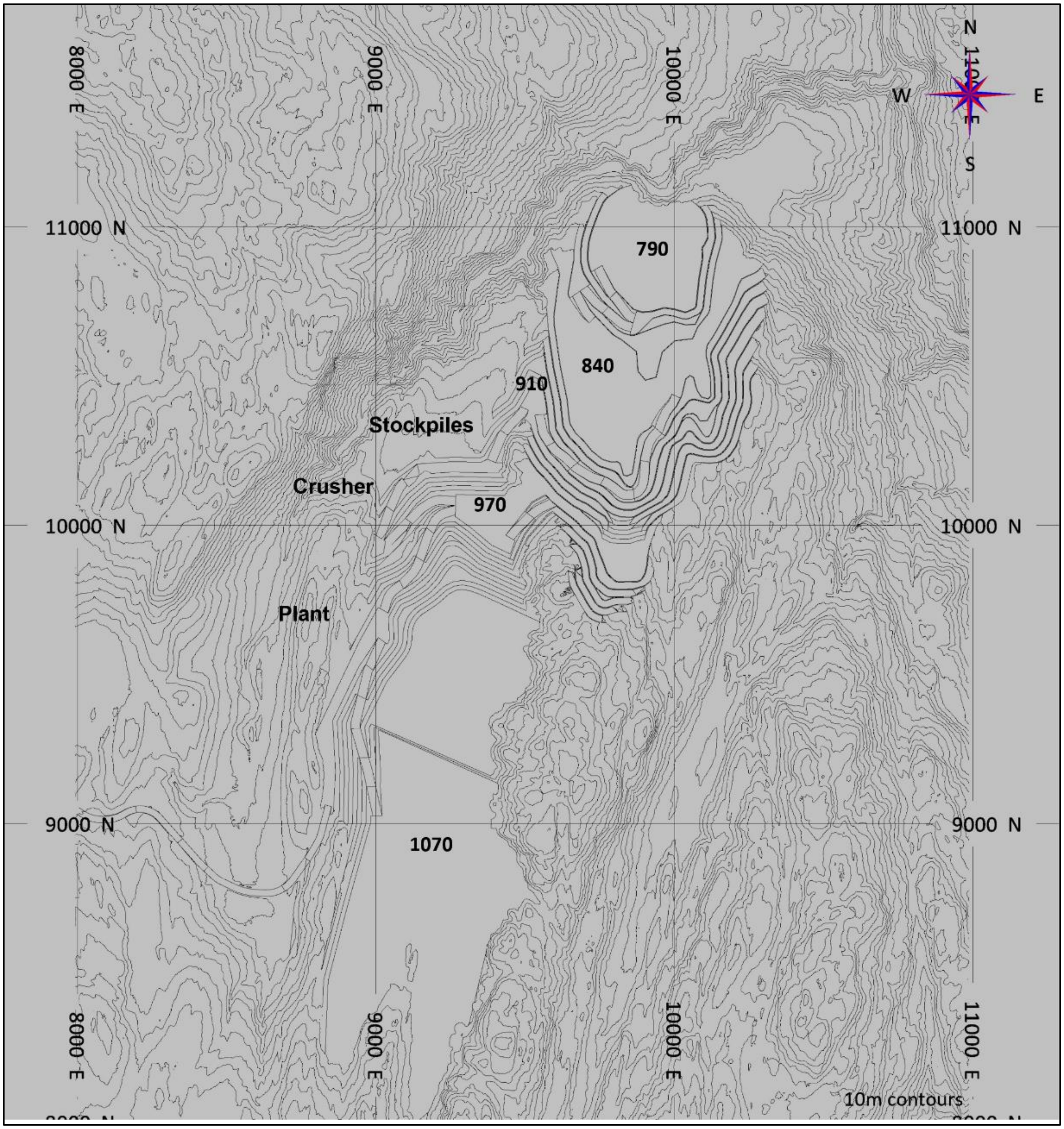
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-25: End of Year 3



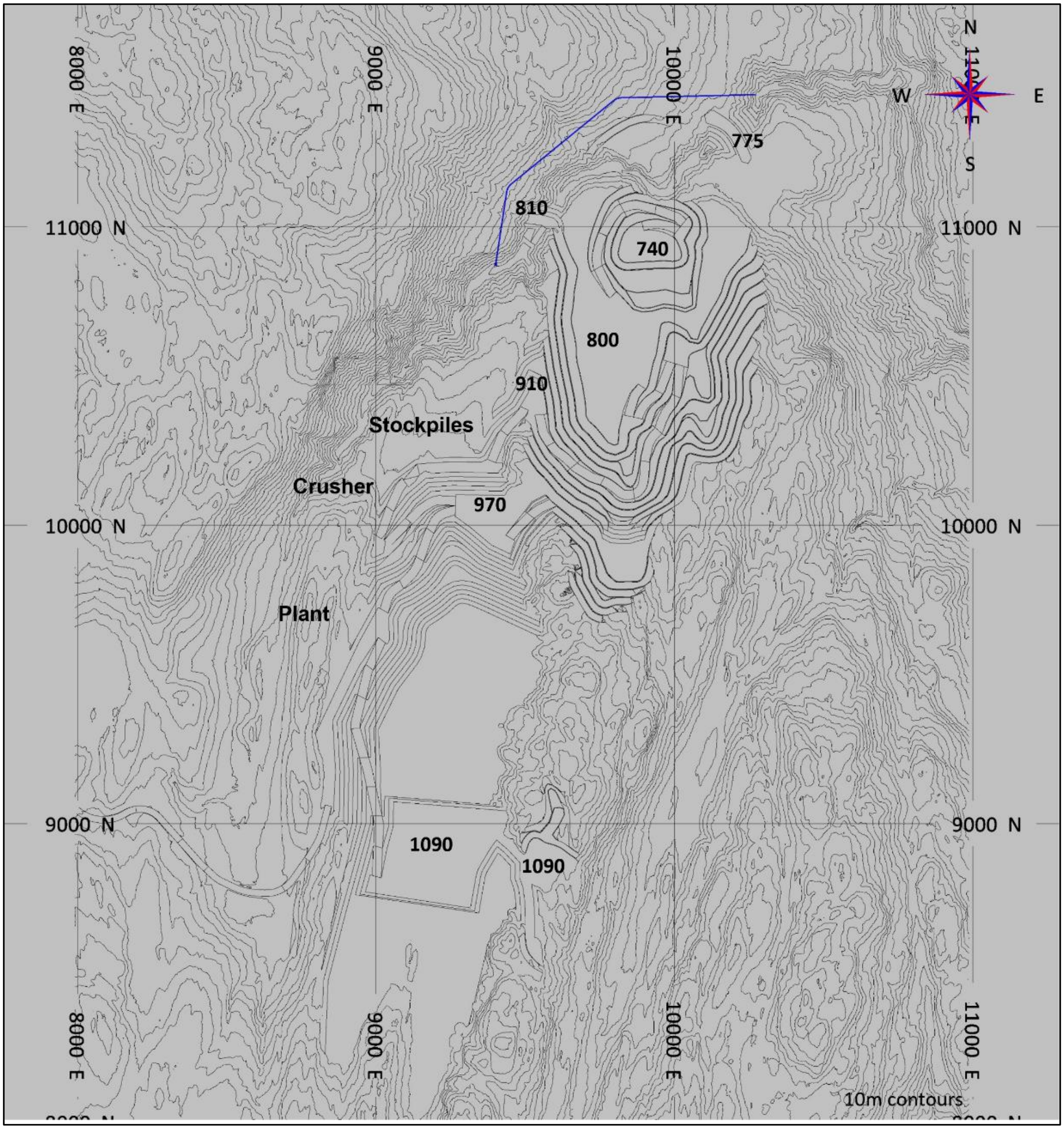
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-26: End of Year 4



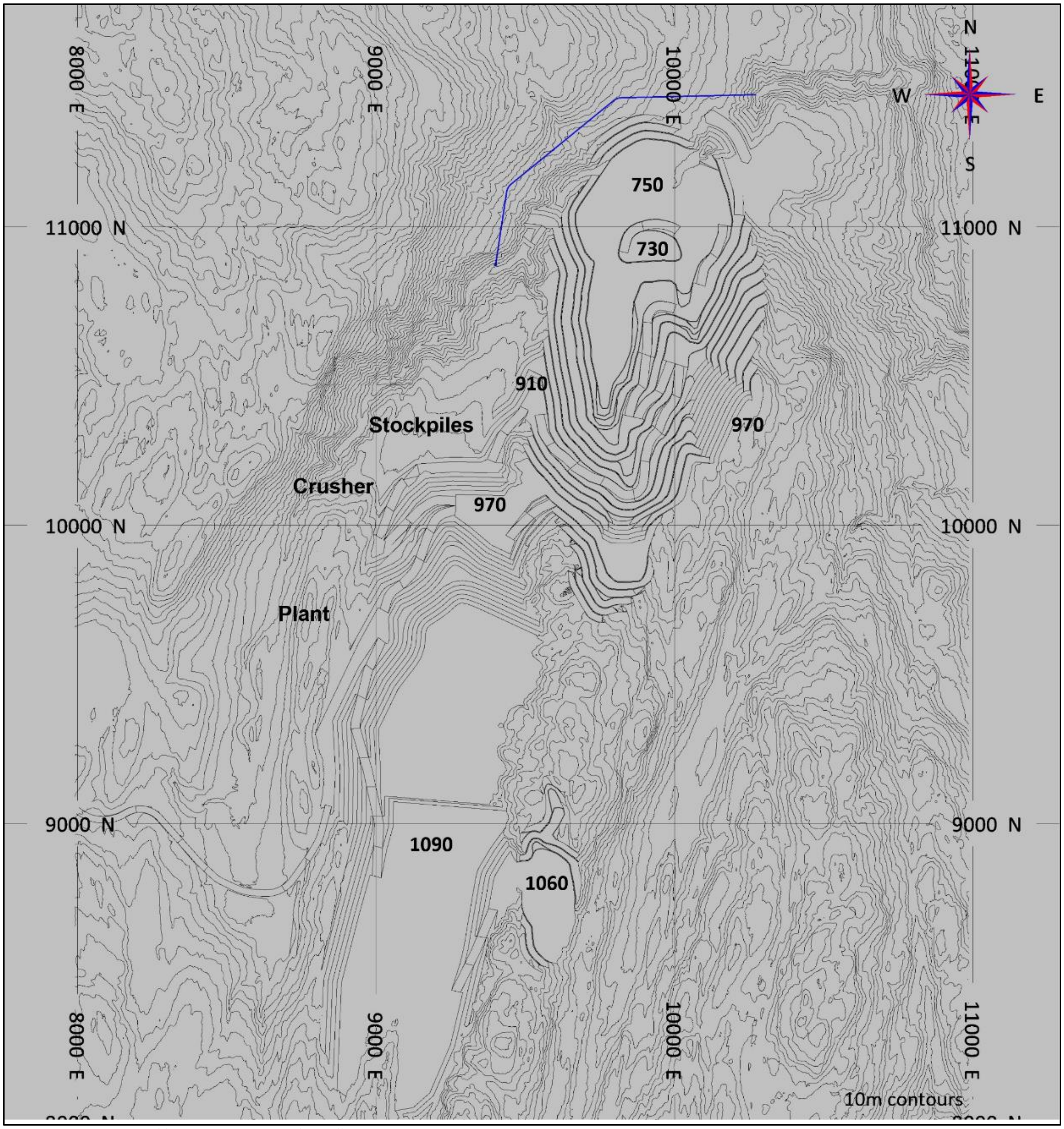
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-27: End of Year 5



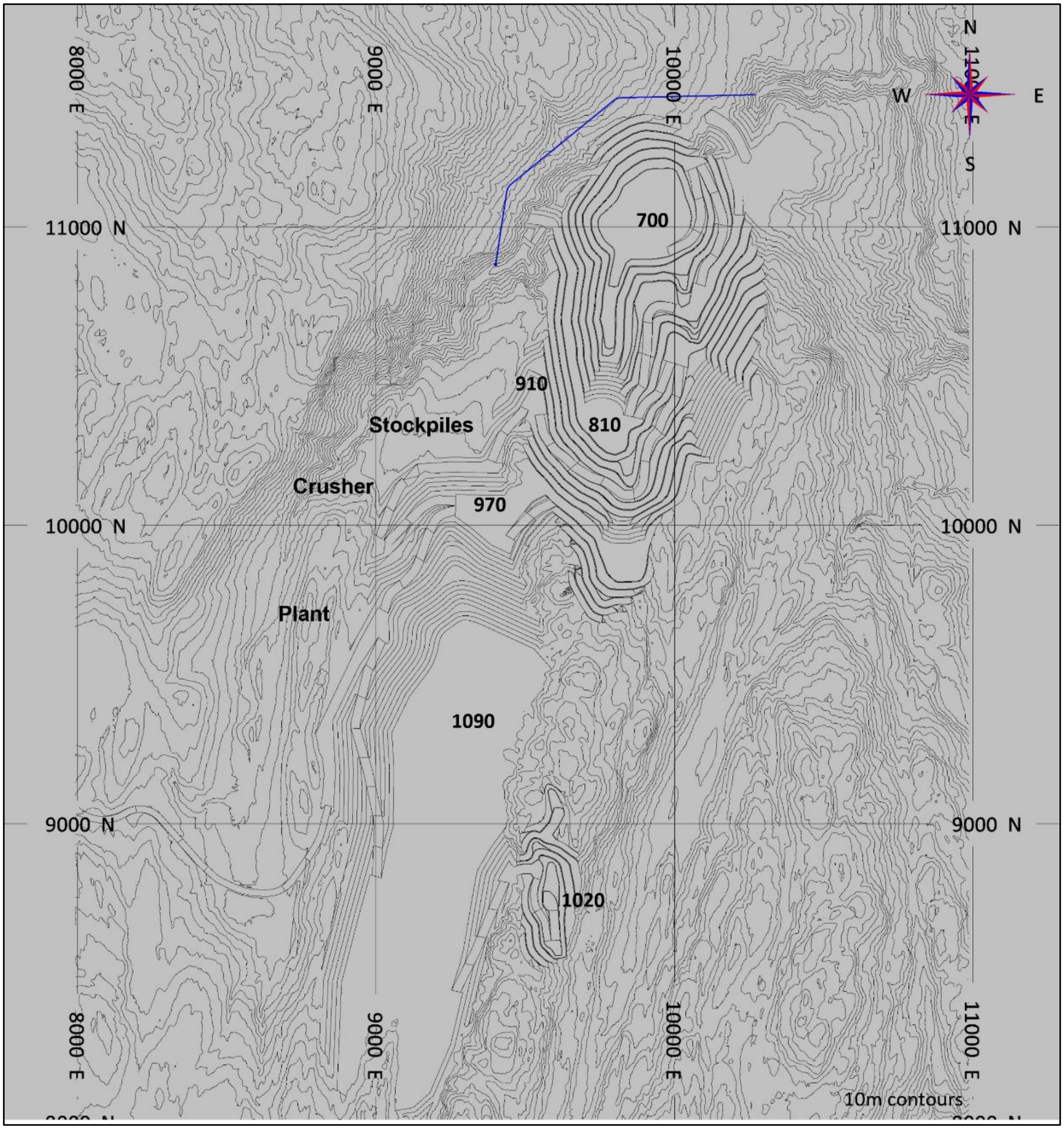
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-28: End of Year 6



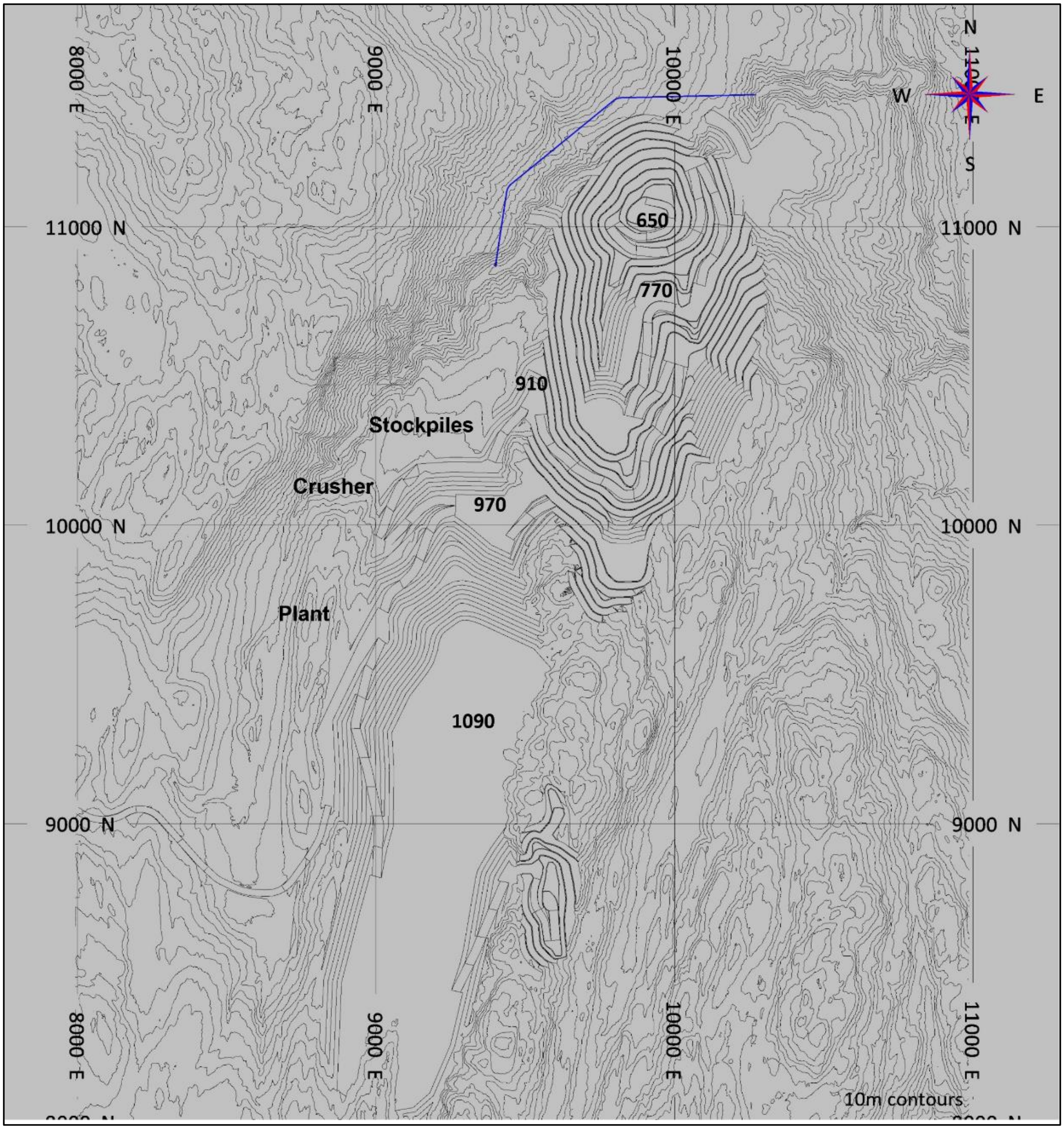
Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-29: End of Year 7



Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

Figure 16-30: End of Year 8



Note: Figure prepared by AGP, 2022. Each gridline represents 1000 metres.

17 RECOVERY METHODS

17.1 Overall Process Design

The testwork provided was thoroughly analysed and several options of process routes were addressed in the initial stages of the feasibility study. Based on the analysis, a process route was chosen as the best suited for the testwork results and subsequent economic analysis for the material. The unit operations selected are typical for this industry.

The project will be constructed in two distinct phases, as follows:

- Initial operation of 3.0 Mt/a for years 1 to 5, which comprises:
  - 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 semi-autogenous grinding (SAG) mill, pebble crusher (installed for year 4 operations), and ball mill in closed circuit with hydrocyclones
  - rougher flotation with concentrate regrind and two stages of cleaning
  - rougher tails slimes classification via two stages of hydrocyclones
  - secondary grinding including ball mill and IsaMill and scavenger flotation, fed from the slimes circuit underflow
  - fines flotation and two stages of cleaning, fed from the slimes circuit overflow
  - concentrate thickening, filtration, drying and storage
  - concentrate load-out by way of front-end loader filling concentrate transportation
  - final tailings pumping to the TMSF.
- Expansion to 3.7 Mt/a for the remaining mine life, which includes the initial equipment with the addition of the following installed for year 6 operation:
  - secondary crushing circuit (cone)
  - A second ball and extra cyclones in primary grinding circuit
  - additional IsaMill in secondary grinding circuit

Key process design criteria are listed below:

- initial operation nominal throughput of 8,220 t/d or 3.0 Mt/a
- expansion nominal throughput of 10,140 t/d or 3.7 Mt/a
- average head grade of 2.99 g/t Au and 79 g/t Ag
- crushing plant availability of 70%
- operate two shifts per day, 365 d/a with process plant availability of 92% for grinding, flotation which equates to 8,059 operating hours per year, with standby equipment in critical areas
- product will be gold concentrate to be sold to refineries
- sufficient process plant design flexibility for treatment of all ore types.

17.2 Process Plant Design Criteria

The key process design criteria for the mill are listed in Table 17-2, and the comminution parameters are provided in Table 17-3. The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery. Annual process production is shown in Table 17-1.

Table 17-1: Eskay Creek Annual Gold and Silver Production

Production Year		1	2	3	4	5	6	7	8	9	Total
Total Mill Feed	Mt	3.09	3.00	3.00	3.00	3.00	3.70	3.70	3.70	3.72	29.91
Au Head Grade	g/t	4.17	3.28	4.21	4.12	4.26	2.50	2.45	1.72	1.12	2.99
Ag Head Grade	g/t	75.93	85.67	114.84	142.27	122.87	64.80	52.41	50.10	26.54	78.55
AuEq Head Grade	g/t	5.02	4.24	5.50	5.71	5.64	3.23	3.03	2.28	1.42	3.87
Contained Gold	kozs	415	316	406	397	411	298	291	205	134	2874
Contained Silver	kozs	7549	8263	11077	13723	11851	7708	6235	5960	3173	75538
Contained Gold Equivalent	kozs	499	409	530	551	544	384	361	271	170	3718
Recovery Au	%	87%	84%	84%	85%	86%	83%	82%	82%	76%	84%
Recovery Ag	%	89%	88%	89%	90%	90%	87%	86%	86%	82%	88%
Recovered Gold in Concentrate	kozs	361.35	266.33	341.96	336.51	354.09	248.72	240.06	167.99	101.59	2419
Recovered Silver in Concentrate	kozs	6740.13	7305.48	9813.72	12317.35	10704.94	6712.56	5360.34	5148.26	2604.54	66707
Recovered Gold Equivalent in Concentrate	kozs	436.68	356.58	463.19	488.66	486.32	331.64	306.28	231.58	133.77	3235

Table 17-2: Eskay Creek Process Design Criteria – Overview

Description	Units	Value
Ore Throughput (base case, years 1-5)	Mt/y	3.0
Ore Throughput (base case, years 6+)	Mt/y	3.7
Process Plant availability	%	92
Filter plant availability	%	85
Daily throughput – average (years 1-5)	kt/d	8.22
Daily throughput – average (years 6+)	kt/d	10.14
Process Plant capacity, nominal @ 92% availability (years 1-5)	t/h	372
Process Plant capacity, nominal @ 92% availability (years 6+)	t/h	459
Recovery to concentrate, mass	% plant feed	5-10
ROM specific gravity	SG	2.9
Concentrate grade, Au	g/t	25-50

Table 17-3: Comminution Design Criteria

Description	Units	Year 1 – 3	Year 4-5	Year 6+
Crushing (Single Stage)				
Availability	%	70		
Primary crusher	type	Jaw Crusher		
Coarse ore stockpile residence time - live	h	8.0		6.5
Primary crushing circuit feed, F_{100}	mm	800		
Bond crusher work index (CWi)	kWh/t	19	18.9	18.5
Secondary Crusher	Type			Cone Crusher
Secondary Crushing circuit feed, F_{100}	mm			278
Primary Grinding				
Availability	%	92		
Circuit type	-	SAG Mill, Ball Mill	Sag Mill, Pebble Crusher, Ball Mill	SAG Mill, Pebble Crusher, Ball Mill
Bond Rod Mill Work Index (RWi)	kWh/t	16.8	18.5	18.9
Bond Ball Mill Work Index (BWi)	kWh/t	18.2	19.0	20.4
A x b	-	42.0	32.6	32.9
Feed particle size, F_{80}	mm	88	105	45.9
Product particle size, P_{80}	µm	100		
Pebble rate, design	% fresh feed	20 (recycled)	30 (crushed)	30 (crushed)

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 – 5	Year 6+
Feed rate	t/h	372	459
Roughers			
Cell type	-	Conventional Tank Cells	
Stage recovery to concentrate, mass	% fresh feed	7-12	
Stage recovery, Au	% fresh feed	35-50	
Regrind Mill			
Type		IsaMill	
Feed rate, design	t/h	41	
Feed rate, nominal	t/h	33	41
Feed size F_{80}	µm	85	
Discharge size P_{80}	µm	15	
Specific grinding energy (SGE)	kWh/t	34.2	
Cleaners		2 stages	
Cell type	-	Conventional Tank Cells	
Second stage recovery to concentrate, mass	% Plant feed	5 – 9	
Second stage recovery, Au	% Plant feed	80-90	
Deslime Cycloning		2 stages	
Stage 1 overflow Size P_{80}	µm	61	
Stage 1: No. of cyclones (operating/standby)	-	6/2	7/2
Stage 2 overflow Size P_{80}	µm	20	
Stage 2: No. of cyclones (operating/standby)	-	8/2	10/2
Secondary Grinding			
Secondary Mill 1			
Type	-	Ball Mill	
Feed rate, nominal	t/h	615	759
Feed size, F_{80}	µm	110	
Discharge size, P_{80}	µm	50	58
SGE	kWh/t	9.0	7.3
Secondary Mill 2			
Type	-	IsaMill	
Feed rate, nominal	t/h	135	130
Feed size, F_{80}	µm	50	58
Discharge size, P_{80}	µm	35	
SGE	kWh/t	10.2	11.1
Secondary Mill 3			

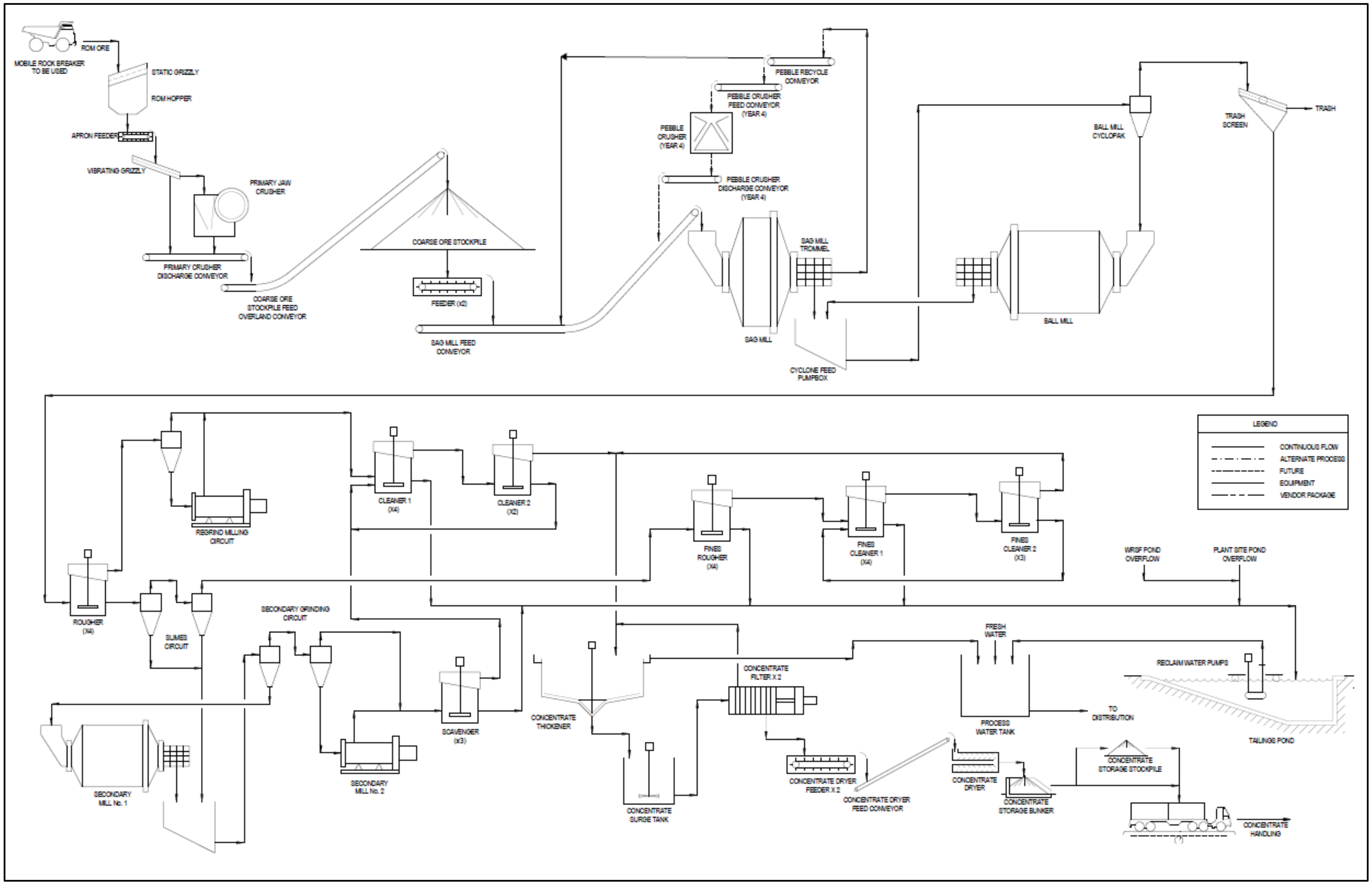
Description	Units	Year 1 – 5	Year 6+
Type	-	-	IsaMill
Feed rate, nominal	t/h	-	52
Feed size, F_{80}	µm	-	58
Discharge size, P_{80}	µm	-	35
SGE	kWh/t	-	11.1
Scavengers			
Cell type	-	Conventional Tank Cells	
Recovery to concentrate, mass	% fresh feed	11 – 15	
Stage recovery, Au	% fresh feed	35-45	
Fines Flotation			
Fines Rougher			
Cell type	-	Conventional Tank Cells	
Recovery to concentrate, mass	% fresh feed	3 – 5	
Stage recovery, Au	% fresh feed	5 -10	
Fines Cleaners		2 stages	
Cell type	-	Conventional Tank Cells	
Recovery to concentrate, mass	% fresh feed	1	
Stage recovery, Au	% fresh feed	4 – 8	
Concentrate Thickener			
Type	-	Hi-rate	
Unit area thickening rate (design)	t/ m^2.h	0.3	
Thickener underflow density	% w/w	55	
Concentrate Filter			
Type	-	Vertical plate pressure filter	
Filtration rate	kg/m2/h	20	
Nominal filter cake moisture	% w/w	15	
Concentrate Drying			
Type	-	Holo-flite	
Dryer feed moisture design	% w/w	21.5	
Dryer cake moisture	% w/w	<13.5	

17.3 Process Flowsheet and Layout Drawings

The simplified process flowsheet is shown in Figure 17-1.

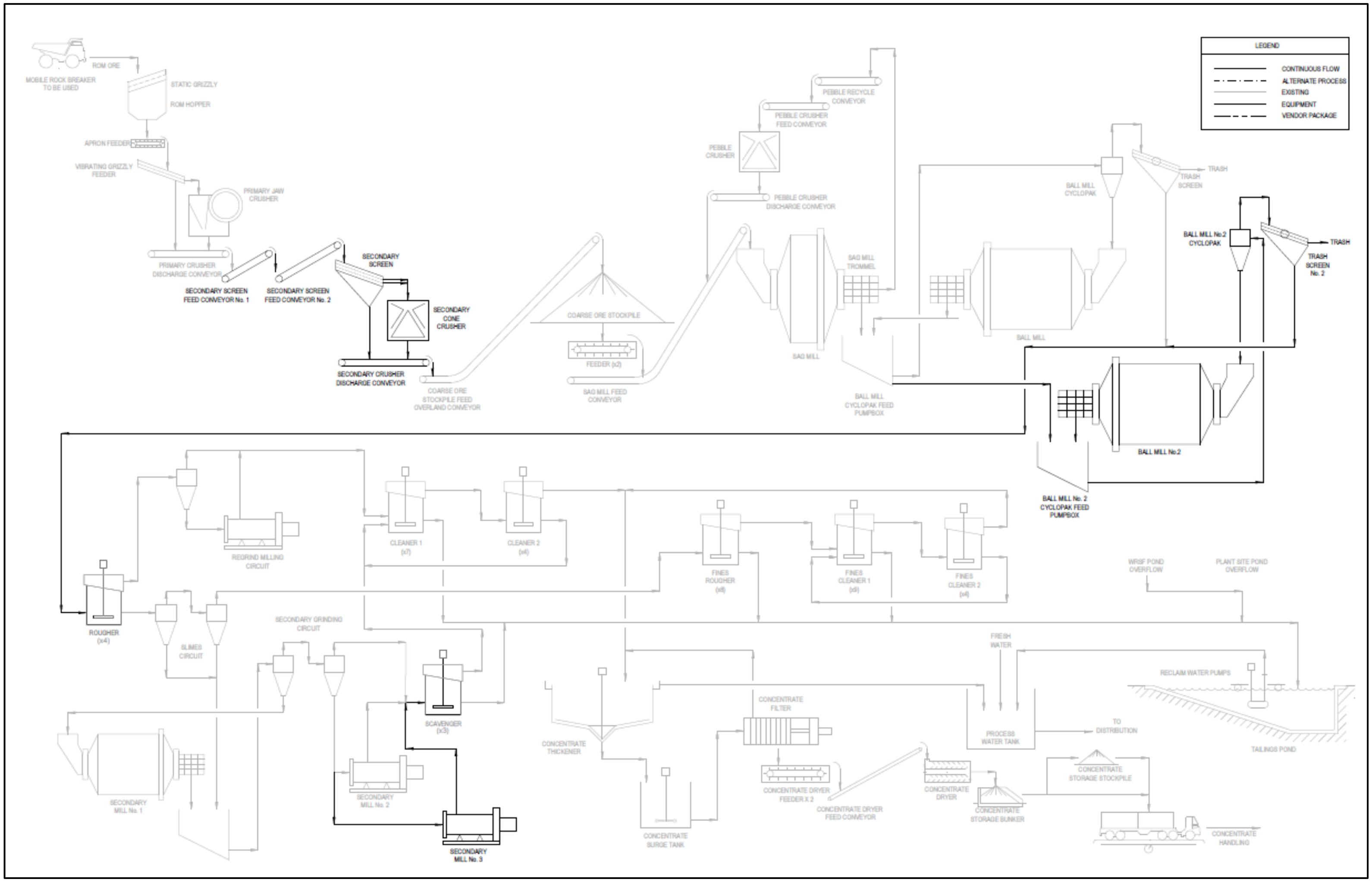
An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1 for the initial plant (Years 1-5), and 17.2 for the expansion (Years 6+). Plans and sections of the proposed process plant are provided in Figure 17-3 to Figure 17-9.

Figure 17-1: Simplified Process Flowsheet (Years 1-5)



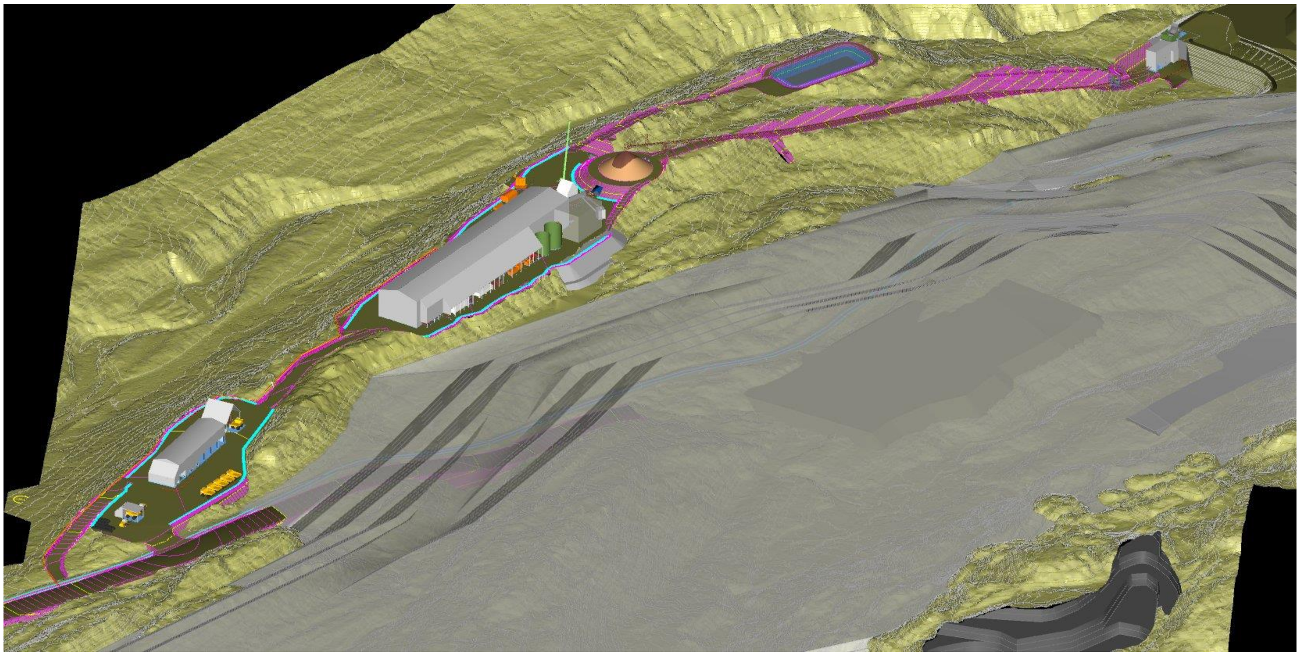
Note: Figure prepared by Ausenco, 2022.

Figure 17-2: Simplified Process Flowsheet (Years 6+)



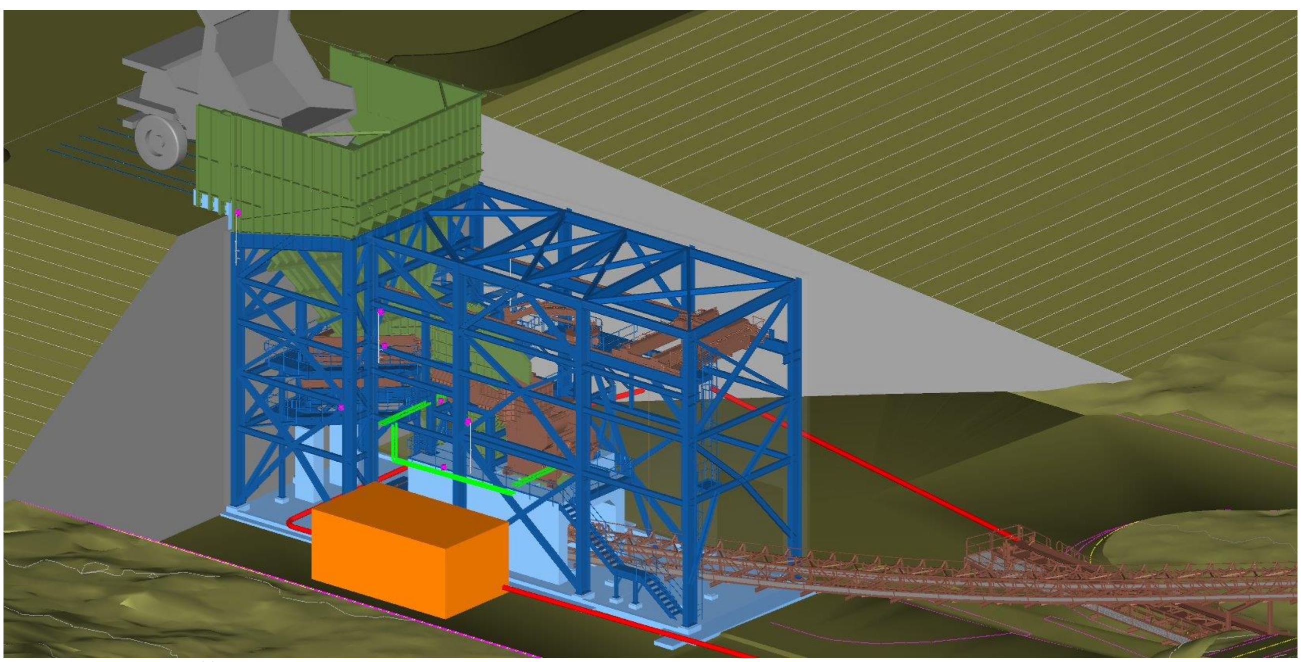
Note: Figure prepared by Ausenco, 2022.

Figure 17-3: Overall Process Plant Layout



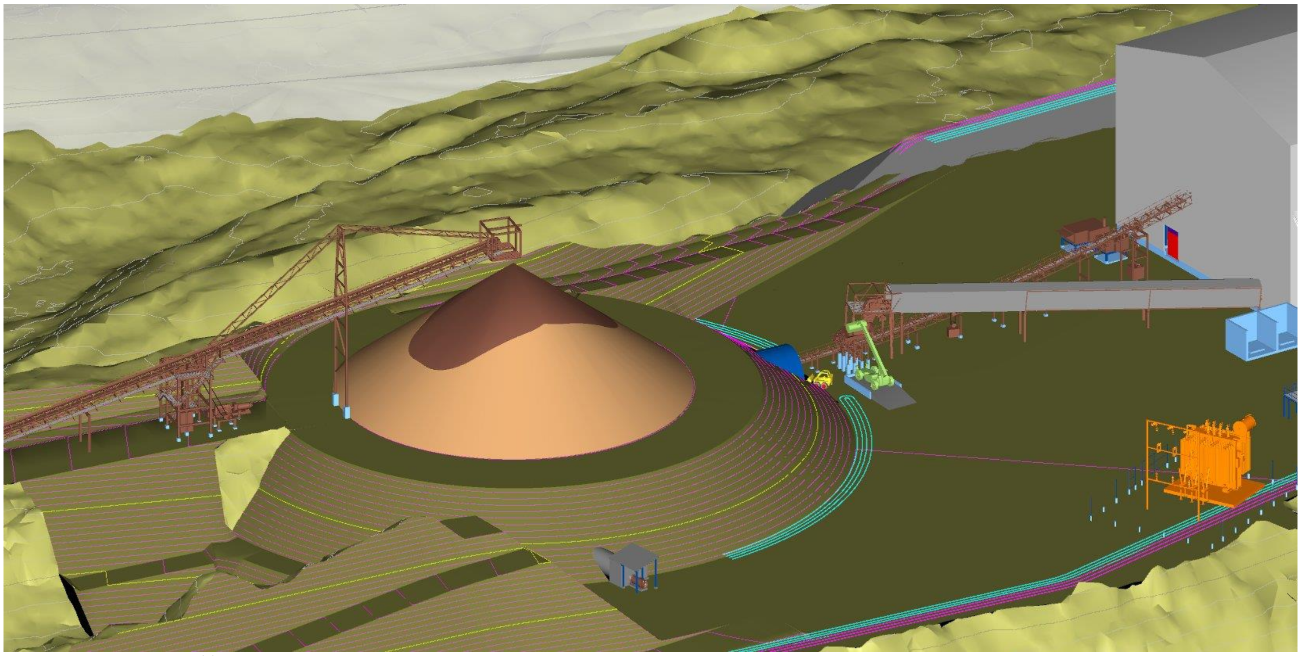
Note: Figure prepared by Ausenco, 2022.

Figure 17-4: Crushing Area Section



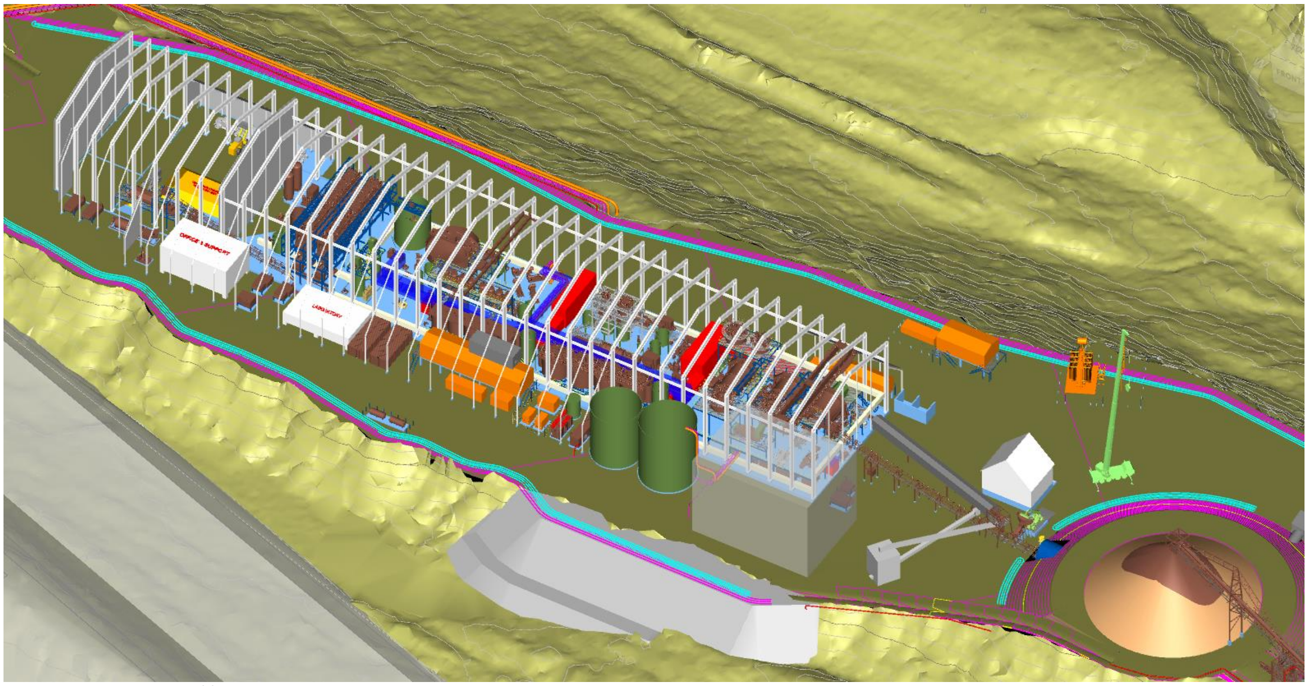
Note: Figure prepared by Ausenco, 2022.

Figure 17-5: Stockpile Area Section



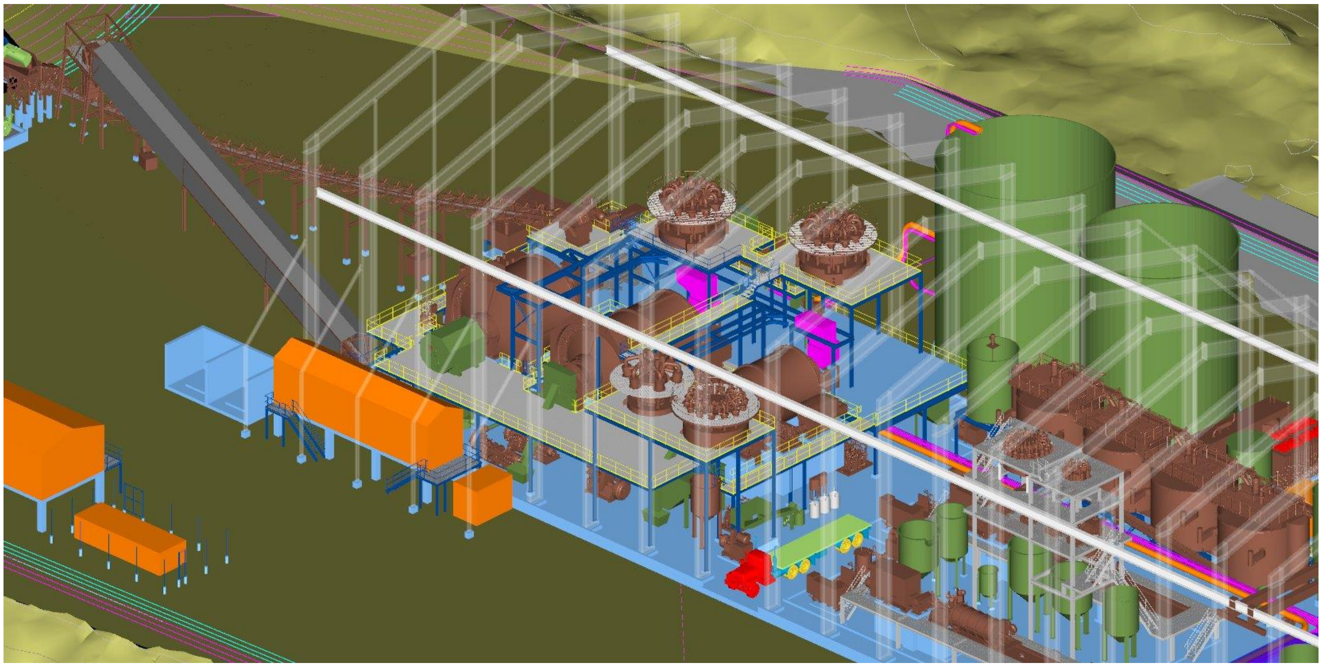
Note: Figure prepared by Ausenco, 2022.

Figure 17-6: Process Plant Area



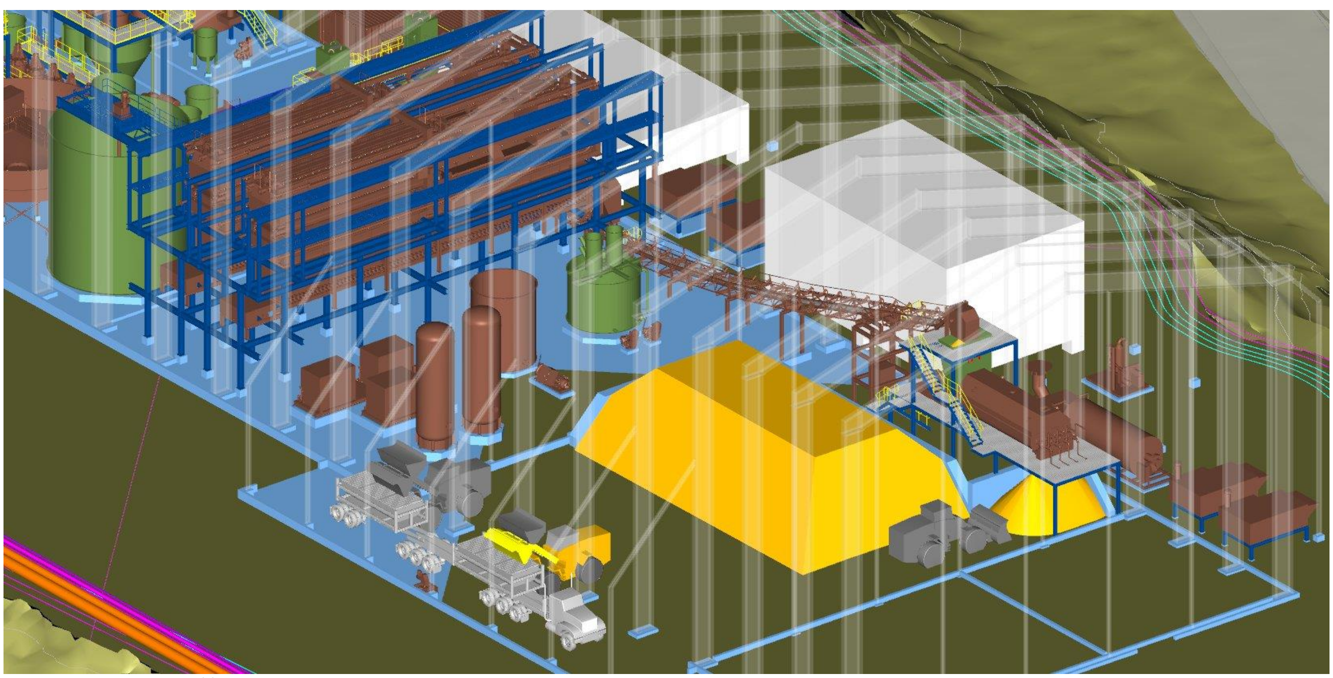
Note: Figure prepared by Ausenco, 2022.

Figure 17-7: Grinding Area Section



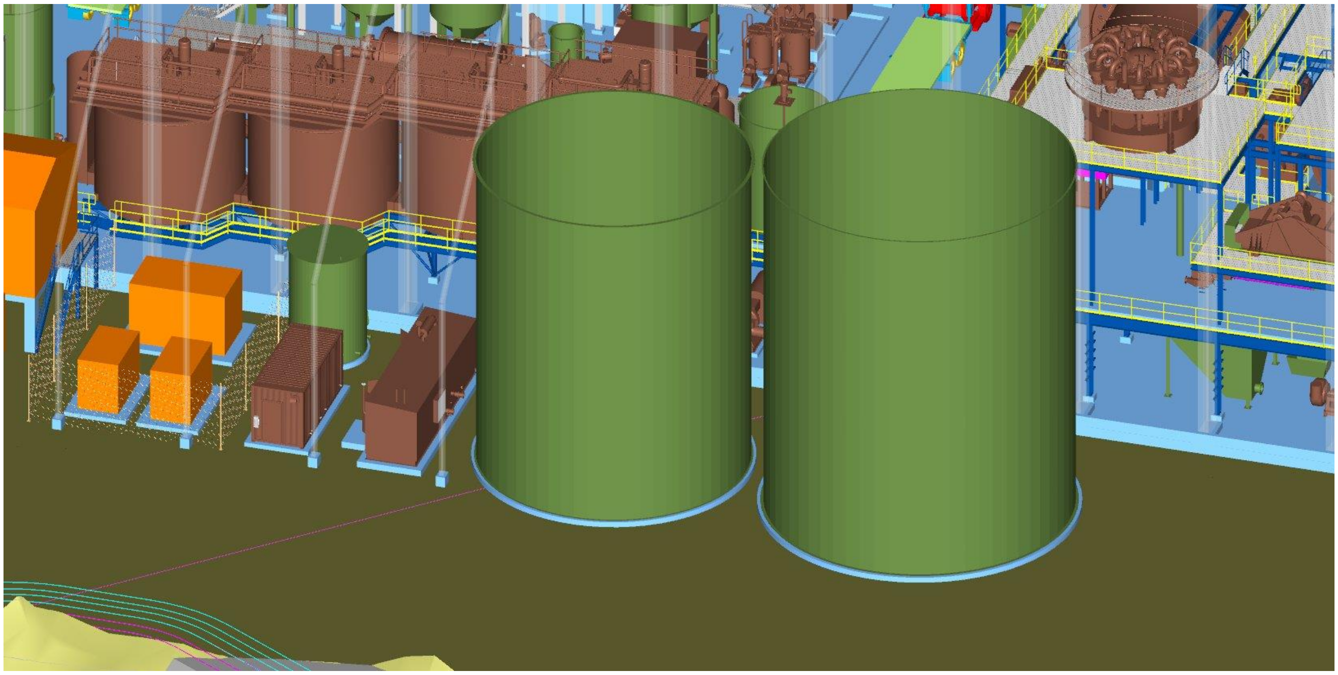
Note: Figure prepared by Ausenco, 2022.

Figure 17-8: Filtration Area Section



Note: Figure prepared by Ausenco, 2022.

Figure 17-9: Process Plant Services Area Section



Note: Figure prepared by Ausenco, 2022.

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	C130 Jaw crusher (or equivalent)
Secondary Crushing	Secondary Crusher	-	Installed for Year 6
		Model/Type	HP400 Cone Crusher
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	4.4 MW
	Primary Ball Mill No. 1	No. of mills	1
		Size	6.1 m diameter (inside shell) x 8.53 m (EGL)
		Mill motor power	5.8 MW
	Primary Ball Mill No. 2 (expansion)	-	Installed for Year 6
		No of mills	1
		Size	4.88 m diameter (inside shell) x 6.72 m (EGL)
		Mill motor power	2.6MW
	Pebble Crusher	-	Installed for Year 4

Area	Type		Specifications
		Model/type	TC51SH Short Head Cone Crusher (or equivalent)
Regrinding/classification	Regrind Mill	Type	IsaMill M7,500
		No. of mills	1
		Mill motor power	2.2 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 Mill No. 1	Type	Ball mill
		No. of mills	1
		Size	5.2 m diameter (inside shell) x 8.4 m (EGL)
		Mill motor power	3.4 MW
	Secondary Mill No. 2	Type	IsaMill M15,000
		No. of mills	1
		Mill motor power	3.8 MW
	Secondary Mill No. 3 (expansion)	-	Installed for Year 6
		Type	IsaMill M5,000
		No. of mills	1
		Mill motor power	1.1 MW
Flotation	Rougher	Type	Conventional Tank Cells
		No. of cells	4
		Size (diameter by height or D x H)	6.9 m x 6.0 m
	Scavenger	Type	Conventional Tank Cells
		No. of cells	3
		Size (D x H)	6.1 m x 5.1 m
	Cleaner 1	Type	Conventional Tank Cells
		No. of cells	4
		Size (D x H)	3.7 m x 4.3 m
	Cleaner 2	Type	Conventional Tank Cells
		No. of cells	2
		Size (D x H)	3.7 m x 4.3 m
	Fines rougher	Type	Conventional Tank Cells
		No. of cells	4
		Size (D x H)	6.1 m x 5.1 m
	Fines cleaner 1	Type	Conventional Tank Cells
		No. of cells	4
		Size (D x H)	2.1 m x 2.5 m
	Fines cleaner 2	Type	Conventional Tank Cells

Area	Type		Specifications
		No. of cells	3
		Size (D x H)	2.1 m x 2.5 m
Concentrate dewatering	Concentrate thickener	Type	High-rate
		Size	13 m diameter
	Concentrate filter	Model/Type	Vertical plate, MCDTC-H2100 x 108/120 Verticle plate, MCDTC-H2100 120
		No. of filters	2
		Size	2,100 x 2,100 mm plates 30 mm chamber depth
		Filtration area	864 m^2
	Concentrate Dryer	Type	Holo-flite
		No. of dryers	1
		Size (L x W x H)	15 m x 5 m x 3 m

17.4 Process Description

17.4.1 Crushing and Stockpile

The crushing facility will initially be a single-stage crushing circuit that will process the run-of-mine (ROM) ore at a nominal processing rate of 489 t/h, at 70% availability, in Year 1 to Year 5. Following the addition of a secondary crusher the nominal processing rate will be 603 t/h at 70% availability for Year 6 onwards. The major equipment and facilities at the ROM receiving and crushing areas will include:

- stationary ROM bin grizzly
- ROM surge bin
- primary crusher apron feeder
- vibrating grizzly
- primary jaw crusher
- secondary screen and cone crusher installed for year 6+ expanded throughput
- coarse ore stockpile (uncovered)
- 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 an apron feeder and discharged onto a vibrating grizzly where the coarse oversize will report directly into a single jaw crusher while the fines drop to the primary crusher discharge conveyor. The feed material will be crushed and will discharge from the crusher onto the primary crusher discharge conveyor. Initially this will transfer to the coarse ore stockpile feed overland conveyor to transport feed material to the coarse ore stockpile.

Following the installation of the secondary screen and crusher for the throughput expansion, the primary crusher discharge conveyor will transfer material to the secondary screen feed conveyor and onto the secondary screen. Oversized material will be crushed in the secondary crusher and report to the secondary crusher discharge conveyor while undersize material will bypass the crusher and report directly to the secondary crusher discharge conveyor. The secondary crusher discharge conveyor will transfer material to the coarse ore stockpile feed overland conveyor to transport feed material to the coarse ore stockpile.

The route length and lift (725 m with 150 m lift) from the primary crusher to the coarse ore stockpile necessitates the need for 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 initially provide approximately 8 hours of live capacity and approximately 6.5 hrs after the expansion for year 6.

Coarse ore from the stockpile will be reclaimed by two apron feeders, each capable of 100% of the initial plant feed, and discharge 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.

17.4.2 Grinding and Classification

The primary grinding circuit for year 1 to year 3 will consist of only a SAG mill and ball mill in a closed circuit with classifying cyclones. A pebble crusher will be installed and will start operating at the beginning of year 4. The primary grinding circuit for year 4 and year 5 will consist of a SAG mill, pebble crusher and ball mill in a closed circuit with classifying cyclones. A second ball mill will be installed in a new building and start operating at the beginning of year 6.

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 (P80) of 100 µm. The SAG mill will be driven by a single 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 fixed speed WRIM with an LRS.

Steel balls will be added into the SAG mill via a ball loader onto the SAG mill feed conveyor and into the 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 3, screen oversize will be returned to the SAG mill feed conveyor via pebble recycle conveyor and recycled back to the SAG mill. 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 approximately 30% solids (by weight) comprising product sized particles, whilst the cyclone underflow fraction of approximately 72% solids (by weight) will report to the ball mill(s).

After the pebble crusher is installed, for year 4, the trommel screen oversize will be transferred to the pebble crusher via a pebble recycle conveyor to the pebble crusher feed conveyor and the crusher product returned to the SAG mill feed conveyor.

After the expansion a portion of the flow into the cyclopak feed pumpbox #1 will be transferred to the ball mill #2 cyclopak feed pumpbox in the new building to feed the new ball mill.

Cyclone underflow will be ground in the ball mill(s). Ball mill discharge will flow through the ball mill discharge trunnion magnet and remove any broken mill balls, which will then be discharged to a concrete ball mill scats bunker. After passing through the trunnion magnet, slurry will discharge into the respective cyclone feed pump box.

The cyclone overflow(s) will report to a trash screen 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.

17.4.3 Flotation and Regrinding

The flotation area will consist of a primary rougher and cleaner circuit, a rougher scavenger circuit, and a fines rougher and cleaner circuit. The area will also include regrinding of rougher concentrate prior to cleaning, rougher tailings classification for slimes (fines) separation prior to secondary grinding and scavenger flotation. Conventional tank cells will be used in all flotation circuits.

Process water will be used for all launder sprays.

Flotation reagents used in the flotation circuit will include potassium amyl xanthate (PAX; collector), copper sulphate (promoter), and methyl isobutyl carbinol (MIBC; frother).

The regrind mill will target a discharge size P80 of 15 µm. The slimes classification overflow will target a size P80 of 20 µm, and the secondary regrinding circuit will target a final discharge P80 of 30 µm.

Primary cyclone overflow will gravitate through a trash screen into the flotation feed pumpbox prior to being pumped to the rougher conditioning tank and a bank of rougher flotation cells. Concentrate from the rougher cells will be pumped through the rougher concentrate regrind mill cyclone cluster. The cyclone underflow will be pumped to the regrind mill and ground to a P80 of 15 µm. The regrind mill discharge and regrind mill cyclone overflow are combined with cleaner 2 tailings, scavenger concentrate and flotation reagents in the regrind mill pumpbox then will be pumped to the cleaner 1 circuit. The flow to the roughers after the expansion remains within the initial design criteria so no additional equipment is required in the rougher circuit.

Cleaner 1 concentrate gravitates to the cleaner 2 feed tank and cleaner 1 tailings will gravitate through a sampler into the cleaner 1 tailings pumpbox and will be pumped to the TMSF. Cleaner 2 concentrate will gravitate to the cleaner 2 concentrate pumpbox and will be pumped to the concentrate thickener while cleaner 2 tailings will be pumped backed to the regrind mill discharge pumpbox to be recycled through the cleaner 1 flotation train. The mass of concentrate after the expansion remains within the initial design criteria so no additional equipment is required in the cleaner circuit.

Rougher flotation tailings will be pumped to the slimes classification circuit, where two stages of cyclones will produce an overflow P80 of 20 µm that will report to the fines roughers flotation circuit. Underflow streams from both stages of cyclones will be combined in a pumpbox and will be fed to the secondary grinding mill #1 cyclones. Cylcone underflow will gravitate back to the closed circuit secondary ball mill while the cyclone overflow will gravitate to the secondary grinding mill #2 cyclone feed pumpbox. The secondary mill #2 cyclones will dewater the secondary mill #1 cyclone overflow to produce a higher percent solids feed for the secondary mill #2 (IsaMill). Secondary mill #2 cyclone overflow and secondary mill #2 discharge will be combined and pumped to the scavenger conditioning tank. The secondary grinding circuit will produce a final discharge size P80 of 30 µm.

For the expansion, extra cyclones will be added into the secondary grinding cyclopaks, filling the extra spots included initially. A portion of the dewatering cyclone underflow will be pumped to the new IsaMill located in the expansion building. The product of both secondary IsaMills will be combined and pumped to the scavenager circuit.

Flotation reagents are added to the secondary grinding product in the scavenger flotation conditioning tank which overflows to the scavenger flotation cells. Scavenger concentrate will be pumped to the regrind mill discharge pumpbox. Scavenger tailings will gravitate to the fines rougher tailings pumpbox and will be pumped to the cleaner 1 (final) tailings pumpbox.

Stage 2 de-slime cyclone overflow will gravitate to the de-slime product pumpbox, be combined with flotation reagents and will be pumped to fines rougher flotation train. Fines rougher concentrate will gravitate to the fines cleaner 1 feed tank, combine with flotation reagents and gravitate into the fines cleaner 1 flotation train. Fines rougher tailings will gravitate to the fines rougher tailings pumpbox and will be pumped to the cleaner 1 (final) tailings pumpbox. The mass of slimes after the expansion remains within the initial design criteria so no additional equipment is required in the fines flotation circuit.

Fines cleaner 1 concentrate will gravitate to a pumpbox, combine with flotation reagents and will be pumped into the fines cleaner 2 flotation train. Fines cleaner 1 tailings will gravitate to the fines rougher tailings pumpbox and pumped to the cleaner 1 (final) tailings pumpbox.

Fines cleaner 2 concentrate will gravitate to the cleaner 2 concentrate pumpbox, combine with cleaner 2 concentrate and pumped to the concentrate thickener. Fines cleaner 2 tailings will gravitate to the fines cleaner 2 tailings pumpbox and pumped to the cleaner 1 feed tank.

17.4.4 Concentrate Dewatering

Concentrate from cleaner 2 circuit will be combined with concentrate from fines cleaner 2 circuit in a pumpbox and will be pumped to the concentrate static trash screen with any oversize foreign material reporting to a bin and the screen undersize gravitating to a thickener feed de-aeration tank and thickener. The final concentrate will be thickened in a high-rate thickener. The mass of concentrate after the expansion remains within the initial design criteria so no additional equipment is required.

Flocculant will be added to the thickener feed stream to enhance settling. The concentrate thickener overflow will report to the concentrate thickener overflow pumpbox and pumped to the process water tank. Gold concentrate solids settle for collection at the underflow cone at a density ranging from 40 to 55% w/w solids (by weight) depending on ore feed. The thickener underflow stream is pumped to the concentrate filter feed tank.

The filter feed tank will provide 12 hours of surge capacity to allow filter maintenance to be conducted without affecting mill throughput. The filter feed will be pumped to one of two pressure filters to produce a filter cake of approximately 15% w/w moisture. The filter cake from each filter will be discharged and drop onto their respective feeders which transfer the cake to the concentrate dryer conveyer that will feed a Holo-Flite dryer. The dryer is required to meet the target moisture levels of <13.5% w/w to transport the concentrate to the designated shipping facility. The dried concentrate will be discharged and drop from the dryer into a bunker. A front-end loader will load concentrate into a truck or move it to the storage area within the building. The normally empty storage area will have a nominal storage capacity of 4 days or 2,407 t to avoid process plant shut down due to disruptions with concentrate trucking.

Process water is used for the filter cloth washing and to flush the filter manifolds. Filtrate, cloth wash and manifold flushing water will report to concentrate filtrate tank and then return to the concentrate thickener. Dedicated filtrate separators remove excess air from the filtrate streams.

Two dedicated air compressors supply high-pressure air for the concentrate filters. One compressor line supplies a dedicated air receiver for membrane squeeze pressing, while the other line supplies a dedicated air receiver for air drying.

17.4.5 Tailings Disposal

Scavenger tailings, cleaner 1 tailings, fines rougher tailings, and fines cleaner 1 tailings will be combined in a pumpbox with any excess contact water, not being used for process water, prior to being pumped to the TMSF. Flow of tailings after the expansion is within the initial design criteria so no additional equipment for tailings disposal.

In-line flocculant is added to the tailing stream to enhance settling and minimize turbidity in TMSF.

17.4.6 Reagents and Consumables

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. Consumption will be approximately 2,100 t/y.

- Promoter: copper sulphate pentahydrate ($CuSO4$) 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. $CuSO4$ will be stored in a separate self-contained area within the process plant and delivered to the $CuSO4$ 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 $CuSO4$ distribution tank prior to distributing to required addition points by dedicated metering pumps. Consumption will be approximately 1,800 t/y.

- Frother: methyl isobutyl carbinol (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. Consumption will be approximately 450 t/y.

- Flocculant: MF336 (or similar) is used as a settling aid in the concentrate thickener. A flocculant mixing, storage and dosing system located in a separate self-contained area within the process plant and delivered to the flocculant mixing area. 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/v. 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/v and fed to the concentrate thickener. Flocculant solution will be made-up and transferred daily with a tank truck from the flocculant mixing tank to the TMSF flocculant storage tank for addition to the tailings line via dedicated metering pump. Consumption will be approximately 10.100 t/y split evenly to the concentrate and tailings thickening.

Consumables will include:

- Crushing liners and wear parts
  - The consumption rates for crusher liners and grinding mill liners for the different comminution equipment were obtained from the equipment suppliers and from experience with similar operations as shown in Table 17-5.

Table 17-6: Consumption Rates for Crusher Liners and Grinding Mill Liners

Item	Units	Consumption (Year 1–3)	Consumption (Year 4–5)	Consumption (Year 6+)
Jaw Crusher Liners	Sets/yr	6	6	6
Sag Mill liners	Sets/yr	1	1	1
Primary Ball mill liners	Sets/yr	1	1	1
Pebble Crusher liners	Sets/yr		5	2
Reline Contractors	Reline/yr	1	1	1
Regrind Mill	#/yr	1	1	1
Secondary Ball Mill liners	#/yr	1	1	1
Secondary Isa Mill Liners – 1	#/yr	1	1	1
Secondary Isa Mill Liners – 2	#/yr			1

- Grinding media
  - The grinding mills will need a regular addition of balls to replace the worn media and exercise proper grinding action on the material. The media consumption (as shown in Figure 17-7) has been estimated from the power input into the material based on steel consumption observed in similar operations and based upon industry-standard calculation methods. These calculations are based upon expected mill operating conditions (loading, ball charge, speed, etc.) as well as abrasion indices obtained via testwork.

Table 17-7: Grinding Media Consumption

Item	Units	Consumption (Year 1–3)	Consumption (Year 4–5)	Consumption (Year 6+)
Sag Mill Grinding Media	t/yr	1,143	1,265	1,117
Ball Mill grinding Media	t/y	2,369	2,460	3,304
Regrind Ceramic Grinding media	t/y	77	77	85
Secondary Ball Mill Grinding Media	t/y	1,308	1,308	1,269
Isamill Ceramic Grinding Media – 1	t/y	149	149	156
Isamill Ceramic Grinding Media – 2	t/y			33

17.4.7 Services

17.4.7.1 Air Services

Two rotary screw air compressors 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 process plant. The expansion concentrate mass, and therefor air flow, is within the initial design parameters so no additional equipment is required.

Two dedicated blowers, providing forced air to the rougher and scavenger flotation cells, the expansion air flow is within the initial design parameters so no additional equipment is required.

17.4.7.2 Water Services

17.4.7.2.1 Fresh Water

Fresh water will be sourced from borehole wells and pumped to the fresh water tank located outside of the main building. Fresh water tank provides 10 hours of capacity, and fresh water 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 process plant by two freshwater pumps in a duty/standby configuration. Consumption of freshwater will be approximately 2,330 m^3/d for years 1-5 and 2,820 m^3/d for years 6+.

17.4.7.2.2 Potable Water

Potable water will be sourced from the fresh water tank and treated in the potable water treatment plant. The treated water will be stored in a potable water storage tank, including 44 hours of storage tank live capacity at the design rate. Distribution of potable water is achieved by two potable water pumps in a duty/standby configuration.

17.4.7.2.3 Gland Water

Gland water will be supplied from the freshwater tank and distributed to the process 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 the filter feed pump.

17.4.7.2.4 Process Water

Process water will consist predominantly of mine dewatering and contact water and concentrate thickener overflow but will be supplemented with TMSF reclaim water as required. Process water will be stored in a process water storage tank with 1.2 h of storage live capacity at average usage and distributed by two process water pumps in a duty/standby configuration.

17.4.7.3 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 flotation feed, final tailings, and final concentrate. 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.

17.5 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 the operation of the process 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 using 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 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 Overall Site

The overall site plan (see Figure 18-1) shows the major project facilities, including the open pit mines, Tom MacKay Storage Facility (TMSF), waste rock storage facility (WRSF), water management ponds, process plant, mine services, historical site and main access road. Access to the facility is from the northern side of the property from the existing Eskay Creek Mine Road. Access to the process plant will be via the existing road to the historical Eskay Creek Site.

The site will not be fenced however, there will be gatehouse to clearly delineate the mining and processing areas to deter access by unauthorised people. The process plant is located south of Tom MacKay Creek, to the west of the open pits. Tailings will be disposed in the existing Tom MacKay Storage Facility west of the process plant and waste rock will be stacked in a new WRSF to the south of the process plant.

Site selection and location took into consideration the following factors:

- Maximising the use of existing infrastructure such as TMSF, historical buildings, site access road;
- careful consideration of surface water management to accommodate the high rainfall and flow during freshet to minimise impact on waterways;
- minimising earthworks (particularly in areas that are PAG) considering the mountainous terrain and management of heavy snow fall during winter;
- locate the ROM pad as close as possible to the open pits, to minimise haul distance;
- ensure the location of the process plant and mining truck area are outside the flyrock exclusion zone from the resource;
- utilise the natural terrain for the ROM pad as much as possible;
- separate heavy mine vehicle traffic from non-mining, light-vehicle traffic;
- locate the process plant in an area safe from flooding;
- locate the heavy equipment foundation on competent bedrock and utilise rock anchors for foundations design;
- place administration and processing plant staff offices close together to limit walking distances between them; and
- locate the ready line close to the truckshop, mining admin/office area and change house.

The Eskay Creek 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: to support the facilities and operations including gate house, administration building, laboratory, plant workshop, and plant offices.

- Accommodations camp: a temporary 210 person camp for construction to be utilized together with the existing 227 bed historical camp. A permanent operations camp will be constructed near the process plant area, which will comprise of a new 180 person camp, together with 200 person existing modules relocated to this area from the historical camp. These camps will use common facilities.

- Power supply: The power supply for the Project will be provided from the 287kV Volcano Creek interconnection point, where a new 287/69kV Substation will be installed and a 17km, 69kV overhead power line will be routed to the main site substation, which will be stepped down to 13.8kV for primary power distribution around the site. Standby diesel generator set(s) will be installed for powering the essential loads within the process plant in the event of a utility power failure.

- 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.

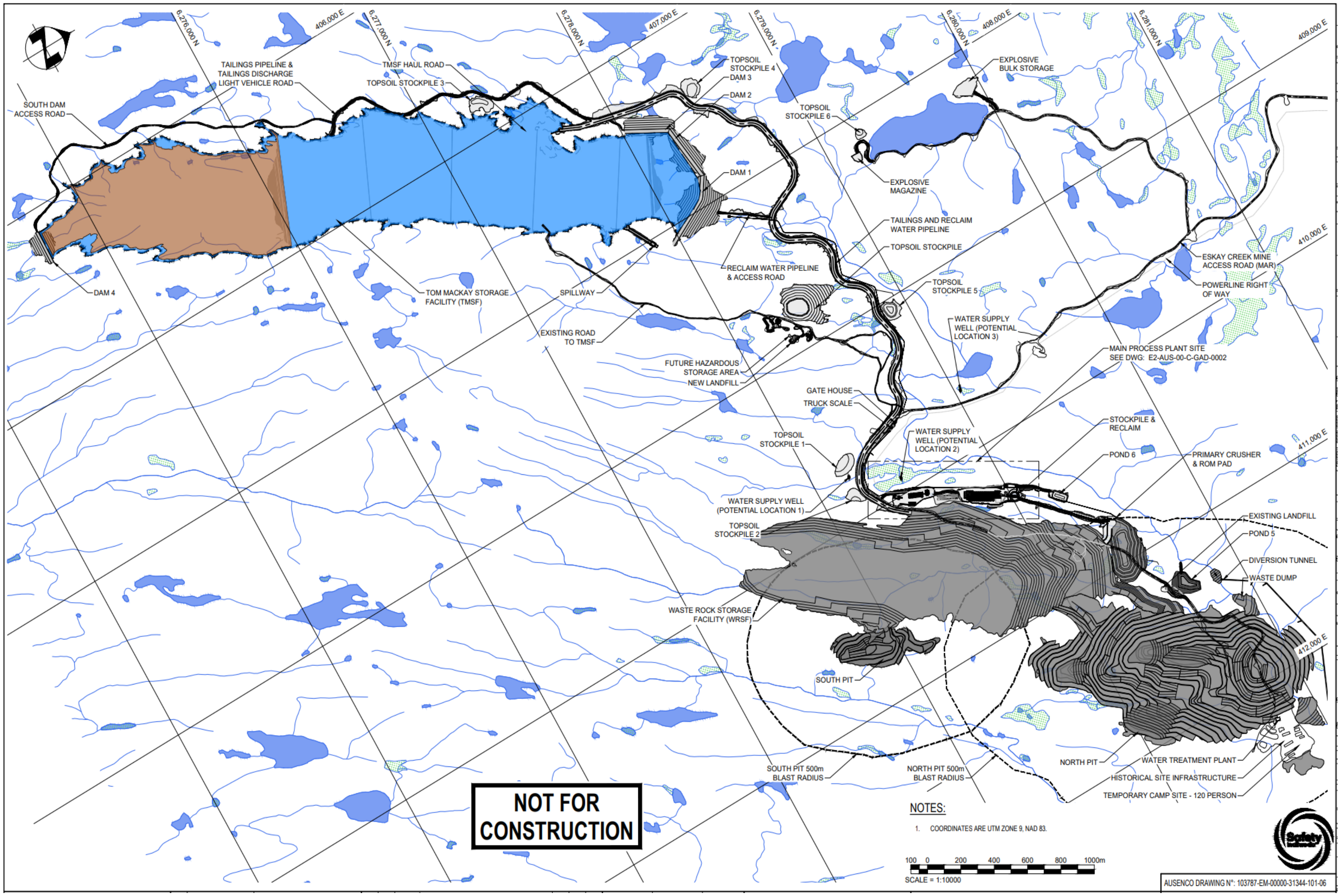
- Communications: Fiber optic communication network within the plant for process control and integration of the control systems.

- 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 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 .

Figure 18-1: Project Proposed Layout Plan



Note: Figure prepared by Ausenco, 2022

18.2 Roads

18.2.1 Access to Site

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 137 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 upgrades to two bridges to accommodate 72,300 kg gross vehicle weight (GVW) trucks will be completed ahead of the delivery of heavy equipment.

The access road is currently in good condition and is maintained on a continuous basis and is providing the main access to existing facilities at camp Km57 and Km 59 (Historical Camp). During construction, this road will be locally re-routed in some limited areas between the future gate-house and historical camp, to accommodate tie-ins to newly constructed roads, or expanded footprint of future infrastructure, however access will be continuously maintained throughout the construction to facilitate optimal utilisation of the existing facilities.

In early stages of operation, and as mining activities develop the extent of the Pit and WRSF, this road will terminate at the new process plant access tie-in, and the remaining length of this road will be decommissioned and replaced with haul roads connecting the Mining Infrastructure Area to WRSF, Stockpiles, and the Open Pit

18.2.2 Plant Site Roads

The roads within the process plant area will be generally 6 m wide, integrated with process plant pad earthworks, and designed with adequate drainage. The roads will allow access between the administration building, warehouses, mill building, crushing buildings, stockpile, mining truck shop, and top of ROM Pad.

18.2.3 TMSF Haul Road

A 5.1 km haul road will be constructed to connect the Plant Site area, to the TMSF area. This haul road will be two 11.0 m wide lanes, as well as provision of required shoulders, road-side collection ditches, and safety berms, as well as a pipeline corridor to accommodate the tailings slurry pipeline, as well as the reclaim water pipeline, that would be running alongside this road, resulting in a 32 m wide prism for a typical section of this haul road.

18.2.4 Haul Road to access Bulk Sample pit and Nag Quarries

Granular fill material for road base and sub-base construction and upgrade will be sourced from permitted borrow pits and quarries located near the identified bulk sample pit. A 1,250 m haul road will be constructed to provide access to the bulk sample starter pit area (Technical Sample Pit), as well as the two quarries identified as source of borrow material. This haul road will tie-in to the TMSF Haul Road, and will consist of two 11.0 m wide lanes, as well as provision of required shoulders, road-side collection ditches, and safety berms, resulting in an average 25m wide prism for a typical section of this haul road.

18.3 Concentrate Transportation

Concentrate will be stored at site in the concentrate load out facility awaiting highway haul truck load out. From this facility, the concentrate will be loaded using front-end loaders into highway haul trucks (72,300 kg GVW) up to 49 t concentrate per truck (24.5 t per tandem dump trailer). The concentrate will be trucked using the main site access road and Highway 37 under a "bulk haul" permit from the Province of BC Ministry of Highways to move concentrate from the mine approximately 250 km to Stewart Bulk Terminals (SBT). SBT is a multi-commodity port facility with up to 16,000t storage for Skeena's gold concentrate in a dedicated storage building with existing conveying load out infrastructure. Concentrate will be loaded onto bulk carrier ships at SBT via its existing ship loading infrastructure.

Concentrate volumes are presented in Table 19-3.

Table 18-1: 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)	2.73	3.0	3.0	3.0	3.7	3.7	3.7	3.7	3.7
Au (g/t)	4.47	3.28	4.21	4.12	4.26	2.50	2.45	1.72	1.12
Conc t (kt/a)	179	192	234	233	234	215	249	290	176

The estimated transportation cost from Eskay Creek mine to Asian smelters and is C$140 / t concentrate and is based on:

- Truck transportation cost from site to SBT: C$64.79/t, based on quotes from local transportation vendors
- Terminal handling cost at SBT: C$ 20/t consolidated handling cost for the receiving, storage and outloading of cargo, with no separate building leasing costs for the proposed 16,000t storage building. Annual cost adjustment of this handling charge is expected to be in accordance with the Consumer Price Index (CPI), however may depend on the multi-year contract in place.
- Ocean freight cost: C$55/t for bulk careers shipping bulk concentrate from Stewart Bulk Terminal to south-east Asia.

Construction materials and mine consumables sourced internationally will also 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.

18.4 Power Supply

18.4.1 Electrical Power Source

BC Hydro will supply power to the Eskay Creek Project where the system supply point will be the existing Volcano Creek interconnection point, which is located approximately 17 km to the northwest of the Eskay Creek Project.

To facilitate the connection, the following infrastructure will be required:

- Upgrade of 287 kV Volcano Creek interconnection point and construction of a new 287/69 kV substation to support the addition of electrical, protection and control, and communications equipment required to provide power to the Eskay Creek site;

- Construction of a new 69kV indoor substation, 69 kV/13.8 kV transformer and a 13.8kV switchgear (Eskay Creek Substation) adjacent to the new Eskay Creek process plant
- Construction of a 17 km 69 kV wood pole transmission line to connect the Volcano Creek interconnection point to the new Eskay Creek Substation
- Communications equipment will also be installed at the Volcano Creek interconnection substation and at Eskay Creek Substation for remote monitoring and protection.
- The Eskay Creek Project has the following electrical load requirements:
- Initial operation: Initial start-up requirement between year 1 to 5 inclusive – 27.1 MW
- Expansion: Full load requirement in year 6 to end of life – 31.2 MW

18.4.2 Eskay Creek Substation

The main substation (Eskay Creek) is located near the process plant. This terminal substation will be with 100% redundancy in transformer capacity. Two 24/32 MVA oil-filled with forced air-cooled type substation transformers are proposed to be installed to carry the maximum power required by the site. This includes future growth and redundancy in the event a single transformer is temporarily out of service.

18.4.3 Electrical Distribution

The plant electrical system is based on 13.8 kV distribution. The 69 kV feed from BC Hydro will be stepped down to 13.8 kV at the Eskay Creek Substation and will supply the plant main 13.8 kV switchgear housed in the main process plant electrical room.

The larger variable frequency drives (VFDs) will have 13.8 kV input, fed by plant main 13.8 kV switchgear. Step down transformers will be provided at the electrical rooms to feed 4160 V and 600 V loads, which will be fed from the plant main 13.8 kV switchgear. Electrical rooms will be provided at the following locations:

- process plant main
- primary crusher area
- flotation / filtration / drying

The main process plant electrical room will house the 13.8 kV switchgear. The other electrical rooms will consist of 13.8 kV/600 V transformers close coupled to the 600 V motor control centres (MCCs), LV VFDs, LV soft starters, plant control system cabinets, lighting and services transformers, distribution boards, and uninterrupted power supply (UPS) power distribution.

Power to mine infrastructure areas such as the office buildings, gate house, water supply wells, ponds, reclaim barge will be provided through 13.8 kV overhead distribution lines with step-down transformers at each location.

To reduce installation time, the electrical rooms were considered prefabricated modular buildings, installed on structural framework 2 m above ground level for bottom entry of cables. The electrical rooms will be installed with HVAC units and suitably sealed to prevent ingress of dust. They will be in the process plant area and as close as possible to the main load points to minimise costs.

18.4.4 Power Reticulation

Overhead power lines of 13.8 kV will provide power to various remote facilities. Pole-mounted or pad-mounted transformers will step down the voltage at each location and supply the low voltage distribution system to respective facilities.

18.4.5 Standby / Emergency Power Supply

Three standby diesel generators in weatherproof enclosures will be provided to supply critical process loads and life safety systems. Each standby diesel generator is located close to the MCCs feeding the critical loads. The generators have been sized based on the assumption that in case of power failure, the power to the tank agitators and rougher flotation cells will be toggled between each of the agitators (i.e., keep two running for 10 minutes and cycle through each).

18.4.6 SAG & Ball Mill Drives

The SAG and ball mills are the largest electrical loads in the plant. Both motors are squirrel cage induction motors, with single VFD and bypass switchgear arrangement to minimise voltage drop impact on the utility supply system during motor start-up. The VFD will be used to start the ball mill and once the ball mill is running on fixed speed, the same VFD will be used to run the SAG mill at variable speed.

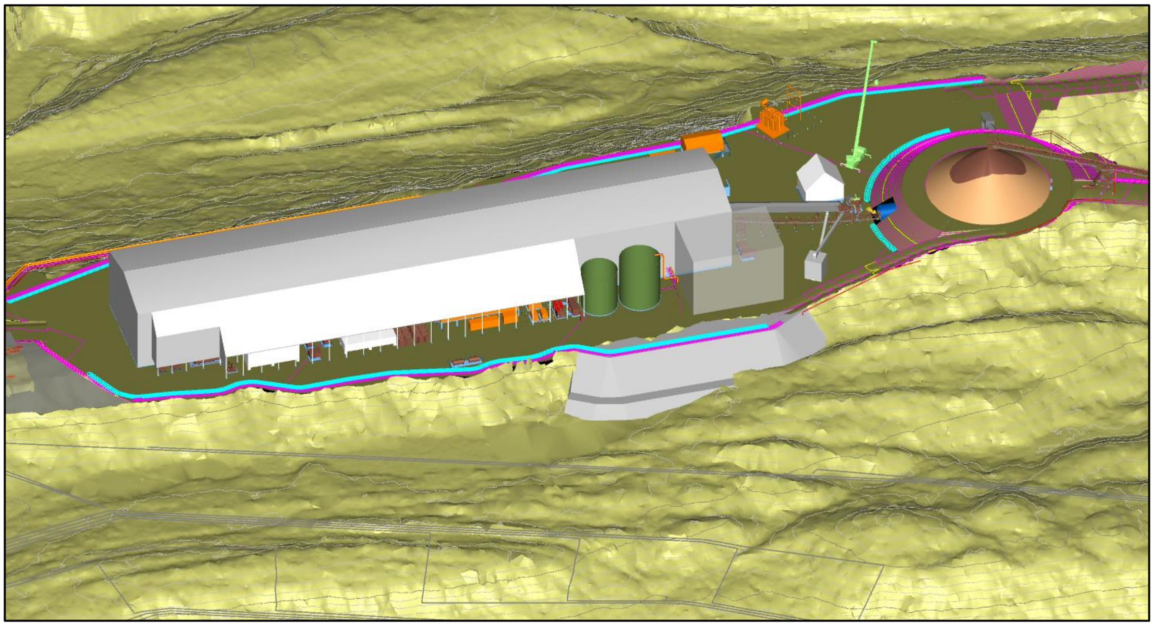
18.4.7 Expansion

Additional 13.8 kV power transformers and a new electrical room will be provided to support the plant expansion.

18.5 Support Buildings

Figure 18-2 shows a 3D model image of the process plant and process infrastructure for the initial operation, with the expansion represented in grey.

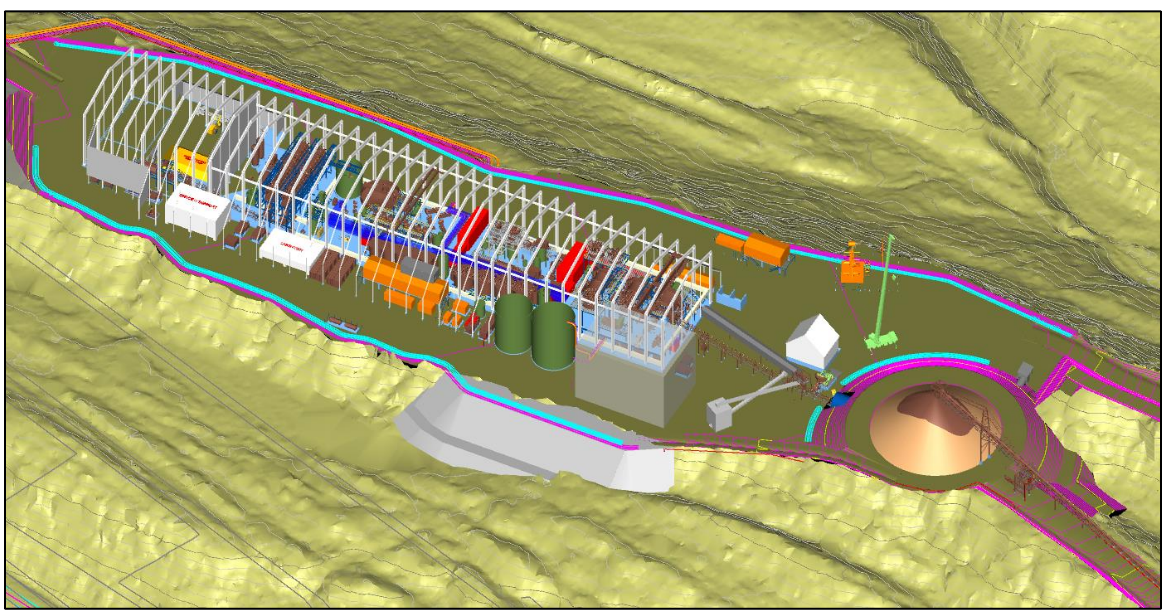
Figure 18-2: Process Plant Building – Looking North



Note: Figure prepared by Ausenco, 2022.

A 3D model shot of the ROM pad and crushing station (building cladding removed for clarity) is illustrated in Figure 18-3 where the expansion (secondary crushing) is represented in grey.

Figure 18-3: Process Plant Building (Cladding Removed for Clarity) – Looking West



Note: Figure prepared by Ausenco, 2022.

18.5.1 Process Plant Buildings

18.5.1.1 Crushing Plant Building

The primary crushing circuit will be in close proximity, and elevation, to the main pit exit. The location and design consider the 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 ROM bin. The crushing station will be mounted at an elevation of 919 masl and separated from the ROM pad by a 16-m-high mechanically stabilized earth wall. The crushing plant consists of one building located over the primary crusher, control room and rock breaker equipment, which is adjacent to the ROM pad. The building (29 m long by 9m wide) will be pre-engineered fully enclosed with metal cladding complete with HVAC and a 7.5 t overhead maintenance crane.

To facilitate the expansion in throughput in later years, space has been reserved adjacent to the primary crusher for the addition on the secondary crusher and associated equipment. This equipment will be housed within a pre-engineered building constructed as part of the expansion.

18.5.1.2 Process Plant Building

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 1012 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 36m wide and 210 m long with an eave height of 24 m. The building will be fully enclosed with metal cladding complete with HVAC. All buildings will be supported on reinforced concrete footings with concrete slabs and pedestals Process equipment within this area will be serviced by two overhead cranes, one of 50 t capacity and one of 45 t capacity. This building will contain three major areas:

- Process plant equipment including as SAG Mill, Ball Mill, Regrind Mills, mill liner handler, cyclones, flotation cells, grinding, flotation, screens, reagent mixing, tanks and pumps
- Concentrate drying, filtration and storage area
- Plant warehouse and workshop.

To facilitate the expansion in throughput in later years, space for a 36 m (long) by 20 m (wide) pre-engineered building has been reserved adjacent to the process plant building for housing the expansion equipment required. Piping and cable corridors would be routed from the main process plant building to this future building.

18.5.2 Mine Infrastructure Area

The truck wash building at the site will be a 25 m (wide) x 18 m (long) fabric building also located north of the ROM pad, and east of the truck shop. The building will be used for washing haul trucks and will be supported on a concrete foundation.

The mine infrastructure area (MIA) is located to the west of the process plant area and includes:

- Integrated truck shop, truck wash, warehouse, workshop, offices, lunchrooms and change rooms for mining operations
- Haul truck tire change and tire storage area with roof
- Fire truck and ambulance storage with pre-engineered building and heating
- Fuel storage and dispensing with roof

Heavy vehicle (including haul trucks) and light vehicles will have separate traffic flow to minimise heavy and light vehicle interaction. Haul truck and heavy vehicle access to the MIA will be via the haul road. Light vehicle access to the MIA will be via the main site access road, as well as from the process plant.

18.5.2.1 Truck Workshop and Offices

The truck workshop building, located to the west of the process plant), will be a 23 m (long) by 85 m (wide) pre-engineered building and will be supported on a concrete foundation. The ground floor will be used for vehicle maintenance and washdown, with upper levels of the building dedicated to the changerooms and offices including:

- two maintenance bays for 144-t haul trucks services by an overhead crane with 20 t;
- a truck wash bay complete with pressure washing equipment, access platforms and wash water collection system;
- tracked vehicle maintenance bay;
- light vehicle maintenance bay;
- lube/oil storage;
- mine offices (upper level – a total of 28 office spaces, combination of open plan workspaces and office areas, and 3 meeting rooms);
- showers, lockers and changerooms (upper level); and
- workshop and warehouse (total of 22 m x 11 m double height).

18.5.2.1 Fuel storage

The fuel station will consist of a 40 m (long) x 25 m (wide) open-air area including truck manoeuvring space. There will be a central area, with reinforced concrete containment. The fuel station will be located adjacent to the truck shop. The fuel station will service the on-site mine equipment and mobile fleet.

Diesel fuel storage and supply will be provided by a fuel supplier and will include a total volume of 200 m³ of double walled fuel and diesel exhaust fluid storage tanks, offloading pumps, three individual dispensing systems, one each for heavy vehicles, light vehicles and diesel exhaust fluid, associated piping and electronic fuel control/tracking. The area will be partially covered by a 23m x 21m roof to cover the tanks and dispensers and protect against snow build-up.

18.5.3 Plant Maintenance Shops & Warehouse

The plant maintenance shops and warehouse will located at the western end of the process plant building with a separated wall and 18 m wide by 36m long. The workshop area will be separated from the warehouse with an interior partition wall.

18.5.4 Explosives Storage & Handling

A 6 m wide access road and 100 m x 150 m pad will be constructed to deliver and store explosives required for mine operations. A design buffer of 1.3 km to all other plant site facilities and operations, as well as a 870m buffer to main Access Road and any Mining Operations is assumed. The pad area will be gated and contain bulk storage facilities, and a garage for mobile equipment, and trailer facilities. A separate 50 m x 50 m pad will be constructed along the access road to store the explosive magazine. Explosives and accessories will be prepared and transported to the mine pits as needed. Power for this area will be by generators.

18.5.5 Main Administration Building & Process Plant Offices

The administration office and process plant offices will be 18 m (wide) x 18 m (long), double-storey building located adjacent to the process plant. The administration offices will be located on the upper level and will include offices, first aid facilities, meeting rooms, a lunchroom, and washrooms.

The process plant offices will be located on the lower level with direct access into the process plant building and will include offices, meeting rooms, a lunchroom, and washrooms.

The building will be of prefabricated modular construction placed on precast concrete block footings.

18.5.6 Assay and Geochemical Laboratory

The assay and geochemical laboratory will be a 19.5 m (long) by 12.5 m (wide) building. This laboratory will house equipment for guiding ongoing mining and process plant operations.

18.5.7 Other Facilities

Other facilities to support the process plant operations include:

- Gatehouse modular building will be 18 m (long) x 3.6 m (wide) x 2.4 m (height) with one boom gate for vehicle access where the gate security personnel will be located;
- Truck scale located within the concentrate loadout area;
- Onsite landfill facilit;
- Propane storage: storage tanks with a covered roof to prevent snow build-up will be located close to the process plant building and will be used for heating the process plant building, as well as fuel for the concentrate dryer.

18.6 Site-wide Geotechnical Investigations

18.6.1 Overview

Ausenco supervised a site-wide geotechnical and hydrogeological field investigation (June to November of 2021) and prepared a factual and interpretive report along with geotechnical design parameters and recommendations in form of an Interpretive Geotechnical Report.

Field reconnaissance was completed at the beginning of the program to verify the locations of the planned boreholes and test pits. The field program included drilling of 26 boreholes and approximately 2,342 meters of drilling, excavation of 70 test pits, installation of 29 hydrogeological instruments (vibrating wire piezometers) and collecting representative core and soil samples for the geotechnical, geomechanical, and geochemical laboratory testing programs. The field program obtained geotechnical, geomechanical, geochemical, and hydrogeological information in the following areas:

- Open Pit Mine: North Pit, and North Pit Extension Zone, and Tom MacKay Diversion Tunnel.
- Waste Rock Storage Facility area – including the NAG waste rock pile, future low-grade ore stockpile, and topsoil stockpiles in this area.
- Tom MacKay Storage Facility (TMSF) North area – including the embankments and Penstock area.
- high-grade ore stockpile (Run of Mine -ROM- Stockpile) and primary crusher area.
- Process Plant site
- collection pond 5 and 6 areas.
- Haul roads.

The following sections summarise the key findings and recommendations from Ausenco's 2021 site-wide geotechnical/hydrogeological investigation.

18.6.2 Geotechnical

The geotechnical laboratory programs included the following tests on representative soil samples collected from the Project's main infrastructures and potential borrow sources for general construction materials:

- grain size distribution determination;
- Atterberg limit tests;
- soil classification determination (according to USCS);
- specific gravity determination;
- proctor compaction tests;

- permeability tests (determination of hydraulic conductivity); and
- consolidated undrained (CU) Triaxial shear tests.

In addition, the geotechnical laboratory programs included the following tests on representative samples collected from the potential borrow sources for aggregates:

- determination of soluble chlorides and soluble sulphates;
- soundness test of aggregates;
- aggregate durability index test;
- micro-Deval abrasion test on fine/coarse aggregate;
- Los Angeles abrasion test;
- flakiness index of aggregate test; and
- elongation index of aggregate test.

Based on the subsurface investigation carried out across the site as part of the 2021 feasibility study investigation, the bedrock encountered in the infrastructure area is mainly Bowser Lake sandstone, siltstone, and mudstone while the bedrock encountered in the Open Pit area (North Pit, North Pit Extension Zone, and Tom MacKay Diversion Tunnel Alignment) are Hazelton Group Iskut River Formation, Andesite, Rhyolite and Contact Mudstone.

Evidence of inactive and active landslides, soil creep and bank erosion were observed around the project site. However, they do not pose a significant risk to the project facilities. No permafrost was encountered in any of the test pits and boreholes during the field campaign.

Considering very dense soil and soft rock for the site in the upper 30 m, average shear wave velocity (Vs30) varies between 360 m/s and 760 m/s for Site Class C (NBCC, 2015) average shear wave velocity of 450 m/s) and between 760 m/s to 1200 m/s corresponding to rock in Site Class B (NBCC, 2015) average shear wave velocity of 1,200 m/s).

The seismic disaggregation based on magnitude and distance analysis showed that events with the highest incidence correspond to active shallow crustal seismic sources, earthquakes with magnitudes between 4.0M to 6.2M at minimum epicentral distance between 0 to 22 km from the site. The return periods of 475, 2,475 and 10,000 years were considered for maximum accelerations (PGA: peak ground acceleration) and Sa 0.2 s and 1s. The seismic disaggregation analysis per seismic source showed that F6 Shallow crustal source has greater influence on the project area than other seismic sources. The study behind the framework for the National Building Code of Canada (NBCC, 2015) contains information about the neotectonic structures that was included in the seismo-tectonic model for active faults nearest to the Eskay Creek Project.

The Freezing Index is measured in degree-days (Celsius) and has been estimated based on the average of three (3) Environment Canada weather recording Stations in the project area – Telegraph Creek (north of site), Bowser Lake (southeast of site) and Hazelton (southeast of site). The values from each site are averages based on almost 30 years data (1976-2005). The predicted variance due to climate change is an estimated 10-25% decrease. The calculated Design Freezing Index is 1764 degree-days. The other variables in predicting frost penetration are based on the soil conditions and the specifications for the rockfill applications. Using this design freezing index, the estimated maximum frost penetration depth for the natural ground at this site is 1.7 m below finished ground surface elevations.

The potential for soil liquefaction during a significant earthquake is considered negligible at this site.

Based on the soil properties identified, the following backfill slope values recommended for the soil types are listed below.

- Rockfill: 1.5:1 (H:V): The recommended 1.5H:1V slope is for max 10 m high backfills without any benching. For backfills more than 10 m, 4 m wide benches are required (every 10 m).

- Structural fill: 1.75:1 (H:V) The recommended 1.75:1 (H:V) slope is for max 8 m high backfills without any benching. For backfills more than 8 m, 4 m wide benches are required (every 8 m). Heights of more than 8 m are not recommended due to the risk of local failures.

- Native soil: 2.5:1 (H:V) The recommended 2.5H:1V slope is for max 6 m high backfills without any benching. For backfills more than 6 m, 4 m wide benches are required (every 6 m). Heights of more than 6 m are not recommended due to the risk of local failures.

Recommended allowable bearing capacity values on this site is based on soil properties identified and range from 1.5 kg/cm^2 (for Silt) to 10 kg/cm^2 (for Conglomerate bedrock) in accordance with the recommendations provided by Bowles (1982). These bearing are values are deemed to be sufficient to support most shallow foundations proposed within the plant site including the raft foundation, slab-on-grade (SOG), and pad footings.

18.6.3 Hydrogeology

Groundwater was often encountered at or near refusal in many of the test pits excavated during the 2021 field campaign, however due to the amount of rain in the Eskay Creek area it may not necessarily be a true indicator of the groundwater level. Groundwater level in the boreholes ranged from 0.4 to 8.3 mbgs across all of the site infrastructures.

Groundwater levels for the piezometers in the pit area were from 31 m to 75 m below surface and thought to be influenced by the maintenance pumping in the former underground workings. In the WRSF area, the groundwater level is closer to surface: from 1.5 m to 4 m in shallow-screened wells, and from 12 m to 33 m in deeper screened wells. Seasonal water level variation is about 3 m – 4 m in deeper screened wells and 1 m – 2 m shallow-screened wells. In areas of the proposed TMSF dams, the groundwater levels vary between 5 m – 10 m below surface but can be within several metres of ground level in the Fall and snow melt periods.

In the areas where groundwater table is shallow at the site, some dewatering will be required for service trenches and excavations. The anticipated rate of groundwater inflow into excavations is expected to be moderate and should be able to be handled by typical sump pump systems and drainage ditches, depending on the actual depth and location of the excavation work.

Previous hydrogeological studies in the mining area (by Golder) had identified moderately permeable hanging wall volcanics and sediments and poorly permeable footwall volcanics (rhyolite). The two-order of magnitude difference in between these materials was attributed not only to stratigraphy (i.e., rhyolites underlying andesites / mudstones) but also to the fine-grained texture and / or alteration of the rhyolite. North-south trending faults systems are considered highly transmissive (e.g., the Andesite Creek Fault, K ~ 1E-03 m/s) whereas east-west trending structures (e.g., Reidel Shears) were considered less transmissive.

Packer testing during the Eskay Creek Feasibility field program, has shown that, the rhyolite has lower hydraulic conductivity (K) than the andesite and mudstone in the top 100 m, but not at greater depths and not the two orders of magnitude considered in previous work. In the 2021 geotechnical program, 12 additional boreholes were drilled in the North Pit area including two (2) boreholes on the Tom MacKay diversion tunnel alignment to compliment the nine (9) boreholes completed previously. Most of the boreholes were drilled in the North Pit Extension area and completed with twinned vibrating wire piezometers. The data gathered from the North Pit Extension areas, are indicating a higher K geomeans for the hydro-stratigraphic units (HSUs) compared to the previous results. This area, is proximal to Tom MacKay Creek, which may be a factor in observing this contrast in the K data values obtained.

The 2021 packer test program provided 71 additional packer tests for a total of 111 for the 2020 and/ 2021 programs. The updated database confirmed the conceptual hydrogeological model of the project area and was used in recalibrating the numerical groundwater model and conducting pit dewatering studies.

18.7 Topsoil Stockpiles

Both organic materials and topsoil will be generated during construction activities. Nine (9) topsoil stockpiles (TSS 1 through 9) have been strategically placed around the Project to store organic materials and topsoil for later use in closure of mine facilities Figure 18-1). The stockpile have 3:1 (H:V) side slope based on a stability analysis. The materials will be stacked in thin lifts and compacted to improve their overall stability. The combined volume of the stockpiles is approximately 280,000 m^3 and range in height from 12 to 15 metres. Each topsoil stockpile is surrounded by a silt fence and hay bales to protect the environment until a vegetative cover is established over the stockpiles.

18.8 Waste Rock Storage Facilities

The waste rock storage facilities are discussed in Section 16.11.

18.9 Tom MacKay Storage Facility (TMSF)

18.9.1 Historical Tailings Deposition

The TMSF 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.

18.9.2 Waste Material Storage Disposal and Site Selection

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. 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 109.4 Mt of tailings and PAG waste rock. The FS TMSF requires three embankments located at the north end of the facility one embankment at the south end to contain the required volume of tailings and PAG waste rock and 3 m of water cover during operations and 5 m post closure. Figure 18-2shows the general physiographic and hydrogeological setting of the TMSF.

NAG waste rock will be deposited into WDW, WDN and WDNE WRSFs next to the open pits along with in-pit disposal towards the end of the Project in the north pit (Figure 16-18).

18.9.3 Dam Break Analysis

The embankments for the tailings and PAG waste rock embankments of the TMSF at Eskay are designed in accordance with Canadian Dam Association (CDA) "Dam Safety Guidelines" (CDA 2013) and Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (2016), 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 FS design for a range of conditions and a failure of these embankments is not likely to occur. The dam/embankment break 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 embankments.

The dam/embankment break and inundation study for the TMSF was completed following CDA guidelines (CDA 2013). The study was undertaken to provide an understanding of the potential consequences of a TMSF embankment failure and was structured to estimate the potential zone of inundation that would result from a breach of an embankment 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 internal 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 headwater of Tom MacKay Creek. Tom MacKay Creek flows into Ketchum Creek and then into Unuk River. Based on an embankment break/breach occurring on any of the three (3) north embankments, water would flow into Tom MacKay Creek. Based on a dam/embankment break occurring on the southern embankment, water would flow into Coulter Creek drainage and then into Unuk River. If a south embankment failure occurred, a portion of the tailings stored in the southern third of the TMSF could be released into Coulter Creek and end up in the Unuk River, but the basin behind the southern dam should act as a very large sediment pond capturing the majority of the mobilized tailings.

Tom MacKay Creek is steep channel with a bed morphology of cobble substrate and vegetated banks with exposed bedrock with multiple fish barriers. Based on information provided by fish presence studies by TEEM (2022), 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 cascades/waterfalls which are barriers and is also a non-fish bearing stream. Similarly, Coulter Creek is a steep creek with fish barriers and is non-fish bearing in its upper reaches and headwaters. The Unuk River is a low gradient, medium to large braided river that flows into the Pacific Ocean through the United States. The Unuk River is fish bearing with a number of species including Pacific salmon, Dolly Varden char, and eulachon.

The outflow hydrographs generated due to hypothetical dam embankment breach, including the PMF, was modelled and routed downstream into Tom MacKay Creek (north) and, separately into Coulter Creek (south) using HEC-RAS model. The model was used to predict the extent of flooding due to the dam embankment 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 embankment breach are combined with occurrence of a PMF in the project area and consideration of potential downstream consequences to the Unuk River. HEC-RAS was used to prepare inundation map for the impacted areas.

Based on the embankment breach analysis and expected area of inundation downstream of the TMSF (i.e. tailings and PAG waste rock storage facility), the consequence rating of an embankment 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". Therefore, the facility was designed in accordance with these documents.

18.9.4 Tom MacKay Storage Facility Design Assumptions/Criteria

The proposed process plant site location in relation to the TMSF is shown in Figure 18-4. The flotation process will produce a combined tailing stream that contains both NAG and PAG materials. In addition, PAG waste rock will also be placed in TMSF. Sub-aqueous disposal requirement was conservatively assumed as a design requirement to prevent acidification and metal leaching and is still one of the best available practices (Refer to Figure 18-2).

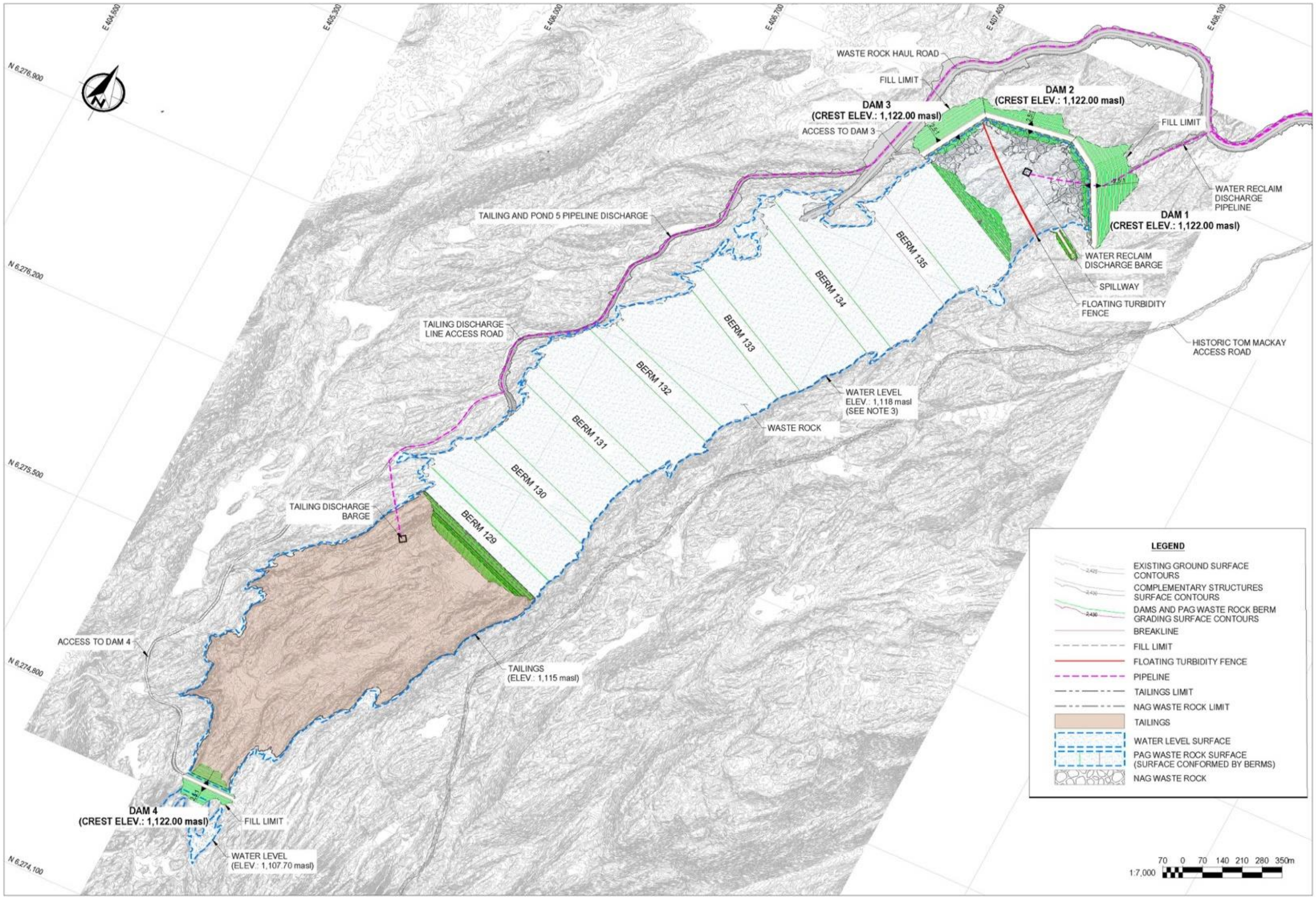
The TMSF was designed based on the following criteria:

- required storage of 27.9 Mt of NAG and PAG tailings;
- tailings particle size P80 = 30 µm;
- tailings discharge solids content = 20% (by mass);
- dry tailings density of 1.41 t/m^3;
- required storage of 81.5 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 7 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; and
- meeting or exceeding applicable regulatory requirements and industry guidelines for stability and design flood events.

Figure 18-4: TMSF General Layout



Note: Figure prepared by Ausenco, 2022.

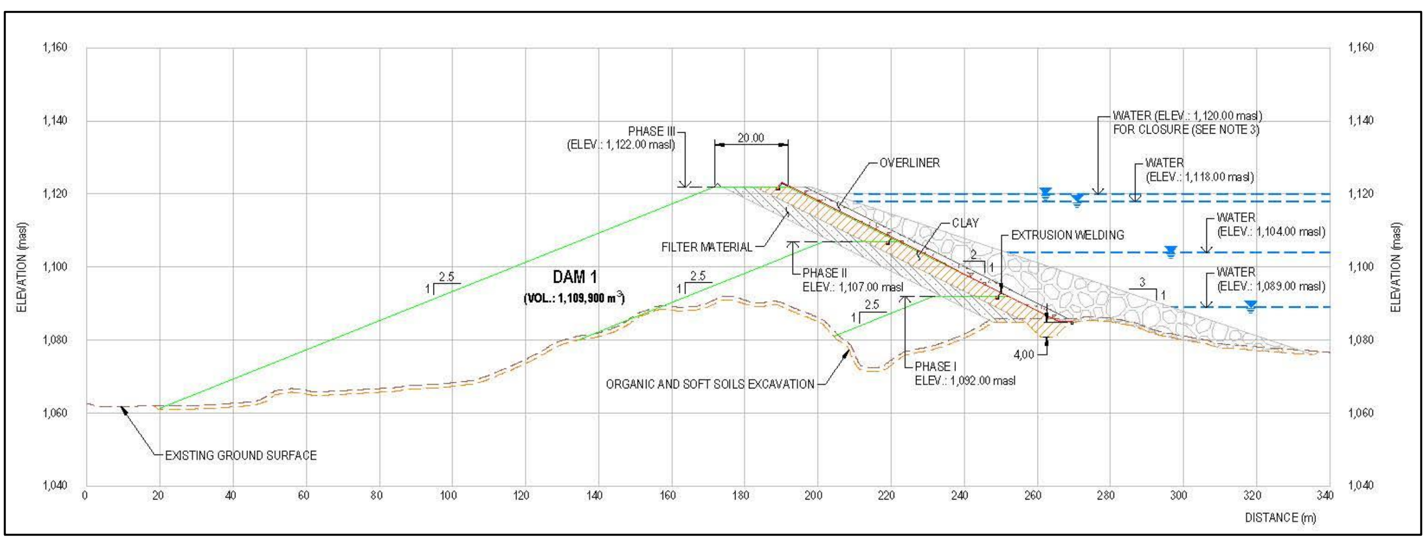
18.9.5 Tom MacKay Storage Facility Design and Construction

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 Mm^3 elevation 1,079 masl, which is the approximate water surface elevation at the outlet of the basin.

The TMSF embankments are 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 for the 3 northern embankment and a stacked center low permeability core construction for the southern embankment shown in Figure 18-6. NAG mine waste from the pit will be utilized as the primary construction material. The upstream side of the northern embankments 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. The southern embankment is different from the other embankments since there will be water on both sides of the embankment. The southern dam utilizes water reservoir design since downstream construction of the dam is not possible due to water impounding on both sides of the embankment. The southern embankment utilizes a central low permeability core with filter zones on either side of the core and a NAG waste rock shell.

TMSF will be constructed in three phases over the life of mine based on storage and operating criteria. The TMSF northern embankments design concept is in shown in Figure 18-5, based on the geotechnical investigation.

Figure 18-5: Northern Embankment Section



Note: Figure prepared by Ausenco, 2022.

The TMSF Southern embankment design concept is shown in Figure 18-6.

Figure 18-6: Southern Embankment Section

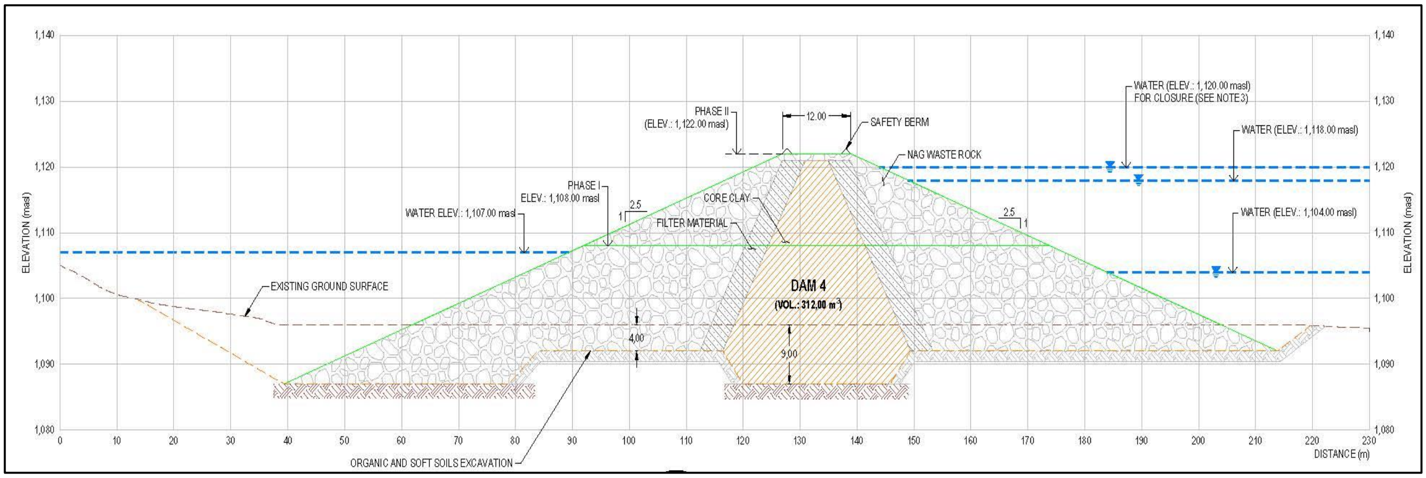
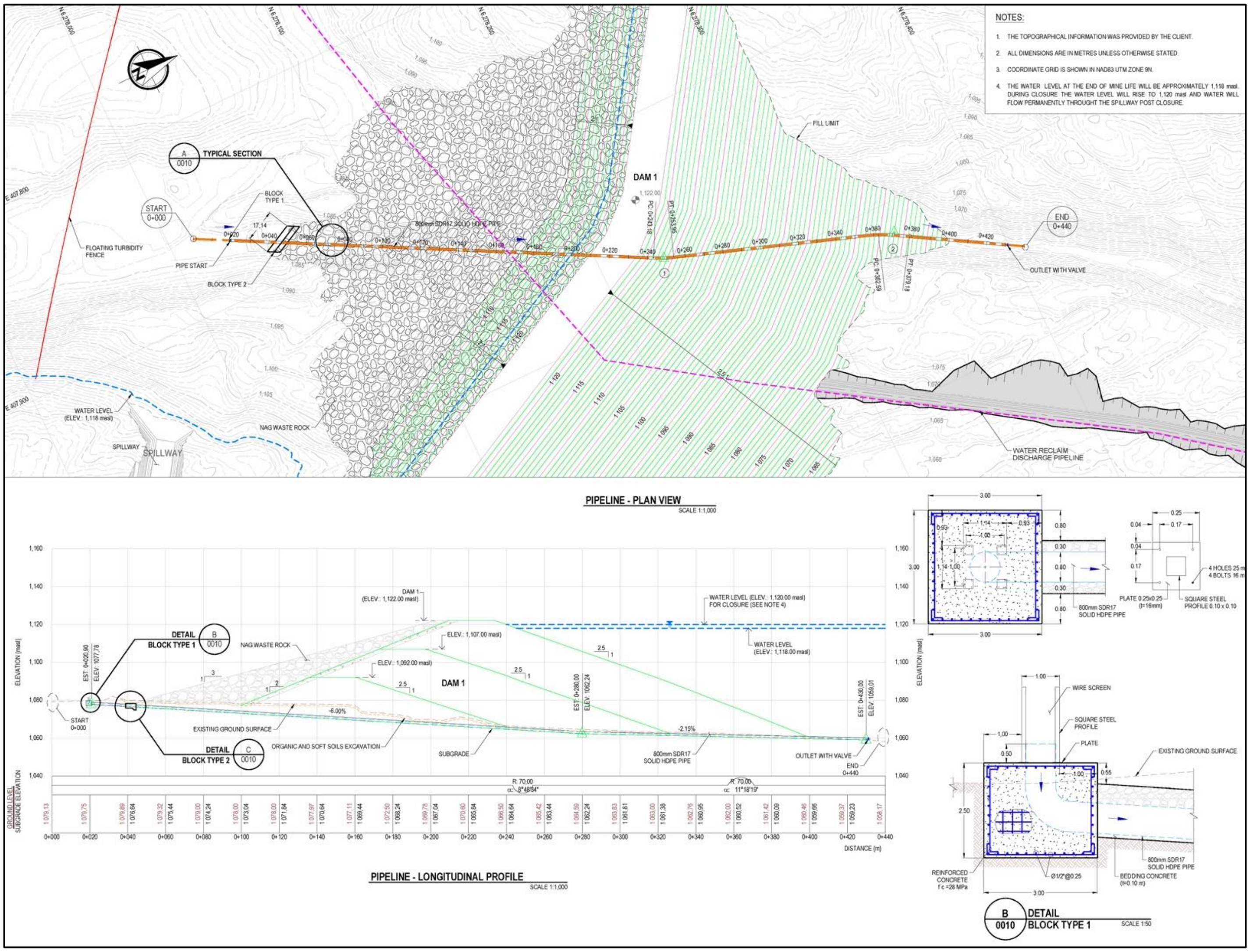


Figure prepared by Ausenco, 2022.

For Phase 1 the three northern 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 embankments have 3.0:1 (H:V) upstream slope and 2.5:1 (H:V) downstream slope. A penstock will be installed through embankment 1 (northeastern embankment) along the thalweg of Tom MacKay Creek. The penstock will be constructed with HDPE Solid wall pipe with a seepage collar on the upstream embankment 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 18-7). The upstream face of the embankments will be lined with gravel and rip rap to protect the liner system. The NAG waste rock shell and protective upstream NAG waste rock liner protection will be placed and compacted by the mining fleet. The filter zone, low permeability soil liner, geomembrane, overliner, and penstock will be complete by a contractor. For Phase 2 all three northern embankments will be raised in Years 1 and 2 of operations to a height of 1107.0 masl and the 1st Phase of the southern embankment will be constructed to an elevation of 1,107.0 masl. For Phase 3 all four embankments (2nd Phase for the southern embankment) will be raise in years 4 and 5 of operations to their final height of 1,122.0 masl. The four embankments will range in height (downstream toe to crest) from 24m to 60m. The Phase 2 and 3 northern embankments construction will utilize downstream construction method and the Phase 2 southern embankment construction will utilize a central core construction method. The design is the same as Phase 1 for the northern embankments with upstream slopes of 3:1 (H:V) and downstream slopes of 2.5:1 (H:V) and a crest width of 20m and the southern embankment with both upstream and downstream slopes of 2.5:1 (H:V) . The closure spillway will be constructed in Year 7 and most of the spillway will be constructed in bedrock and any soil zones will be constructed with Riprap and grout to convey the PMF. Post-closure will utilize a water cover of 5 m to prevent the PAG tailings and waste rock from becoming acidic , which the elevation of the water level will be controlled by the inlet invert (1,120 masl) of the spillway.

Figure 18-7: Tom MacKay Storage Facility Penstock



Note: Figure prepared by Ausenco, 2022.

A waste rock haul road and tailings discharge pipeline access road will be constructed from the open pit/plant to TMSF. In addition, the southern embankment access road will be constructed from the tailings discharge pipeline access road to southern dam. 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. In addition, floating turbidity fences will be installed around active PAG waste rock berm construction areas and tailings discharge areas. 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.

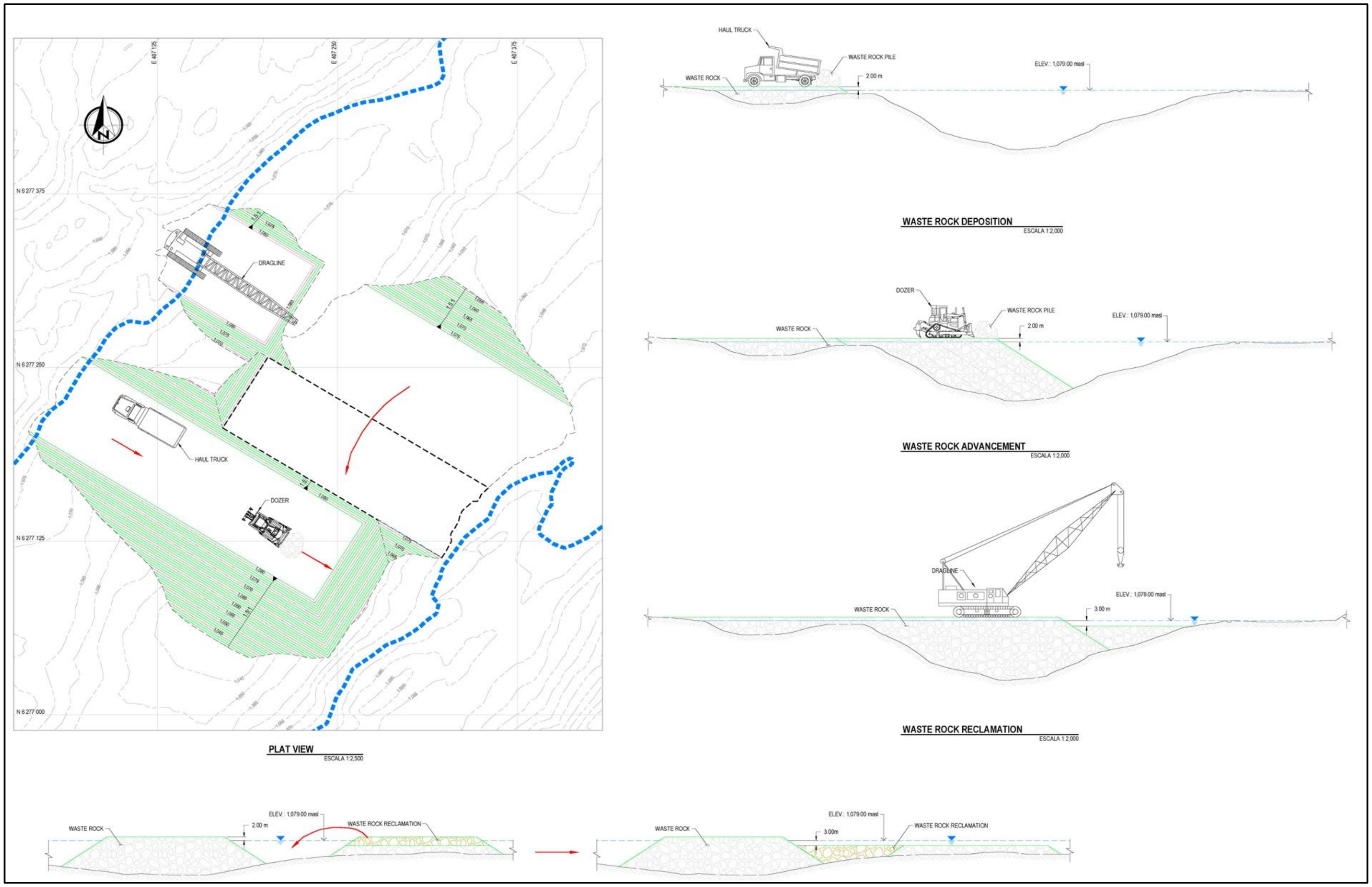
18.9.6 Operations of the Tom MacKay Storage Facility

The operation 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 manifold with multiple spigots, to reduce the discharge velocity, at the bottom of the facility to promote faster settlement of the tailings. In addition, to promote tailings settlement, an inline flocculant dosing system will be installed along the shoreline of TMSF. The discharge line will be moved around the south end of the facility to promote layering deposition of the tailing. During winter months when TMSF is covered by a thick layer of ice, holes will be dug into the ice and the tailings discharge line and manifold will be placed in the hole to the bottom of TMSF along with a floating turbidity fence. Periodically, new holes will be drilled dug and the tailings line and floating turbidity fence will be repositioned.

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 PAG waste rock will be by creating berms across the facility from west to east. The berms 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 dragline excavator. The heavy equipment will utilize the center 30 m, while lighter equipment, dozers, will push the PAG waste rock to create the 65 m wide berms. Once completed the next berm will be constructed next to the completed berm. During the construction of the next berm, dozers 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 18-8). The final height of the berm will be 3 m below the water surface during operations.

Figure 18-8: PAG Waste Rock Deposition Plan



Note: Figure prepared by Ausenco, 2022.

Each berm will be constructed approximately 0.3 to 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 and snow melt, run-on from the surrounding watershed, and TMSF). The design calls for the release of a base flow and only allows for retention of water during peak runoff months. Based on 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 5 m of water cover post closure. It will take 1 to 1.5 years, depending on the end of operations, to achieve the 5 m of water cover and have continual flow through the closure 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 or holes in the ice. Due to the fine ore grind, P80 = 35 µm, the end of the pipeline will be positioned close to the bottom of facility (deposited tailings) with a multi-outlet manifold to reduce discharge velocity and to maximize settling while minimizing entrainment of fine particles to the surface of the TMSF. The minimum water depth over the tailings would be 3 m during operations and 5 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 tailings barge will be moved around the TMSF to develop an even tailings distribution across the TMSF floor and during winter multiple holes will be dug in the ice to distribute tailings evenly in TMSF.

The tailings and PAG waste rock deposition rates are provided in Table 18-2 and the projected TMSF storage capacities are outlined in Table 18-3. Tailings are planned to be discharged at 20% solids and will have an overall dry bulk density of 1.41 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 three small northern embankments to an elevation of 1092.0 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 years 1 and 2 of operations, embankment raises will be required to be constructed to an elevation of 1,107.0 masl and a final raise in years 4 and 5 to an elevation of 1,122.0 masl to store the balance of the LOM tailings and PAG waste rock while maintaining 5 m of water cover post closure.

Table 18-2: Planned Tailings and PAG Waste Rock Deposition Schedule

Year	Annual Tailings Production (Mt)	Annual PAG Waste Rock Production (Mt)	Total Waste Deposition (Mt)
-2	-	1.0	1.0
-1	0.3	2.3	2.6
1	2.4	5.0	7.4
2	2.8	11.7	14.5
3	2.8	12.8	15.6
4	2.8	12.1	14.9
5	2.8	11.9	14.7
6	3.5	12.7	16.2
7	3.5	9.7	13.2
8	3.5	2.3	5.8
9	3.5	-	3.5
Total	27.9	81.5	109.4

Note: Table prepared by Ausenco, 2022

Table 18-3: TMSF Projected Tailings and PAG Waste Rock Storage Capacity

Elevation (masl)	Accumulated Capacity (Mm^3)	Comments
1,050	0	TMSF Lowest Portion of Bed
1,079	12.6	Existing TMSF Water Surface
1,086	19.3	Top of Waste Materials Phase 1
1,089	23.7	Water Surface Phase 1
1,092	27.4	Top of Embankments Phase 1
1,101.0	40.0	Top of Waste Materials Phase 2
1,104.0	44.9	Water Surface Phase 2
1,107.0	50.1	Top of Embankments Phase 2
1,115.0	64.5	Top of Waste Materials Phase 3
1,120.0	76.9	Water Surface Phase 3
1,122.0	81.4	Top of Embankments Phase 3

Note: Table prepared by Ausenco, 2022.

18.9.7 Tailings Storage Facility Stability Analysis

A section through the tallest portions of the embankments were selected as the most critical sections. 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.

18.9.8 Instrumentation and Monitoring

Geotechnical instrumentation will be installed along plane through TMSF embankment. The instrumentation will be installed during construction phases and monitored over the life of the Project, and into post closure. Geotechnical instrumentation is comprised of vibrating wire piezometers and slope inclinometers and will be installed into the foundations and embankment fill.

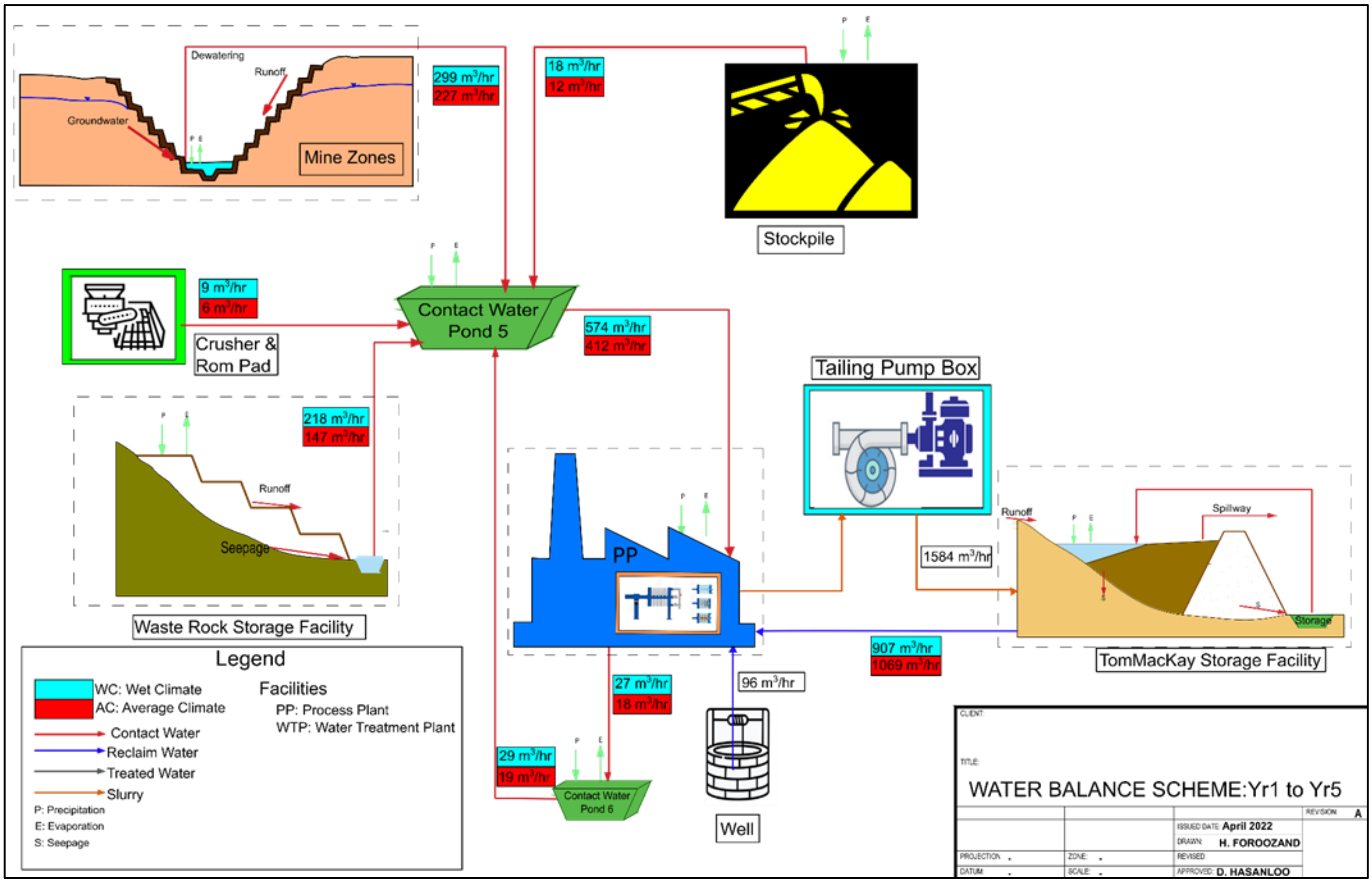
18.9.9 Tom MacKay Storage Facility Closure

TMSF closure will consist of removing the tailings discharge line and barge, process water reclaim 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 5 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 5 m of water cover even during extreme drought conditions.

18.10 Side Wide Water Balance

A site-wide water balance (GoldSim) was developed based on the conceptual model shown in Figure 18-9 to Figure 18-10, which was used to inform water management design and predict the potential contact water volumes through the life of mine. This analysis does not cover water need for dust control.

Figure 18-9: Eskay Creek Project site-wide water balance based on average flow rates (Yr 1 to Yr 5).



Note: Figure prepared by Ausenco, 2022.

Figure 18-10: Eskay Creek Project site-wide water balance based on average flow rates (Yr 6 to Yr 9).

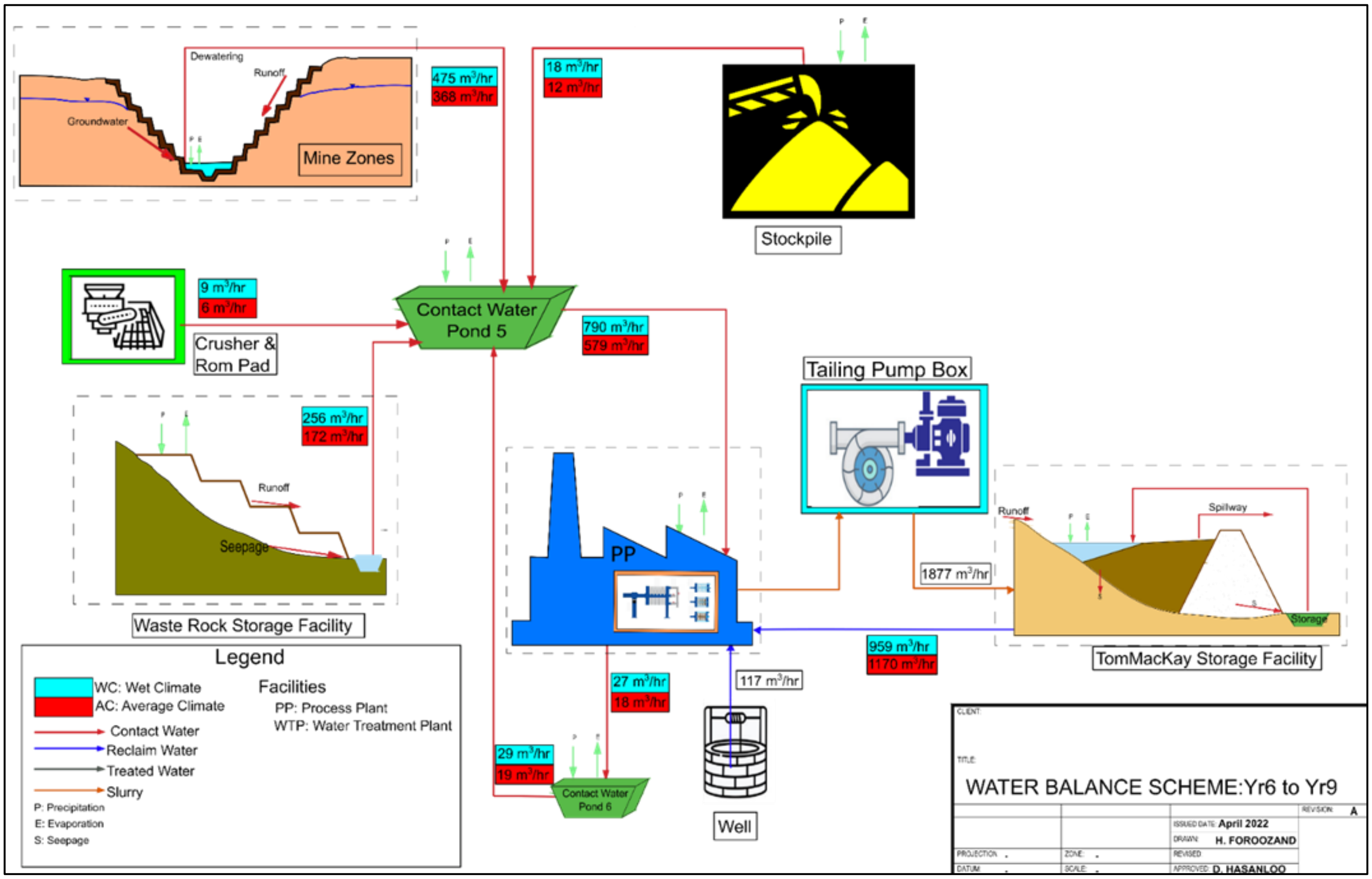


Figure prepared by Ausenco, 2022.

The model was set up to run for simulations with an output time of 11 years (Yr -2 to yr 9). The simulation timestep was daily for flow estimates under average and wet climates. It should be noted that this analysis does not cover water need for dust control.

The Main Pit will be mined during construction and operation from Yr-2 to Yr9). Any water arriving in Pit bottom (intercepted Precipitation or influent seepage) is contact water. A portion of the water will be lost from the Pit via evaporation.

Two scenarios were originally considered for water balance modelling: In the passive dewatering scenario, contact water in wet climate and high flow can reach to 326 L/s while high flow in average climate is 224 L/s. In the active dewatering scenario, contact water in average climate and high flow can reach 263 L/s while Pit residual flow is 1.5 L/s. The South Pit contact water in wet climate and high flow can reach 30 L/s while high flow in average climate is 20 L/s.

Pond 5 plays an important role in receiving contact water from WRSF, Stockpile, Crusher & ROM Pad, Pit dewatering and overflow from Pond 6. The contact water in Pond 5 is distributed to TMSF. Pond 5 overflow to TMSF reaches its peak in Yr9, when the main Pit dewatering will be collected. Maximum overflows to TMSF are estimated to be 648 and 444 L/s under the wet and average climate, respectively. Pond 6 stores contact water from Plant Site and overflow to Pond 5. Maximum overflows are estimated at 28.3 and 18.7 L/s in Yr-2 to Yr9 under the wet and average climate, respectively.

The Feasibility Study Water Balance report has investigated the potential impact of Precipitation variability on important model's outputs. It should be noted that Precipitation is one of the most critical input variables for water balance calculations because it is the immediate source of water for the land surface hydrological budget.

Model uncertainties are an intrinsic part of all large mining projects. Each of the measured or assessed flow rates or volumes in the site-wide water balance model (the Model) has random uncertainties caused by combinations of several factors. Uncertainty is quantified by the Monte Carlo simulation (MCS) method. MCS produces distribution functions for the model's outputs by repeated sampling of the input variable (in this case, Precipitation data). Latin Hypercube Sampling method was used to simulate the entire system with 1000 realizations. The MCS results of important Model's outputs under the average climate are summarized below:

- The MCS results show that WRSF average high flow is estimated at 140.2 L/s, with 95% confidence interval of [67, 207] L/s.
- The MCS results show that Stockpile average high flow is estimated at 11.8 L/s, with 95% confidence interval of [6.6, 16.6] L/s.
- The MCS results show that Mine Zone average high flow is estimated at 230 L/s, with 95% confidence interval of [108, 345] L/s.
- The MCS results show that Pond 5 average high inflow is estimated at 442 L/s, with 95% confidence interval of [220, 650] L/s.
- The MCS results show that Pond 6 average high inflow is estimated at 18.6 L/s, with 95% confidence interval of [7.9, 30.3] L/s.

18.11 Surface Water Management

As shown in Figure 18-9 and Figure 18-10, the contact water originating from the mining footprint will flow or be pumped to contact water pond 5. During construction years, contact water from this pond will be pumped to the TMSF. During operations all contact water will be used in the processing plant and any excess water sent to the TMSF with the tailings. Water inflow into TMSF comes from the tailings being discharged into the facility, direct rainfall and snow melt, and surface runoff from rainfall and snow melt. The outflow from this facility is water reclaim, seepage, and discharge through the penstock. A base flow through the penstock will occur year-round into Tom MacKay Creek. All roads will have diversion ditches to convey the design storm event and discharge into strategically placed sediment ponds that discharge into the environment. Non-contact water will be conveyed around mine facilities in diversion channels where possible.

The Surface Water Management details are provided in Section 20.2.3

18.12 Water Supply

The freshwater makeup for the plant will be supplied by groundwater sourced from water wells located southwest and south of the processing plant. Additional process water will be recycled from the TSF. Pilot boreholes have demonstrated potentially high-yielding bedrock aquifers with driller flow yields of up to 10 L/s.

The nominal freshwater makeup flowrate will be approximately 100 m^3/hr or 30 L/s. Vertical pumps within the wells will pump the fresh water, via a 100 mm nominal bore HDPE pipeline, to the freshwater storage tank within the process plant.

18.13 Snow Management

The snow management plan for these facilities is based on the following working assumptions:

- Only active areas of Pit, Waste Rock Storage Facilities (WRSF), Primary Crusher & ROM Pad, Haul Road, and Plant Site require snow management.
- It is assumed that snow on Pit and WRSF can be managed on-site by using Leap-Frog Method (moving snow from active areas and depositing it on inactive areas).
- According to Ausenco Technical Sample Site-Wide Surface Water Management Plan (2022) report, runoff and snow from WRSF and Haul roads to Tom MacKay Storage Facility (TMSF) were assumed to comprise of NAG material.
- There will be benches in the Pit that advance slowly or are inactive. Therefore, 25% of Pit area is assumed to be active at each time, which requires snow management.
- Only a relatively small amount of snow can be stored along the Haul Road to TMSF during snow season. It is assumed that 150-ton haul truck with 7-metre width is the largest vehicle using the road; It should be noted that a minimum Haul Road width of at least 3x the width of the largest vehicle using the road shall be maintained. Therefore, Haul Road to TMSF is not wide enough for storing the majority of road snow along the road.
- Snow is assumed to be Damp New Snow type (Paterson, 1994), with a snow density of 100-200 kg/m^3.

Potential snow compaction is conservatively assumed to be 50%.

General considerations for snow management plan:

1. Pit snow management plan

   Pit benches that advance slowly or are inactive can be used to store snow as much as possible. The snow from the pit area is considered PAG (Potential Acid Generating)-snow and the resulting snowmelt should be managed.

2. WRSF snow management plan

   According to Ausenco Technical Sample Site-Wide Surface Water Management Plan (2022) report, WRSF is NAG (Non Acid Generating), and its snowmelt will be directed to Pond 5. It is assumed that snow during WRSF construction can be managed on-site by using a leap-frog method (moving snow from active areas and depositing it on inactive areas). Top clean snow (0.5m above ground and higher) of the inactive area can also be cleared and deposited into deposited into Tom MacKay Creek. Then the inactive area can be used for managing snow on active areas as well as excess snow of other facilities (e.g., plant site and haul roads) in above normal weather conditions.

3. Stockpile area snow management plan

   Snow on the stockpile and the primary crusher & ROM Pad is considered in contact with PAG material. The first option to manage its snow is to use inactive areas of low-grade stockpile for snow deposition, and its snowmelt should be collected in pond 5. In full-scale operational scenarios and extreme snow conditions, excess snow should be deposited in the pit's inactive area due to its PAG snow type.

4. Plant site area snow management plan

   The Plant site has both NAG and PAG snow. The NAG snow can be deposited in the WRSF's inactive areas, while the PAG snow should be transported to the pit's inactive area.

5. Haul Road to TMSF snow management plan

   According to the Technical Sample Site-Wide Surface Water Management Plan (Ausenco, 2022) report, the Haul Road to TMSF is considered NAG. Even though snow on the Haul Road is considered NAG, the snow from these areas must be managed. In normal operational conditions, any snow on Haul Road should be removed frequently to maintain the flow of traffic. It should also be noted that all the road snow cannot be stored along the Haul Road to the TMSF during the snow season since it is not wide enough. Therefore, its snow could either be transported to the TMSF or deposited on the WRSF's inactive area. If traffic is stopped due to weather conditions, the non-contact, clear snow can be pushed or blown away to the environment.

18.14 Camps and Accommodation

A temporary camp is planned to be established at the start of construction at a location near the BC Hydro Forest Kerr facility and will be in operation for 5 years (60 months), by which time the Permanent Camp will be established near the Processing Plant.

The temporary camp will be sized to accommodate 210 beds, complete with dormitories, kitchens/dining areas, recreation room, and associated services, security and medical facilities, based on a 5-year lease. These 210 beds, together with the camp facilities at the existing historical site, will be able to accommodate the requirements during two years of construction. Once the construction is complete, the existing historical camp will be gradually relocated closer to the processing plant facility, and the newly established Temporary Camp will continue to operate during the first three years of operation. During this time, the staff will be bussed to the mine and process plant area using designated shuttle services.

The new permanent camp will be constructed and installed prior to end of third year of operation, next to the relocated modules from the historical camp, near the process plant area, with modular units comprising of additional 180 individual dormitories with designated washrooms for each bedroom. Kitchen/dining area, recreation room, a boot/jacket room for personnel to enter and leave accommodations, and security/medical facilities.

The permanent camp will need to be multiple-level type to minimize footprints. Both camps will be heated with propane and will be connected with portable generator(s) for emergency power supply.

19 MARKET STUDIES AND CONTRACTS

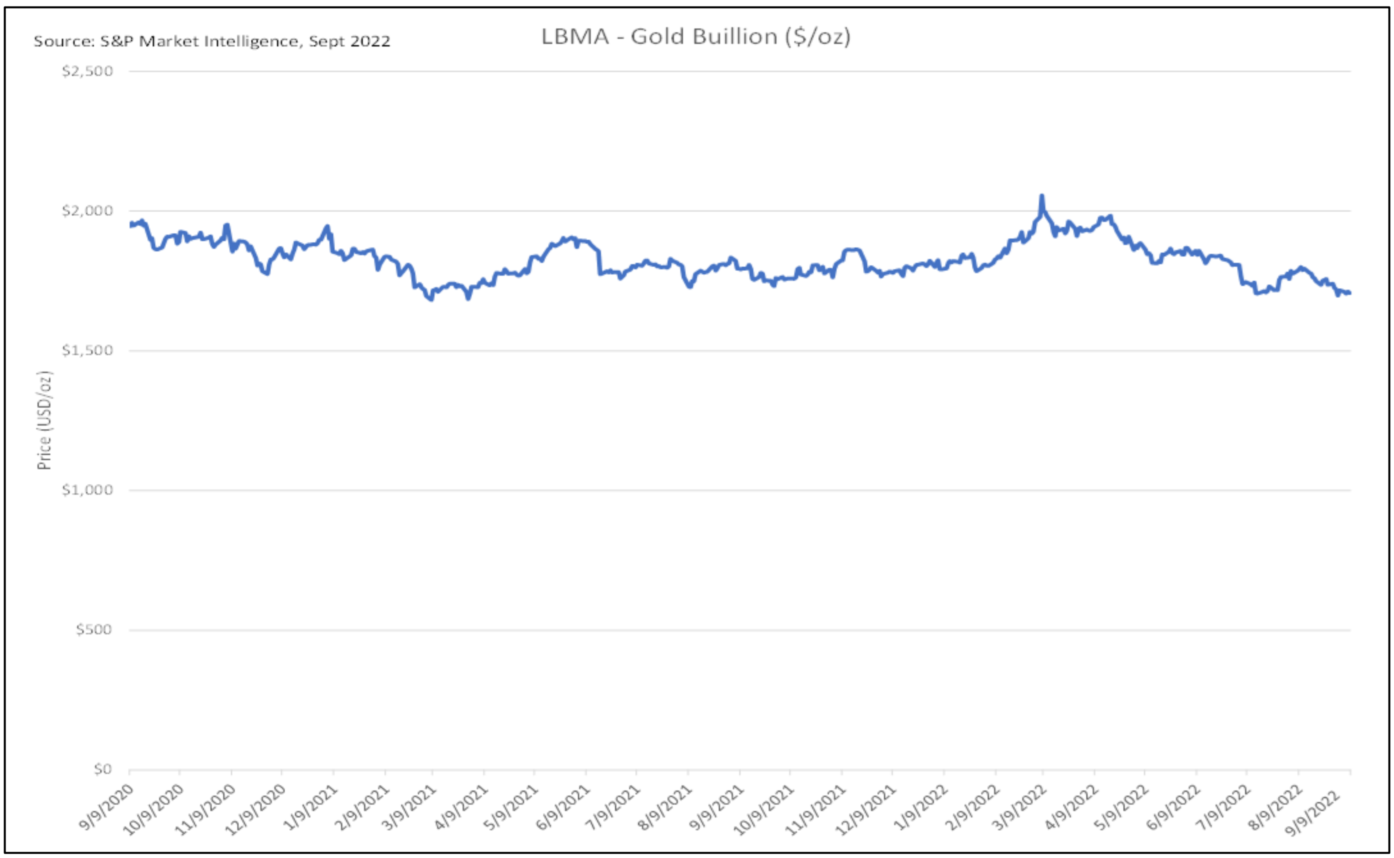
19.1 Introduction

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 Metal Prices

Metal price selection of US$1700/oz Au and US$19/oz Ag was based on reviewing other recently published feasibility studies, long-term analyst consensus prices and the two-year trailing average of gold (US$1,826/oz) and silver (US$24/oz) prices as of September 9, 2022.

Figure 19-1: LBMA Daily Closing Gold Price (2-year span, US$/oz)



Note: Figure from S&P Market Intelligence, September 2022.

Figure 19-2: LBMA Silver Daily Closing Price (2-year span, US$/oz)

Note: Figure from S&P Market Intelligence, September 2022.

19.3 Market Studies

The proposed concentrate is a complex precious metal concentrate with gold content from 20 g/t to 50 g/t 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, an independent market study was completed by Open Mineral AG to support the NSRs used in the 2022 FS.

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 analysis, the 2022 FS 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 concentrate marketing specialist Open Minerals. The key conclusions and considerations from the marketing study include:

- producing higher-grade concentrates produces better NPVs for most of the variability tests depending on recovery loss incurred with increases in concentrate grad
- 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
- an additional minimum Chinese 13% value-added tax (VAT) is expected if gold grades are lower than 15 g/t and would significantly decrease its marketability. However, this is not expected within the production profile of the Project
- penalties for all the deleterious elements are shown in Table 19-1.

19.4 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 based on 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.

19.5 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 Table19-1.

Table 19-1: Payabilities Contract

Item	Info	Units	Payabilities Contract
Gold Payable (%) Gold	Concentrate Grade Ranges	%	
	0–5 g/t	%	0
	5–10 g/t	%	65.8
	10–15 g/t	%	67.7
	15–20 g/t	%	72.3
	20–25 g/t	%	76.6
	25–30 g/t	%	80.1
	30–35 g/t	%	83.8
	35–40 g/t	%	87.1
	40–45 g/t	%	87.9
	45–50 g/t	%	88.2
	50–55 g/t	%	89.6
	55–60 g/t	%	90.5
	60–65 g/t	%	91.4
	65–70 g/t	%	92.4
	70+ g/t	%	93.3
Silver Payable (%) t	Concentrate Grade Ranges	%	
	200–500 g/t	%	60
	500–1,500 g/t	%	80
Deductions Gold	–	g/t	0
Deductions Ag	–	g/t	100 (if<500 g/t)
Treatment Charges	–	$/dmt	0
Recovery Charges Gold	–	($/oz payable)	0
Recovery Charges Ag	–	($/oz payable)	0
Hg	Free limit	g/t	1000
	Per 100 g/t over	USD/t	1
Sb	Free limit	%	2
	Per 0.5% over	USD/t	4.5
As[1]	Free limit	%	0.5
	Per 0.1% over	USD/t	3

Note: [1]Max As = 3.5% if Au < 50 g/t and Max As = 6.5% if Au > 50 g/t

19.6 Insurance, Representation and Marketing

No allowance has been made for insurance, marketing or representation.

19.7 Comments on Market Studies and Contracts

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analysis to support the 2022 FS.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY

20.1 Environmental Studies

Many environmental studies were completed at the Eskay Creek Mine under various owners, prior to and throughout the mine life. 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. Additional environmental 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 and routine reporting was completed during and after operations.

This section summarizes Skeena's recent studies which began in 2020 and includes additional environmental, social, economic, historical and health baseline studies to reflect current environmental and social conditions. Where available and to provide context, pre-2020 data was reviewed and summarized for the current baseline studies and where suitable for the Project, sampling sites used in earlier studies were re-visited. Information on the Project site climate and physiographic setting is included in Section 5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography. These studies will support further Project design and the assessment process for provincial and federal authorizations and permits.

There are no known environmental issues that could materially impact the issuer's ability to extract the mineral resources or mineral reserves.

Table 20-1 identifies the recent studies in the project area.

Table 20-1: Recent Environmental Studies in the Project Area

Subject Area	Years
Climate and meteorology	• 1981 to 2019 Historic regional data; 2020 to 2021 Local mine site data (Tahltan ERM Environmental Management 2022a)
Air quality	• 2020 (Tahltan ERM Environmental Management 2022b)
Noise	• 2020 (Tahltan ERM Environmental Management 2022c)
Surface water quality and quantity (hydrology)	• 1990 to 2020 Historic and recent data (summarized in Tahltan ERM Environmental Management 2021a) • 1990 to 2021 Historic and recent data (summarized in Tahltan ERM Environmental Management 2022o • 2020 (Tahltan ERM Environmental Management 2021b) • 2021(Tahltan ERM Environmental Management 2022d)
Groundwater quality and quantity	• 2017 to 2021 (Tahltan ERM Environmental Management 2022e)
Aquatic resources	• 1991 to 2020 (Tahltan ERM Environmental Management 2021c) • 1990 to 2021 (Tahltan ERM Environmental Management 2022j)
Fish and fish habitat	• 2020 (Tahltan ERM Environmental Management 2022f) • 2020 and 2021 (Tahltan ERM Environmental Management 2022k)

Subject Area	Years
Soils	• 2020 (Tahltan ERM Environmental Management 2022g) • 2020 and 2021 (Tahltan ERM Environmental Management 2022l)
Geohazards, terrain stability, and soils erosion	• 2020 (Tahltan ERM Environmental Management 2022h)
Vegetation and ecosystems	• 2020 (Tahltan ERM Environmental Management 2021d) • 2020 and 2021 (Tahltan ERM Environmental Management 2022m)
Wildlife	• 2020 to 2021 (Tahltan ERM Environmental Management 2022i) • 2021 Habitat Suitability (Tahltan ERM Environmental Management 2022n)
Social, economics, land and resource use	• Variable years (Falkirk 2022)
Tahltan social, economics, and land and resource use	• Variable years (Falkirk 2022)
Tahltan country foods	• 2020 (Skeena Resources Limited draft undated)
Archaeology	• 2018 (Le Beau, D. and K. Burdeyney 2020) • 2020 (Walker, D. and D. Le Beau. 2021)
Paleontology	• 2020 FIA Preliminary Study (Lifeways of Canada 2022)

20.1.1 Air Quality

The 2020 current conditions indicates that concentrations of potential airborne pollutants are low and representative of an area where there are no sources of air pollution emissions.

20.1.2 Noise

Noise levels documented in 2020 are due to weather, wildlife and anthropogenic activities such as helicopter traffic and vehicles. Some of this anthropogenic noise from other projects in the surrounding area (e.g., projects owned by Coast Mountain Hydro, Seabridge Gold and Garibaldi Resources) are also intermittently present. These projects all have camps or staging areas in the general vicinity of the Project and use the Eskay Creek Mine Access Road.

20.1.3 Surface Water Quality and Quantity

Measurements and sampling of the surface waters since 2020 characterize the current conditions of water resources and will be used to assess potential Project effects on the water catchments of Eskay Creek, Argillite Creek, Tom MacKay Creek, Unuk River and Ketchum Creek. Ketchum Creek drains the project area into Unuk River. Sections of the Unuk River situated upstream of the confluence of Ketchum Creek and the Unuk River are considered reference (i.e. not affected by mining activity) stream sections.

Seasonal stream flow regimes within the project area can be broadly classified as snowmelt driven (i.e. nival). Most annual runoff occurs during spring freshet as the winter snowpack melts, followed by generally decreasing flows in summer with precipitation events producing substantial peak events in late August and early fall. The lowest annual flows generally occur in the late winter months (e.g., February, March), prior to the start of snowmelt. Base flows from the outlet of the historic underground mine portal occur year round and follow a similar seasonal cycle to the local streams and precipitation patterns.

Water quality data from pre-1993 found the Project area had two types of water quality: glacial-fed streams and mountain runoff streams (no glacial inputs). Eskay and Tom MacKay creeks drain the plateau area immediately around the historical mine site and were characterized as neutral to slightly acidic pH, moderately conductive, clear with low turbidity and low TSS, moderately low in dissolved solids with low hardness and alkalinity. The conditions documented in the early 1990's for the glacial influenced streams of Ketchum Creek and Unuk River were circumneutral pH, high conductivity, moderately to high turbidity with moderate to high TSS, high dissolved solids and moderately high hardness and moderate alkalinity (HKP 1993). The water quality of both stream types reflects the highly mineralized bedrock geology through which fast streams have eroded significant, steep, canyon-like features. These streams carry erosion products downstream as suspended particulates, bedload, and dissolved constituents of the water, and deposit fine sediments and gravels in the fish-bearing and turbid Unuk River.

The hydrological regime affects water quality by diluting concentrations of dissolved solids during the open water period, while increasing sediment load and transport, causing high concentrations of suspending sediments and particulate-associated metals. Concentrations of water quality parameters generally followed one of two distinct patterns depending on whether they were present primarily in the particulate or dissolved fraction. Parameters that had a large proportion of particulates, including most metals, were greatest during periods of high flow, while parameters with high dissolved fractions were diluted during periods of high flow and peaked with dissolved solids concentrations in the low flow winter months.

A compilation of historic (i.e. pre-2020) and 2020 water quality data was summarized and showed spatial differences in water quality observed throughout the Project area, which can be attributed to contrasting source waters (as was noted in the early 1990s) and the influence of the former mine operation. The water quality in Eskay and Tom MacKay creeks was typical of unglaciated systems, with greater groundwater, rainwater, and snowmelt influence, resulting in extremely low to (at times) no concentrations of suspended solids and low concentrations of dissolved solids.

In contrast, the elevated glacial erosion in the headwaters of Ketchum Creek and the Unuk River resulted in higher sediment inputs and suspended solids and greater metal concentrations. In general, the non-glacial streams were circumneutral with soft water, while the glacial-fed streams were slightly basic with soft to moderately hard water. Non glacial streams also had low to moderate total alkalinity, conductivity, and total dissolved solids (TDS). Ketchum Creek, although glacial-fed, tended to show characteristics that were intermediate between non-glacial Tom MacKay Creek and the glacially turbid Unuk River. Total alkalinity, conductivity, hardness, and TDS were highest in the winter low flow period and in the glacial-fed streams.

Between 1990 and 2012, the long-term temporal trends in water quality in Tom MacKay and Ketchum creeks were influenced by operation of the historical Eskay Creek underground mine in addition to the natural stream flow patterns and glacial sources. Waste rock, tailings, and sludge were deposited into Albino Storage Facility from 1995 to 2001 and were diverted to the TMSF from 2001 until mine closure in spring 2008. Both of these facilities drain into Tom MacKay Creek, which empties into Ketchum Creek. During operations, mine contact water, mill effluent, and sewage effluent were treated at the mine site and discharged at site D7 beside the mine site into Ketchum Creek.

Although some parameters were naturally elevated during the data collection in historic and recent sampling, especially in the glacial-fed watersheds, trends in certain water quality parameters increased and occasionally exceeded guidelines during mining operations (pre-2008) or the post-mining period, although exceedances of discharge permit limits were not common, with reduced potential for effects tied to higher stream flows. Trends in metal concentrations, including occasional exceedances of water quality guidelines in receiving environment streams related to discharge from the mine in recent years, are being studied to inform water management and mitigation planning. The minor exceedances of water quality guidelines in the receiving environment (e.g., occasional dissolved zinc in Ketchum Creek in recent years) are linked to both natural elevated levels due to erosion and background conditions in this creek, as well as occasional elevated concentrations in mine water discharge.

The only fish-bearing waters are the Unuk River, and there were no apparent water quality guideline exceedances in the Unuk River related to the mine discharge, despite frequent natural exceedances of water quality guidelines for total metals in both upstream reference (i.e. non-mine influenced) and sites downstream of the Unuk/Ketchum Creek confluence. These exceedances of water quality guidelines in the Unuk River were mainly attributed to natural sediment loading (glacial runoff) and highly mineralized geology of the watershed.

The 2020 aquatic biology monitoring results do not indicate that adverse environmental effects were occurring seasonally related to slightly elevated concentrations. The aquatic invertebrate community at and downstream of the mine site indicated suitable water quality in streams (Golder, 2021).

Baseline water quality samples exhibit elevated metal concentrations from previous mine operations; however, only zinc (total and dissolved) exhibited occasional BC Water Quality Guideline exceedances downstream on Ketchum Creek, which were not observed at upstream sites. Water quality in the Unuk River is above guidelines seasonally for 12 parameters (i.e. total metals) during the high flow glacial melt when suspended sediments are particularly high in concentration, while copper is the only dissolved metal that exceeds guidelines. Unuk River metal concentrations are consistently elevated above water quality guidelines at both upstream reference (non-mine affected) and locations downstream of the Ketchum Creek/Unuk River confluence for reasons mentioned above, which indicates potential influence of the water quality from Ketchum or Eskay creeks may be limited.

20.1.4 Groundwater Quality and Quantity

The Eskay Creek deposit is located along the Eskay anticline, along the transition between Hazleton Group volcanic rocks and Bowser Lake group marine sedimentary rocks. The groundwater potentiometric surface generally mimics a subdued form of topography. Groundwater recharges by precipitation along ridges and flows toward the valleys where it discharges via seeps on valley slopes, or directly to streams. Part of the groundwater recharged over the footprint of the historic underground mine area is likely intercepted by the historical underground mine workings. Those workings serve, along with faults and other hydraulically active rock discontinuities, as preferential flow paths, carrying groundwater to discharge area further north, around Tom MacKay Creek.

The current condition of groundwater samples at all locations are typically neutral to slightly alkaline. Major ion chemistry indicates that groundwater is predominantly calcium bicarbonate and sodium bicarbonate with the exception of the Old Eskay Camp Area where groundwater is calcium bicarbonate and calcium sulphate. The marked increase in sulphate concentrations suggests that the monitoring wells at the Old Eskay Camp Area receive an input from nearby weathered mineralized (i.e., sulphides) zones around the historical underground mine, whereas groundwater at other locations reflect background carbonate-rich waters. Metals concentrations are variable, and do not exhibit clear spatial trends. Carbonate phases likely play an important role in buffering water pH, in turn limiting metal solubility. This suggests predominantly neutral drainage conditions in the vicinity of the former Eskay Creek underground Mine.

20.1.5 Aquatic Resources

The recent study for aquatic resources includes sediment quality, primary producers (periphyton), and secondary producers (benthic invertebrates). Comparisons of 2020 and historical data indicates that both reference sites (i.e. not influenced by former mine) and sites exposed to historical mining activity have frequently exceeded applicable guidelines for arsenic, cadmium, chromium, copper, iron, lead, manganese, mercury, nickel, selenium, silver, and zinc. These findings are consistent with supporting the trends in suspended sediments influence (Section 20.1.3). Benthic invertebrate tissue residue analysis indicated that selenium concentrations exceeded the BC tissue guideline at both the reference site (i.e. not influenced by former mine) and exposure site on the Unuk River situated downstream of the Ketchum/Unuk River confluence .

The aquatic resources study also includes: 1) biomass and diversity of periphyton biomass, and 2) abundance and diversity of benthic invertebrates.

20.1.6 Fisheries and Aquatic Resources

Waterfall barriers on Ketchum Creek and Eskay Creek immediately upstream of their confluences with the Unuk River prevent fish movement into these waterways and to tributaries upstream near the Project area within the Tom MacKay Creek watershed (i.e., Tom MacKay Creek and Argillite Creek). The recent fish and aquatic resource studies support the historical studies that the waterways adjacent to the historic mine and proposed Project; including Tom MacKay Creek, Argillite Creek, Ketchum Creek and Eskay Creek, TMSF, ASF, Little Tom MacKay Lake are non-fish-bearing. The Unuk River is fish-bearing in stream reaches adjacent to the confluence of Ketchum Creek and the Unuk River, with Dolly Varden observed approximately 7.7 km upstream of the Ketchum Creek influence, and salmon species situated downstream several kilometers from the Ketchum/Unuk River confluence.

20.1.7 Soils

Soils in the Project area are variable, typically of moderate depth developed on glacial deposits and loose sediments with smaller areas of floodplain and organic soils. Terrain is ridged and varies in slope steepness, soil drainage and depth over relatively short ground distances. Soils are very strongly to slightly acid and low in salinity. Analysis was carried out for 35 metal elements. Eleven of them, antimony, arsenic, chromium, cobalt, copper, lead, molybdenum, nickel, selenium, thallium and zinc exceeded at least one environmental standard threshold which is similar to other regional mine sites and may be attributed to natural geological conditions or historical mining, but further study is required. Soil studies also measured soil properties such as pH, salinity, texture, and organic carbon to assist in determining the availability and suitability of salvageable soil for use in reclamation planning.

20.1.8 Geohazards, Terrain Stability and Soil Erosion

Geohazards (landslides) include (most common to least common): rockfall, debris flow, debris slide, tension cracks, lateral spread, slump in surficial material and slump in bedrock. Nineteen percent (19%) of the Project footprint is potentially exposed to the risk of one or more geohazard(s). Support facility design should consider these features.

Terrain stability (TS) classes indicate the likelihood of slope instability resulting from resource development activities that occur in the upper few metres of the land surface within in-situ surficial materials and bedrock. Within the Project area, 28% of the area is classified as potentially unstable and unstable (TS Classes IV and V).

Soil erosion potential refers to the removal of surficial material, particle by particle, by rain splash and the tractive force of surface runoff. Due to the abundance of coverage of glacially-scoured landscape, bedrock outcrops, thin discontinuous surficial materials and blocky colluvium (terrain that is generally of low erodibility) within the Project area, the area rated with high and very high soil erosion potential account for only 6.4% of the footprint.

20.1.9 Vegetation and Ecosystems

The Project is accessed by the Eskay Creek Mine Access Road which begins near the Iskut River in the mostly forested Interior Cedar Hemlock (ICH), and parallels Volcano Creek up to the Mine Site (i.e. increasing elevation climbing up to the Prout Plateau) in the Engelmann Spruce Subalpine Fir (ESSF) and Mountain Hemlock (MH) Biogeoclimatic (BGC) units. The Volcano Creek valley that the road runs through is comprised of steep upper slopes, dominated by avalanche tracks and talus, whereas at the former mine site the terrain plateaus out. The lower portion of the Volcano Creek valley slopes have deeper soils which support forests dominated by subalpine fir, western hemlock, and Engelmann spruce on productive sites. The forests are primarily mature and old structural stages with a dense understory of blueberry on mesic sites and devil's club on rich, moisture receiving sites. Much of the area around the mine site is above 1,000 m in elevation, different than the access road, and is parkland ecosystems with krummholz, heathers, alpine meadows, and patchy tree islands which are a mix of mountain hemlock and subalpine fir trees.

The key results from the vegetation and ecosystems studies include:

- Riparian floodplain ecosystems are infrequent in the Project area and are located adjacent to larger creeks and rivers (e.g. lower Volcano Creek, Iskut River, Unuk River) and support unique vegetation communities that are maintained by fluctuating water tables and intermittent flood events.
- Wetlands account for 1.2% of the project area and immediately surrounding area. Most of the eleven wetland units detected are fens, one of these was a blue-listed wetland site unit.
- Ten exotic and invasive plants were registered in the region around the Project.
- Nineteen culturally valued plants were found in the Project area and region surrounding the Project. Thirty-seven combinations of BGC Unit and Site Unit presented culturally valued plants.
- No rare species were detected in the Project area.

20.1.10 Wildlife

The wildlife studies included a literature review of Tahltan Knowledge to determine which species are of cultural interest to the Tahltan. 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). Wildlife information was updated in 2020 with additional studies.

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).

The key wildlife study results include:

- The 2020 moose surveys observed 65% fewer moose than the 2005 and 2009 surveys. The decrease in observed animals may represent a population decline, differences in sightability of animals between surveys due to poor weather, observers or equipment, or a combination of factors. Recent Ministry of Forests, Land, Natural Resource Operations and Rural Development (BC MFLNRORD) moose surveys in the Skeena region indicated a 19% decline in moose density from 2013 to 2019 (BC MFLNRORD 2019a).
- Aerial surveys for mountain goat were flown during the winter and summer of 2020. For areas surveyed in both winter 2020 and winter 2009, there were three times as many goats observed in the winter of 2020 compared to the winter of 2009. This magnitude of the change likely indicates an increase in the goat population in these units between 2009 and 2020. These areas would not be exposed to the proposed Project.

- Hoary marmots are present throughout high elevations in the Project area, although habitat appears to be limiting colony size. Only two study areas, west of the Tom MacKay Storage Facility (TMSF) and planned Project development area, contained areas of connected habitat capable of supporting multiple larger colonies.

- The Project has very few areas with suitable bat habitat (low elevation mature forest and wetlands). The little brown myotis is federally listed as Endangered on Schedule 1 of the Species at Risk Act and was detected with a high degree of confidence (SARA; Government of Canada 2021a). The northern long-eared myotis (Myotis septentrionalis) is Blue-listed in BC (Special Concern) and listed as Endangered on Schedule 1 of SARA (ECCC 2018; BC CDC 2021) but was detected with a low level of confidence. Further studies would be required to confirm its presence.

- Six raptor species were recorded during the 2020 and 2021 surveys. Swainson's hawk (Buteo swainsoni is the only species of conservation concern detected whereas Northern goshawks (Accipiter gentiles) were not detected (both species are Red listed in BC; BC CDC 2021).

- A total of 36 species representing seven waterbird groups were detected. The central area of the Project area around the TMSF contains a variety of mid-elevation ponds which support breeding waterbirds. Three species of conservation concern are present: harlequin duck (Histronicus histronicus), long-tailed duck (Clangula hyemalis), and surf scoter (Melanitta perspicillata). Harlequin ducks are provincially ranked as Vulnerable during the non-breeding season, and both the long-tailed duck and surf scoter are Blue-listed as a species of Special Concern in BC (BC CDC 2021).

- A total of 60 species of upland breeding birds (migratory songbirds) birds were detected. Three species of conservation concern were detected in the local study area (LSA) or in the region around the Project: barn swallow (Hirundo rustica), common nighthawk (Chordeiles minor), and olive-sided flycatcher (Contopus cooperi). All three species are listed as Threatened on Schedule 1 of SARA (Government of Canada 2021a); barn swallows and olive-sided flycatchers are also Blue-listed in BC (BC CDC 2021).

- Western toad (Anaxyrus boreas) is listed on federal conservation rankings (Schedule 1, Government of Canada 2021) but provincially secure (Yellow, Government of BC 2021). Surveys identified a few western toad breeding sites (although not under Project footprint) with the majority of sites found at high elevation, deep ponds in fens, or had high water flow features associated with rivers. Columbia spotted frog (Rana luteiventris) was also confirmed to be on the Project area, but this species is not considered to be of conservation concern.

20.1.11 Social, Economics, and Land and Resource Use

The Project is located in the Regional District of Kitimat-Stikine (RDKS), the second largest regional district in BC spans a land area over 100,000 km² inhabited by nearly 40,000 to 45,000 people. Approximately one-third are Indigenous, which is higher than the provincial average (MSBED, 2005). Within the RDKS are six electoral areas and five municipalities, including Terrace, Kitimat, and Stewart, the Hazeltons, and Dease Lake. 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 separated from one another by lengthy roads as well as from the main regional centre of Terrace.

The RDKS provides services to residents including, waste disposal and recycling, fire protection and emergency support, transit, land use planning and bylaws, parks and recreation, and library services among others. Regional land and resource use includes: parks and recreational areas, recreational and commercial fisheries, mining, forestry, outdoor tourism and recreation (fishing, hunting, camping, hiking, snowmobiling, all-terrain vehicle (ATV) riding and skiing) and hunting (subsistence hunting, recreational hunting and commercially guided) and trapping (registered traplines). In the vicinity of the Project, there are mineral, water and range tenures.

There are no federal, provincial, or regional parks, wilderness or conservancy areas, ecological reserves, or recreational areas near enough to the Eskay Creek Project to be affected by the mining activities.

Over the Project life, direct employment would be an estimated 3,800 person-years, (214 hourly, 80 salary excluding contractors/consultants), in addition to indirect employment for workers in supplier industries and in businesses benefiting from workers spending their income locally, regionally and provincially. The community and socio-economic impacts of the Project can therefore potentially be favourable for the region, as new long-term opportunities are created for local and regional workers, as well as local/northern businesses and contractors.

Indigenous nations in the Project region include: Tahltan Nation, Tsetsaut Skii km Lax Ha Nation (TSKLH), Nisga'a Nation (a Treaty Nation), and Gitanyow Nation. More recently, TSKLH has produced maps indicating that the Project falls within their area of traditional land use (Rescan, 2009). Ongoing engagement with Indigenous Nations and governments helps to inform Skeena Resources of their interests and concerns and how to incorporate solutions and approaches into the Project assessment and design.

20.1.12 Tahltan Social, Economics, Land and Resource Use

The Project is located within the traditional territory of the Tahltan Nation and the asserted traditional territory of the Tsetsaut Skii km Lax Ha. 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.

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). There are seasonal use cabins along the Eskay Mine Road.

An extensive study of the Tahltan-specific socio-economic conditions within Tahltan Territory has been completed in 2021. The study provides the Tahltan Central Government and Nation, Skeena Resources and study partners with an updated characterization of their economic status, social conditions, current access to land and cultural supports, health, educational outcomes, interests, concerns and current access to land and cultural supports and land uses in support of community health.

20.1.13 Tahltan Country Foods

The Tahltan value many of the country foods harvested in the region for Food, Social and Ceremonial (FSC) purposes, as well as for medicine. The quality and availability of country foods is directly connected to the quality and condition of the environment and habitat in which they are living and growing. A country foods study identifies the animals harvested including: moose, caribou, bear, fish (salmon, trout, Dolly Varden, Burbot, Mountain Whitefish, and Lake Whitefish), and game birds (geese, ducks, grouse and ptarmigan). Plants harvested includes berries, fungi, trees, and other food and medicinal plants.

20.1.14 Archaeology

The Project area was subject to previous disturbances over the past century relating to exploration activities and the development of the previously operating Eskay Creek Mine. A single previously recorded archaeological site, HdTo-6, is located within the Project area. Site avoidance is the preferred mitigation measure. Under HCA Heritage Inspection

Permit 2018-0208 and HCA Permit 2020 0195, no new archaeological sites have been identified in the Project area. An Archaeological Chance Find Procedure will be applied prior to commencement of ground altering activities.

20.1.15 Paleontology

Clusters of documented fossil sites are present within or near the sedimentary and volcanic rock deposits in the Project area. The ages of these range from Lower Jurassic to Lower Cretaceous. There is a chance for Project activities to impact fossil resources. A Chance Find Protocol for Paleontological Resources has been developed.

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, approximately 0.41 Mm^3 of tailings (i.e., about 11% of the existing waterbody volume) 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 down a small tributary 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 and minimal waste generation.

20.2.2 Waste Management – Waste Rock and Tailings Disposal

The following sections are a summary of the current understanding of the geochemical characteristics of geological materials for the project and the implications for management of wastes to limit potential for water quality effects.

20.2.2.1 Geochemical Setting

Eskay Creek is classified as a high-grade, precious metals-rich volcanogenic massive sulphide (VMS) deposit. 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 (BLG). The immediate host rocks for the mineralization are rhyolite and mudstone which are overlain by sedimentary rocks intruded by andesite sills. This hanging wall package is volumetrically dominated by andesite and several stratigraphically continuous mudstone layers have been recognized.

The BLG occurs at low volumes relative to other rock types in the planned open pit but is a significant unit at planned locations of infrastructure such as the process plant and haul road to the Tom Mackay Storage Facility (TMSF). In the project area, the BLG consists of a conglomerate overlain by a package of interbedded mudstones, siltstones and sandstones (undifferentiated sediments).

The mineralization itself contains a wide variety of minerals including sulphides (pyrite, sphalerite, galena, chalcopyrite, stibnite, realgar, arsenopyrite), sulphosalts (tetrahedrite, boulangerite, bournonite) and sulphates (gypsum and barite). The carbonate minerals ankerite, calcite and dolomite are present but are relatively unimportant compared to sulphide minerals. The ore itself is therefore considered to be potentially acid generating and has potential for leaching of several constituents of interest (COIs) including at least antimony, arsenic, cadmium, copper, lead and zinc. Sulphate leaching is expected.

The hanging wall rocks have simpler mineralogy which is limited to pyrite, calcite and dolomite. The relative quantities of these minerals are variable, but sulphide content generally decreases with increasing distance from the mineralization. The BLG also has variable sulphide and carbonate content.

20.2.2.2 Datasets

Geochemical characterization data have been obtained by both Barrick and Skeena Resources. Barrick's data collection was focused on the rock types associated directly with the ore and exposed by underground mining and ground support fill sourced in part from gravels borrowed from the Iskut River. The data include bulk chemical and mineralogical characteristics, data from laboratory weathering tests (humidity cells and subaqueous columns), drainage chemistry for a temporary waste rock stockpile, and underground water drainage chemistry. Research studies have also evaluated chemical reactions occurring in sub-aqueously deposited tailings.

Skeena's dataset includes bulk chemical and mineralogical characteristics of diamond drill core obtained by Skeena from the planned open pit and infrastructure sites, laboratory and field weathering tests (humidity cells, barrel tests and subaqueous columns). Legacy tailings in the TMSF and overburden have also been characterized by Skeena.

20.2.2.3 Geochemical Classification Criteria

Classification of acid rock drainage (ARD) potential is based on the ratio of acid neutralization potential to acid generation potential (NP/AP). The preferred mineralogical source of NP is carbonate associated with calcium and magnesium though standard NP measurement methods may include minerals that are weak sources of NP under field conditions such as silicates and iron carbonates. Acid potential is calculated from sulphur associated with sulphide which is calculated from total sulphur less sulphate associated with non-acid generating sulphates such as gypsum and barite.

Interpretation of NP/AP is as follows:

If the mineralogical form of NP is not well understood, non-PAG (low potential for acid generation) materials are defined by NP/AP>3. If mineralogy data have been used to refine the calculation of NP, non-PAG is defined as NP/AP>2.

It is conventional to define PAG materials as NP/AP≤1 with materials having NP/AP between 1 and 2 or 3 as having uncertain potential for ARD. However, from a regulatory perspective PAG is defined as NP/AP≤2 or 3 unless a site-specific NP/AP criterion has been defined. A component of material classified as "uncertain" may therefore be re-classified as non-PAG which represents an opportunity for the project to reduce the volume of PAG materials.

Materials with less than 0.1% sulphide sulphur are classified as non-PAG regardless of the NP/AP, on the basis that low rates of acidity formed by sulphide oxidation will be buffered by release of alkalinity from silicate dissolution.

Metal leaching potential is primarily controlled by the ARD potential because many elements leach much more rapidly at lower pHs (for example, aluminum, copper and iron). However, some elements, (including antimony, arsenic, molybdenum, selenium and zinc) can leach at environmentally significant rates under non-acidic conditions. Metal leaching potential is evaluated by estimating enrichment relative to average crustal abundance, and by interpreting results of leaching and weathering tests performed on waste materials.

20.2.2.4 Geochemical Characteristics of Waste Rock

Rhyolite and mudstone waste from adjacent to the ore zones is generally classified as PAG due to relatively elevated sulphur content compared to carbonate mineral content. Acid generation has been observed to occur in laboratory tests and at full scale. Depending on the specific characteristics of the waste rock, acid generation may occur immediately or may take years to appear as carbonate minerals are depleted. The test work has confirmed the expectation for leaching of the COIs indicated above and sulphate both before and after onset of acid generation. Although not present as a discrete mineral, leaching of selenium was shown by test work.

In the hanging wall units, acid rock drainage potential is associated with sedimentary units and andesite where it is in contact with mudstone. However, this potential decreases with distance from the mineralization resulting in higher sedimentary units being classified as uncertain or non-PAG. Bulk chemistry data show that arsenic is enriched throughout the hanging wall package compared to average crustal abundance for these rock types. Kinetic test work has confirmed acid generation can occur in PAG hangingwall sedimentary units and leaching of antimony, arsenic and selenium occurs prior to the onset of acid generation.

20.2.2.5 Geochemical Characteristics of Construction Materials

BLG rocks occurring in infrastructure areas including quarries along the access road vary from PAG to uncertain and non-PAG. This is consistent with regional experience of BLG at other mining and infrastructure projects. ARD potential is lowest for the conglomerate and highest for the undifferentiated sediments. Non-acidic leaching of selenium has been shown by laboratory weathering tests.

20.2.2.6 Geochemical Characteristics of Tailings

Processing of ore will include initial rougher flotation to yield a bulk sulphide concentrate followed by cleaning of the concentrate to yield a gold concentrate which is shipped off-site for refining. Combined tailings from the rougher flotation and cleaning will be discharged for subaqueous disposal in the TMSF.

Geochemical testing is being performed on samples of simulated tailings obtained by metallurgical testing of ore samples. The test work includes bulk chemical and mineralogical characterization of separate rougher and cleaner tailings, and the combined final waste product.

Pyrite is the dominant sulphide mineral in both legacy and metallurgical test work tailings. Other sulphides include chalcopyrite and sphalerite with lesser galena, arsenopyrite and realgar. Other sulphides were generally not detectable. Consistent with the ore, the carbonate minerals, ankerite, calcite and dolomite are present in the ore. Barite is present.

Generally, rougher tailings are classified as non-PAG and cleaner tailings are uncertain or PAG. As a result, combined tailings are variably uncertain to non-PAG. Legacy tailings were mostly classified as uncertain or PAG.

Under the planned subaqueous disposal conditions in the TMSF, the potential for acid generation is effectively eliminated. However, samples of porewater collected from legacy tailings and subaqueous column tests performed on simulated tailings and PAG waste rock samples show that antimony leaching occurs under these conditions. The mechanism causing leaching under subaqueous conditions is believed to be due to galvanic interactions between sulphide mineral grains of different rest potentials.

20.2.2.7 Conclusions

- The following are conclusions on the geochemical characteristics of mines wastes at the project and application to waste management.
- Waste rock close to the mineralization (i.e., rhyolite and mudstone) is classified as dominantly PAG. Acid rock drainage may appear weeks to years after exposure to atmospheric weathering conditions depending on the specific geochemical characteristics.
- The package of interlayered mudstone and andesite in the hanging wall has mixed ARD potential classification of PAG, uncertain and non-PAG. ARD potential decreases with distance from the mineralization so that the uppermost part of the package is non-PAG.
- Bowser Lake Group rocks that occur in the planned open pit and at project infrastructure locations including the ore processing plant and haul road varies from PAG to non-PAG which is partly controlled by rock type. Coarser-grained strata (conglomerate) tend to be non-PAG though not exclusively so. Finer-grained strata (undifferentiated sediment package) tend to be PAG.
- Bulk (i.e., combined rough and cleaner) tailings are currently classified as PAG to uncertain.
- Potential for COI leaching under non-acidic conditions is a consideration for all waste materials expected to be generated by mining, ore processing and infrastructure construction. The primary elements of concern are antimony, arsenic and selenium. All sulphide-bearing materials at the site will leach sulphate to varying degrees
- The project will involve disposal of PAG waste rock and all tailings under subaqueous conditions at the TMSF. This approach will limit oxidation of sulphide minerals and prevent the onset acid generation. However, test work shows that antimony leaching occurs under subaqueous conditions.
- Segregated non-PAG waste rock will be disposed in a waste rock facility outside the open pit and as backfill into the open pit as allowed by the mining schedule. The segregated waste rock is not expected to generate acid though leaching of some elements under non-acidic condition is expected.
- Pit walls remaining exposed to long-term weathering will have PAG, uncertain and non-PAG components which can be expected to leach metals into the final pit lake to varying degrees.
- Excavations for infrastructure will need to consider the potential for acid generation and metal leaching in fill materials and exposed surfaces.

20.2.2.8 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.

20.2.2.9 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 Surface Water Management

The objective of surface water management is to protect groundwater and surface water resources. Feasibility Study infrastructure and upstream catchments for the Project were delineated based on topography data and footprints of facilities provided. The total on-site infrastructure area, excluding roads, diversion channels, collection ditches and sedimentation pond areas, is estimated to be range from 49 ha to 487 ha over the life of mine. The site water management has been designed in accordance with BC regulations.

Contact and non-contact water are managed separately for the Project. Contact water is captured and transported in collection ditches and pipelines to sediment ponds, sumps, and contact water ponds. For roads, runoff will be captured in collection ditches and conveyed to sediment ponds, to remove greater than 10 microns particles prior to discharging into the environment. Contact water from the open pits, WRSF, Ore Stockpile, Process Plant Pad will be capture in collection ditches and conveyed to Sumps, Ponds 5 and 6. All runoff collected in the sumps and Pond 6 will be pumped to Pond 5. All water from Pond 5 will be pumped to the process plant and used in mining operations or pumped with the tailings to TMSF. Currently, there are no diversion channels, collection ditches, or water treatment facility for the subaqueous deposition of the PAG waste rock and tailings in TMSF. Non-contact water is diverted around other mine infrastructure, where possible, through diversion channels, culverts, and creek crossings.

The SWMP outlines Skeena's strategies to responsibly manage surface water during operational activities currently projected over a nine (9) year period and post closure. Included in the SWMP are the following facilities:

- North and South Open Pits;
- NAG WRSF and PAG Stockpile Haul Road;
- TMSF Haul Road;
- Reclaim Barge Access Road;

- Tailings Discharge Light Vehicle Road;
- Pond 5 Access Road;
- Pond 6 Access Road;
- Pond 6 Pipeline Access Corridor;
- Process Plant Pad and Laydown Access Road;
- Crushing Area and Conveyor Corridor;
- Explosives Magazine Access Road;
- NAG Waste Rock Storage Facility (NAG WRSF);
- Ore Stockpile;
- Topsoil Stockpiles;
- Process Plant Pad and Laydown;
- Explosive Magazine;
- Explosive Bulk Storage;
- Construction and Demolition Waste Landfill;
- Tom MacKay Storage Facility (TMSF);
- Pond 5;
- Pond 6;
- Sedimentation Ponds; and
- Other Contact Water Management Structures.

The following stormwater design parameters were developed for the project infrastructure surface water management structures, in alignment with British Columbia Regulations (Refer to Table 20-2).

Table 20-2: Design Parameters for Surface Water Management Structures

Design Parameter	Unit	Value
Collection Ponds		
Storm Event Captured	Return Period	24 Hour - 100 Year Event[1]
Particle Size Captured	µm	all
Freeboard	m	0.5
Sedimentation Ponds		
Storm Event Captured	Return Period	24 Hour - 10 Year Event
Particle Size Captured	µm	>10
Storm Event Survived	Return Period	24 Hour - 200 Year Event
Particle Size Captured	µm	n.a.
Freeboard	m	0.5
Collection Ditches		
Storm Event Conveyed	Return Period	24 Hour - 100 Year Event
Freeboard	m	0.3
Storm Event Survived	Return Period	24 Hour - 200 Year Event
Freeboard	m	Contained Within Channel Limits
Diversion Channels		
Storm Event Conveyed	Return Period	24 Hour - 100 Year Event
Freeboard	m	0.3
Storm Event Survived	Return Period	24 Hour - 200 Year Event
Freeboard	m	Contained within Channel Limits
Culverts		
Storm Event Conveyed	Return Period	24 Hour - 100 Year Event
Silt Fence and Hay Bales		
Particle Size Captured	µm	>10

Note: 1. Capture the storm event for collection ponds includes pumping during storm event.

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. HEC-RAS computer model was utilized to develop the runoff from the WDW and the watershed above the WDW for average year, WDW (contact water) for 1:100 year storm event with 0.3 m freeboard and convey within the limits of the water management structure the 1:200 year storm event and pass the 1:2,475 year storm event for non-contact water upstream of the WDW through a rock drain below the facility which discharges into the lower section of the exposed portion of Argillite Creek.

The contact surface water ditches are design to convey up to the 1:100 year storm event and within the limits of the water management structure the 1:200 year storm event to the Contact Water Pond 5 (Pond 5). These ditches consist of a series 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 and replaced with new temporary ditches. The compacted consists of compacted subgrade, riprap, and grouted riprap in high velocity zones.

The Pond 5 is constrained by the open pit, haul road and stockpile and has a design storage capacity of approximately 49k m^3. The pond is designed to remove any coarse grain suspended sediments transported from surface runoff before pumping to the process plant for water makeup and any excess water will be transport with the tailings to TMSF. The pond has the capacity to contain the 1:100 year storm event for the WDW, Open Pits, and Process Plant Pond 6 (Pond 6)) using the Open Pits, and Pond 6 as temporary detention facilities and pumping to the process plant and TMSF at the Pond 5 maximum pumping capability during the design event. The retention time, for the average annual runoff, 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 1:2,475-year storm event is approximately 6.2 m^3/s.

The rock drain has been designed with a factor of safety of two for the predicted flow rate for the 1:2,475 year storm event. The rock drain considers end dumping waste rock from 20 m above the toe of the working face. As the material travels down the slope, where it will self-segregate, smaller particle at the crest of the slope and larger boulders at the toe of the slope, creating a rock drain along the bottom of the slope. In addition, at the 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 downstream toe of the WDW, the rock drain will discharge into lower section of Argillite Creek that flows into Tom MacKay Creek. A small coffer dam will be constructed just downstream of the confluence with Argillite Creek and Tom MacKay Creek that will redirect the water from Argillite Creek into the tunnel diverting water around the North open Pit back into Tom MacKay Creek.

The Main Pit will be mined during construction and operation of the Project. Any water arriving in Pit bottom (intercepted Precipitation or influent seepage) is contact water. A portion of the water will be lost from the Pit via evaporation. In addition, the open will have a dewater program to depressurize the pit walls. Both surface water and water from the dewater program will be captured in the bottom of the pit and pumped to Pond 5. The pit will also act as a detention facility during storm events greater than or equal to the 1:100 year design storm event. The south pit due to it location, water collection in the bottom the pit will be pumped up to the WDW eastern collection ditch and convey with surface runoff from the WDW.

There are no non-contact diversion channels for TMSF due to the steep terrain. The facility is design to capture and pass the Probable Maximum Flood (PMF) through the penstock or the closure spillway.

The Process Plant Pad and Laydown (PPP&L) are located on the west side of the WDW. For the feasibility study, the pad will contain an Ore stockpile, process plant, fuel storage and dispensing, and a laydown area. The platform is approximately 7.8 ha and situated mostly on PAG Bedrock. However, the PAG bedrock will be covered with a layer of NAG rock. The platform will drain both to the west and to the east side where collection ditches lined with geomembrane and riprap will capture surface runoff and convey it to Pond 6. Pond 6 is design to contain the 1:100-year storm event and safely pass the 1:200-year storm event to Tom MacKay Creek.

Ore shall be placed in multiple ore stockpiles for grade control and plant recovery. This facility occupies an area of 6.4 ha and is located north of the north end of the WRSF. The area will be used to store Ore that will be crushed and sent to the process plan. The ore stockpile has a capacity to store approximately 1Mt of ore. The Ore Stockpile pad will be lined with a geomembrane and an overliner (crushed rock) to collect any seepage from the piles since the material is potential acid generating and metal leaching. Runoff from this facility will be conveyed in geomembrane lined collection ditches to Pond 5.

The crushing area and conveyor corridor water management consists of a diversion ditch conveying runoff to the crusher area sump. The sump will pump the runoff directly to the ore stockpile collection ditch.

Ponds 5 and 6 are both lined with geomembrane since they are receiving contact water from the process plant area, WDW, ore stockpile, and open pits. Each pond is designed to contain the 1:100-year storm event with 0.5 m of freeboard and pass the 1:200-year storm event.

The mine site roads consist of hauls, and infrastructure support roads. The collection ditches are located along these roads and are designed to convey up to the 1:100-year storm event with 0.3 m of freeboard and pass the 1:200-year storm event within the limits of the channel. The collection ditches terminate into strategically location sedimentation pond that are designed to retain sediment above 10 microns from storm events equal to or less than the 24-hour 10-year storm event and the pond is also designed to pass the 1:200-year storm event with 0.5 m of freeboard. The water is released into the environment through a spillway into the surrounding environment.

There are three perennial stream crossings along haul roads: one (1) on Argillite Creek and two (2) on Tom MacKay Creek during the development of the technical sample. The crossings will be designed to pass a minimum of the 1:200-year storm event in accordance with BC MOTI (2019). The Argillite Creek crossing will be removed during the expansion of the WRSF during the revitalization program.

Numerous culverts will be installed along the haul roads, Reclaim Barge Access Road, and Tailings Discharge Light Vehicle Road at low spots on the uphill slope, ephemeral streams, and intermittent streams to prevent ponding of water above the roads. The culverts are sized based on the area above the culvert. The culverts are corrugated metal pipes or corrugated HDPE pipes. The inlets and outlet are protected with riprap aprons to prevent localized erosion. The culverts are designed to pass a 1:100-year storm event in accordance with BC MOTI (2019).

The Explosive Magazine is located next to Albino Lake. The surface water management consists of collection ditches are to convey the 1:100-year storm event with 0.3 m freeboard and pass the 1:200 year storm event within the limits of the channel to a sediment pond. The sediment pond is design to contain the 24-hour 10-year storm event and capture sediment equal to/or less than 10 microns prior to releasing water into the environment. In addition, the pond is design to pass the 1:200-year storm event with 0.5 m of freeboard.

The Explosive Bulk Storage is also located along Albino Lake. The surface water management consists of a pad for the bulk explosive storage is covered with road base. Collection ditches around the facility convey runoff to a sediment pond that is design to contain the 24-hour 10-year storm event and capture sediment equal to/or less than 10 microns prior to releasing water into the environment. In addition, the pond is design to pass the 1:200-year storm event with 0.5 m of freeboard.

As part of the development of the feasibility and technical sample facilities both organic materials and topsoil will be generated during construction. Nine (9) topsoil stockpiles (TSS 1 through 9) have been strategically placed around the Project to store organic materials and topsoil for later use in closure of these facilities. The combined area of the stockpiles is approximately 6.0 ha. Each topsoil stockpile is surrounded by a silt fence and hay bales to protect the environment until a vegetative cover is established over the TSSs.

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 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 land uses. A Closure and Reclamation Plan will be developed during the permitting process to achieve post-mining 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 revegetating decommissioned areas to perpetuate a long-term revegetated state; and
- 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 are governed by environmental legislation. 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 BC Environmental Assessment Act 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 government 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 undergo an Impact Assessment (IA) 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 provincial and federal EA processes can occur concurrently within a single process delivered by the BC Environmental Assessment Office. The purpose of these processes is to address 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, and to inform government's decision of additional requirements that may be a condition of project approval.

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 EA Certificates, one or both may be amended through a substituted EA/IA process or a new EAC issued and older certificates modified.

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 an 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 considered 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 off-site 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. Skeena Resources submitted a Detailed Project Description to the federal and provincial regulators and Taltan Central Government on August 11, 2022, to initiate the second phase (Readiness Decision) of the EA process.

No technical or policy issues have been identified that would prevent 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 and operations. No permits for project commercial development will be issued before an EAC is obtained. Consequently, Skeena will apply concurrently for permits within the environmental review process schedule for all permits. Strategies to expedite the permitting process and reduce the time to start construction are being examined.

20.5.3 Anticipated Federal Approvals and Authorizations

Table 20-3 presents a preliminary list of the key federal authorizations, licences, and permits required for project development.

Table 20-3: 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, landfill
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
Groundwater Well Registration and Groundwater Usage	ENV	*Water Sustainability Act*	Authorize storage, use or diversion of groundwater for one or more purposes

Authorization	Responsible Agency	Legislation	Purpose
(Section 2)			
Short Term Water Use (Section 10)	ENV	*Water Sustainability Act*	Authorize short -term storage, use or diversion of surface water or groundwater for one or more purposes
Water Licence (Section 9)	ENV	*Water Sustainability Act*	Authorize storage, use or diversion of surface water or groundwater for one or more beneficial 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)
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
Licence of Occupation (crown land)	FLNRORD	*Land Act*	Authorizes occupancy of crown land for approved purpose - for example offsite power line right of way or quarry.

Table 20-4: 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

Authorization	Responsible Agency	Legislation	Purpose
			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 Engagement and Consultation

20.6.1 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 implementing 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 serves as a guide for Skeena's engagement activities with identified Indigenous Nations and stakeholders throughout 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, federal and provincial government requirements.

20.6.2 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.

20.6.3 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 Capital Cost Estimates

21.1.1 Overview

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

The capital cost estimate summarized in Table 21-1 provides a summary of the Project capital cost estimate, with costs grouped into major scope areas as presented in Skeena's new release dated September 8, 2022 "Skeena Completes Robust Feasibility Study for Eskay Creek: After-Tax NPV (5%) of C$1.4B, 50% IRR and 1 Year Payback. Following this, additional tables are included to illustrate Project costs at detailed WBS level (WBS 1000 to 10000).

The costs are expressed in Q1 2022 Canadian dollars and include all costs related to the Eskay Creek Project (e.g., mining, site preparation, process plant, tailings facility, power infrastructure, camp, Owners' costs, spares, first fills, buildings, roadworks, and off-site infrastructure).

Table 21-1 provides a summary of the estimate for overall capital cost shown by WBS. The estimate includes costs for mining, site preparation, process plant, tailings facility, power infrastructure, camp, owners' costs, pares, first fills, buildings, roadworks, and the off-site infrastructure.

The estimate is based on an EPCM execution approach for the process/infrastructure areas, and an EPCM execution for the civil-earthworks camp and power infrastructure packages, as outlined in Section 24. The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations;
- There is no escalation added to the estimate from base date Q1 2022 forward;
- A growth allowance was included;
- Data for the estimates have been obtained from numerous sources, including:
    - mine schedules;
    - feasibility-level engineering design;
    - topographical information obtained from the site survey;
    - geotechnical investigations;
    - firm and budgetary equipment quotes from Canadian and international suppliers;

- budgetary unit costs from numerous local contractors for civil, concrete, steel, electrical, piping and mechanical works; and
- data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analysed. A contingency was applied under the Provisions section of the cost estimate and was based on ranging the accuracy of the data by discipline and WBS level 3 and applying a probabilistic method (Monte Carlo Simulation). An overall contingency amount was derived in this fashion.

As outlined in Table 21-1, the LOM capital cost of the project will be approximately C$911M which 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 3.0 Mt/a operation. The initial capital cost is estimated to be C$592M.
- Expansion and sustaining costs: include the capital cost required to expand the throughput to 3.7Mt/a operation and required to sustain operations, with the most significant component being open pit mine development. The expansion and sustaining costs are $180M 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 C$138M.

Table 21-1: Project Capital Cost Estimate with WBS Allocation

	Initial	Expansion & Sustaining	Closure	LOM Total	WBS Allocation
Mine					
Mine Development (C$M)	98	10	-	108	Includes WBS 1100 + WBS 1240
Mine Other (C$M)	19	9	-	28	Includes WBS 1300
Mine Equipment (C$M)	8	21	-	29	Includes WBS 1200 except for WBS 1240
Sub-Total Mine (C$M)	**125**	**40**	**-**	**166**	
Process Plant					
Processing (C$M)	178	32	-	210	Includes WBS 3000 & 4000
Earthworks (C$M)	19	2	-	21	Includes WBS 6100
Sub-Total Processing (C$M)	**197**	**34**	**-**	**231**	
Infrastructure					
Onsite Infrastructure (C$M)	69	65	-	134	Includes WBS 6000 except for WBS 6100

	Initial	Expansion & Sustaining	Closure	LOM Total	WBS Allocation
Offsite Infrastructure (C$M)	50	23	-	73	Includes WBS 5000 and 7000
Sub-Total Infrastructure (C$M)	**119**	**88**	**-**	**207**	
Total Directs (C$M)	**442**	**162**	**-**	**604**	
Indirects (C$M)	74	10	-	84	Includes WBS 9000
Total Directs + Indirects (C$M)	**516**	**171**			
Owner's Costs (C$M)	30	-	-	30	Includes WBS 8000
Total excluding Contingency (C$M)	**546**	**171**	**-**	**717**	
Project Contingency (C$M)	47	9	-	56	Includes WBS 10000
Sub-Total Including Contingency (C$M)	**592**	**180**	**-**	**773**	
Closure (C$M)	**-**	**-**	**138**	**138**	
Total (C$M)	**592**	**180**	**138**	**911**	

* Numbers above are rounded to the nearest integer, therefore some sub-totals may not balance due to rounding.

Table 21-2: Total Project Costs Summary by Major Discipline

WBS	Description	Initial Phase Total Cost (C$M)	Expansion Phase Total Cost (C$M)	Sustaining Total Cost (C$M)	Total Cost (C$M)	% of Total
B	Earthworks	28.6	0.0	29.9	58.5	7.6%
C	Concrete	18.0	2.0	2.0	22.0	2.8%
S	Structural Steelwork	18.3	9.3	0.0	27.6	3.6%
F	Platework	7.2	0.0	0.2	7.4	1.0%
M	Mechanical Equipment	101.8	10.9	4.1	116.8	15.1%
P	Piping	32.3	1.1	3.5	36.9	4.8%
E	Electrical Equipment	24.4	2.4	0.2	26.0	3.4%
L	Electrical Bulks	18.3	3.5	0.9	22.7	2.9%
I	Instrumentation	2.7	0.1	0.0	2.7	0.4%
A	Architectural	31.0	1.6	17.7	50.3	6.5%
N	Mobile Equipment	1.8	0.0	3.3	5.1	0.7%
R	Third Party Estimates	158.0	0.0	66.9	224.9	29.1%
U	Field Indirects	29.8	0.0	2.0	31.8	4.1%

WBS	Description	Initial Phase Total Cost (C$M)	Expansion Phase Total Cost (C$M)	Sustaining Total Cost (C$M)	Total Cost (C$M)	% of Total
V	Other (Spares, Fills, Vendors)	7.3	0.6	2.9	10.8	1.4%
T	Project Delivery	36.5	6.0	0.02	42.5	5.5%
O	Owner's Costs	29.3	0.0	0.0	29.3	3.8%
Y	Provisions	46.4	2.1	6.7	55.3	7.2%
	Project Total (less Closure)	**591.7**	**39.7**	**140.4**	**771.8**	**100.0%**

21.1.2 Mining (WBS 1000)

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

- pre-production stripping costs
- mining equipment capital
- mine infrastructure capital.

The cost breakdown for Area 1000 – Mining is shown in Table 21-3.

Table 21-3: Mining Capital Cost Estimate (C$M)

Mining Capital Category	Initial Cost ($M)	Sustaining Cost ($M)	Total Capital Cost ($M)
Pre-production stripping (WBS 1100)	83.3	-	83.3
Mine equipment capital (WBS 1200)	16.9	20.9	37.8
Mine Infrastructure (WBS 1300)	6.6	5.8	12.4
Total	**106.8**	**26.7**	**133.5**

This breakdown covers specific mining areas and is a portion of WBS 1000 in Table 21-1.

21.1.2.1 Pre-Production Stripping (WBS 1100)

Mining activity commences in advance of the process plant achieving commercial production. This includes the movement of 12.2 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 $83.3 M. This cost covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs, and financing costs.

21.1.2.2 Mining Equipment (WBS 1200)

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).

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 21-4.

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 for the various locations around the mine. This includes such things as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc. For Eskay Creek, significant 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, nineteen (19) 144-t units and five (5) 91 t trucks are necessary to maintain mine production. This happens from Year 3 onwards. It should be noted that three different truck fleets are included in the estimate. The first equipment on site preparing the mine for full production includes four articulated trucks with a capacity of 40 t being loaded by two hydraulic excavators (6.7 m^3) together with two small boom drills (165 mm). The early works/pre-stripping fleet then transitions to five (5) 91 t units with the appropriately sized loading fleet (11.5 m^3 loaders) and uses the same drills (165 mm). When the mine starts production (Year 1) the transition to the larger 144-t trucks will occur with hydraulic shovels (22 m^3) and the larger drills (165 mm). The smaller trucks will act as additional stripping capacity when required, snow removal and tailings dam maintenance duties. The earlier loading fleet will be used in the pit but also 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 21-4: Major Mine Equipment – Capital Cost, Full Finance Cost and Down Payment

Equipment	Unit	Capacity	Capital Cost (C$M)	Full Finance Cost (C$M)	Down Payment (C$M)
Production drill	mm	165	1.5	1.7	0.3
Production drill – electric	mm	165	3.6	4.0	0.7
Production/crusher loader	m^3	11.5	2.6	2.9	0.5
Hydraulic shovel	m^3	22	8.3	9.4	1.7
Haulage truck	t	40	1.1	1.3	0.2
Haulage truck	t	91	1.9	2.1	0.4
Haulage truck	t	144	3.1	3.5	0.6
Dragline (44 m boom)	m^3	3.8	3.8	4.3	0.8
Track dozer	kW	474	1.7	1.9	0.3
Grader	kW	163	0.4	0.4	0.1
Support excavator	m^3	6.7	2.2	2.5	0.4

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 21-5.

Table 21-5: 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
Production drill (165 mm)	1	2	2	2	3	3	3	3	3	2	2	2
Electric drill (165 mm)	-	-	2	2	2	2	2	2	2	2	2	2
Production loader (11.5m^3)	-	1	2	3	4	4	4	4	4	3	3	2
Hydraulic shovel	-	-	-	2	2	2	2	2	2	2	2	2
Haulage truck (40 t)	4	4	4	4	4	4	4	4	4	4	4	4
Haulage truck (91 t)	-	3	5	5	5	5	5	5	5	5	5	5
Haulage truck (144 t)	-	-	-	8	16	19	19	19	19	19	19	19
Track dozer	4	4	4	5	5	5	5	5	5	5	5	5
Grader	2	2	2	3	3	3	3	3	3	3	2	2
Support excavator	2	2	2	2	2	2	2	2	2	2	2	2
Snowplows	1	3	3	3	3	3	3	3	3	3	3	3
Snowblowers	-	1	2	2	2	2	2	2	2	2	2	2
Dragline	-	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 only expected to in Year -3 and Year -2 with the smaller truck fleet activity in preproduction work.

The expected equipment lives are:

- Production drill: 25,000 hours (165 mm)
- Production drill (electric): 45,000 hours (165 mm)
- Production loader: 35,000 hours (11.5 m^3)
- Hydraulic shovel: 72,000 hours
- Haul trucks: 25,000 hours (40 t), 35,000 hours (91 t), 50,000 hours (144 t)
- Dragline: 10 years
- Track dozer: 35,000 hours
- Grader: 25,000 hours
- 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.

21.1.2.3 Mine Infrastructure (WBS 1300)

The mine infrastructure capital includes various separate line items in the costing. These are shown in Table 21-6.

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.

An explosives bulk plant will be established prior to the mine production start. Preproduction explosives will be provided by temporary facilities, but the main plant will be constructed in Year -1 for use in the mine as production ramps up.

Pit area preparation will include the removal of merchantable timber, grubbing, and any topsoil removed and stockpiled.

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 C$1.2M, with the majority being the mining/geology software.

Electrified hydraulic shovels and drills will require a power line around the pit. The enclosed line is expected to be 6 km in length from the main substation. 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.

A mine truck shop/mine office, tire change and tire wash are included in the cost estimate and will service the Mining fleet

The Mine Facilities and services includes for Emergency station and the general building services for the truck shop (HVAC, grounding etc.).

Table 21-6: Mine Infrastructure Capital

Miscellaneous Mining Capital	Initial Cost (C$M)	Expansion phase (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Dewatering System – Pumps/Pipe	1.4	-	5.8	7.2
Explosives Bulk Plant	2.1	-	-	2.1
Pit Area – Clear and Grub	0.2	-	-	0.2
Mine Office Equipment	1.2	-	-	1.2
Pit Power and Communications	1.8	-	-	1.8
Mine Truck shop	11.5	-	2.7	14.2
Mine Facilities and Services	2.0	0.2	0.6	2.8
Total	20.2	0.2	9.1	29.5

21.1.3 Mine Waste Dump Storage Facility and Surface Water Management

Mine Waste Dump Storage facility — for description of the storage facility, refer to Section 18.4.

Water management pond and water contact channels — for description of the pond, refer to Section 25.10.4.

The following Table 21-7 shows the costs for the Mine Waste Dump Storage Facility and Surface Water Management.

Table 21-7: Mine Waste Dump Storage Facility and Surface Water Management Capital

Miscellaneous Mining Capital	Initial Cost (C$M)	Expansion Phase (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Mine Waste Dump Storage Facility	1.4	0.0	6.7	8.1
Water Management Pond	2.2	0.0	0.0	2.2
Water Contact Channels	1.0	0.0	3.1	4.1
Total Directs- Ausenco scope	**4.6**	**0.0**	**9.8**	**14.4**

21.1.4 Process Plant (WBS 2000 & 3000) & Site Infrastructure (4000 & 3000)

A summary of the Process Plant (WBS 2000 & 3000) and Site Infrastructure (WBS 4000) capital cost estimate is presented in Table 21-8.

Table 21-8: Capital Cost Estimate Summary – Process Plant & Site Infrastructure

WBS	Description (Processing)	Initial (C$M)	Expansion Phase Total Cost (C$M)	Sustaining Total Cost (C$M)	LOM Total Cost (C$M)
3100	Coarse Ore Crushing	15.9	8.4	0.2	24.5
3200	Ore Conveying	6.5	0.1	0.0	6.6
3400	Ore Reclaim	1.0	0.0	0.0	1.0
4100	Primary Grinding/Milling/Classification	61.5	13.2	2.2	76.9
4100	Separation/Concentrating	83.3	6.9	0.0	90.2
4300	Reagents	2.4	0.0	0.0	2.4
4400	Process Utilities	8.0	0.1	0.1	8.2
5100	Tailings Disposal	15.3	0.0	22.7	38.0
5200	Reclaim Water	5.0	0.0	0.0	5.0
6100	Site Preparation	19.2	0.0	1.6	20.8
6200	On-site Roads and Earthworks	14.5	0.0	38.1	52.6
6300	On-site non-process facilities	8.1	0.0	16.7	24.8
6400	On-site mobile equipment	1.8	0.0	3.3	5.1
6500	On-site Bulk storage	7.0	0.0	0.5	7.5
6600	On-site Other Utilities	18.2	0.0	3.6	21.8
6700	On-site communications	0.2	0.0	0.0	0.2
6800	On-site Power Supply and Transmission	18.9	2.0	0.0	20.9
	Total Processing (excluding contingency)	**286.8**	**30.7**	**88.9**	**406.4**

21.1.4.1 Estimate Sources

The process plant initial capital cost is C$286.8M, and includes provision for: ore handling; crushing, grinding and classification; concentration; product filtration and drying; reagents and process utilities; tailings and reclaim water; site preparation; onsite roads; onsite power transmission; and other onsite infrastructure. The process expansion phase capital is C$30.7M and includes for additional secondary crushing, additional ball mill, additional flotation capacity and additional regrinding. The Sustaining capital costs is C$88.9M and includes provision for pebble crusher and associated equipment, permanent operations camp, sewage treatment plant, incinerator, light vehicle fuel storage and distribution system, tailings pipeline.

All major processing equipment for all phases were sized based upon the process design criteria. Once the mechanical equipment list was outlined, the mechanical scopes of work were derived and sent to the market for firm and budgetary pricing by Canadian and international equipment suppliers (see Table 21-9 and Table 21-10). Once the price quotations were reviewed and integrated, in total 94% of the value of the mechanical equipment was sourced from firm and budgetary quotations, with the remainder of the minor process equipment pricing sourced from other recent Canadian gold projects and studies.

Table 21-9: Mechanical Equipment Supply Price Basis

Source	Initial Phase Supply (C$M)	Expansion (Year +5) Supply (C$M)	Sustaining Supply (C$M)	Total (*) Supply (C$M)
Firm quote	30.5	5.8	0	36.3
Budget Quotes	26.5	2.3	2.3	31.1
Database	3.7	0.3	0.3	4.3
Factored	0	0	0	0
Allowance	0	0	0.4	0.4
	60.7	**8.4**	**3.0**	**72.1**

*Note: costs exclude freight.

Table 21-10: Mechanical Equipment Supply Price Basis

Package No.	Equipment
P0001	Primary Crushing Station
P0002	Grinding Mills
P0003	Secondary Cone Crusher
P0004	Flotation Cells
P0005	Regrind / Secondary Mills
P0006	Conveyors
P0007	Cyclones
P0008	Concentrate Thickener
P0009	Concentrate Filter
P0010	Compressed Air Equipment
P0011	Barges/Pontoons (tailings deposition, reclaim water)
P0012	Potable Water Treatment
P0013	Sewage Treatment Plant
P0014	Pumps – Slurry and Other Process
P0017	Screens (vibrating, screw, trash)
P0018	Apron Feeders
P0019	Pebble Crusher
P0106	HVAC – design and supply
PTBD111	Agitators
PTBD112	Samplers / Analysers
PTBD113	Dust Collection systems
PTBD115	Air Dryers and Air Receivers
PTBD116	Feeders (dryer feeders, ball feeders, etc.)
PTBD120	Diesel Fuel Storage and Distribution

Package No.	Equipment
PTBD121	Lube Systems
P0204	Overhead Cranes / Hoists
P0206	Filtration Dryer
P0207	Incinerator
P0208	Vertical Turbine Pump Barge

Similar to the above, all major electrical equipment for all phases were sized based on the project equipment list. Once the electrical equipment list was outlined, scopes of work were derived and sent to the market for budgetary pricing by Canadian equipment suppliers, as outlined in Table 21-11 and Table 21-12. Once the budgetary quotations were reviewed and integrated, in total 91% of the value of the electrical equipment was sourced from budgetary quotations, with the remainder of the minor equipment pricing sourced from other recent Canadian gold projects and studies.

Table 21-11: Electrical Equipment Supply Price Basis

Source	Initial Phase Supply (C$M)	Expansion (Year +5) Supply (C$M)	Sustaining Supply (C$M)	Total (*) Supply (C$M)	% of Total
Tendered	0	0	0	0	0%
Budget Quotes	18.9	0.6	0.1	19.6	91%
Database	0.4	0.65	0	1.05	5%
Factored	0	0	0	0	0%
Allowance	0	0.75	0.1	0.85	4%
Total	**19.3**	**2.0**	**0.2**	**21.5**	**100%**

Table 21-12: Electrical Equipment Supply Price Basis

Package No.	Equipment
C0508	Main 69kV Substation and switchyard
C0505	On-site O/H power lines (13.8kv)
P0201	Transformers
P0202	Pre-Fab (5) Electrical Rooms, (1) process control room, (1) Crusher control room
P0202 (part of)	Expansion phase Electrical room
P0202 (part of)	13.8 kV / 4.16 / 600 Voltage Switchgear
P0202 (part of)	4.16 and 600V MCCs
P0202 (part of)	VFDs
P0202 (part of)	Remote I/O cabinets
P0202 (part of)	Harmonic Filter
P0203	Standby Diesel Generators

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design were completed to a feasibility study level of definition, allowing for the bulk material quantities (earthworks, concrete, structural steel, platework, piping, electrical and instrumentation bulks) to be derived for the major commodities, as outlined in Table 21-13.

Table 21-13: Material Commodity Codes

Commodity Code	Commodity Description
A	Architectural
B	Earthworks
C	Concrete
D	Mining
E	Electrical Equipment + Bulks
F	Platework
I	Instrumentation + Bulks
M	Mechanical Equipment
N	Plant & Ancillary Equipment
O	Mobile Equipment
P	Pipework
S	Structural Steel
U	Field Indirects
V	Third-Party Packages/Other
W	Project Delivery
Y	Owner's Costs
Z	Taxes & Duties

After the derivation of all the bulk material quantities, for the process plant and infrastructure areas, major construction contracts were formed, and tendered to experienced Canadian contractors for budgetary pricing bids, as per Table 21-14.

Table 21-14: Construction Contract Packages

Package No.	Equipment
C0501	Earthworks
C0502	Concrete Works
C0503	Steel, Mechanical, Platework and Piping works
C0504	Electrical and Instrumentation works
C0506	Pre-engineered Buildings
C0507A	Permanent Accommodation Camp
C0507B	Temporary Construction camp (incl. operations & messing)
C0509	Modular Buildings
P0103	Metallurgical Laboratory

The mobile equipment fleet (WBS 6400) is for the purposes of supporting the ongoing operations of the Process plant requirements. The fleet includes safety and maintenance vehicles as well as service equipment such as front-end loaders, cranes, bobcats and forklifts. Mostly all the mobile equipment has been based on a "lease-to-own" scenario, with pricing (10% down payment) based on informal vendor quotations, with the exceptions of the fire truck, CAT 908K front-end load and a pipe fusing machine for which full purchase up front has been included in the cost estimate.

21.1.5 Offsite Infrastructure (WBS (7000)

Off-site infrastructure initial capital cost is $29.7 M, and includes provisions for the following:

- electrical 287kV/69kV substation at Volcano Creek;
- 17 km 69 kV overhead transmission line to plant site;
- widening of the access road and bridge upgrades on access road; and
- staging area at KM 2.

The 287kV/69kV substation and the 69kV overhead transmission line design/build costs were provided by Carisbrooke Consulting. The access road modifications and bridge upgrade costs were provided by AllNorth and well as the costs to development the staging area at KM 2.

21.1.5.1 Estimate Summary

The capital cost estimate has been developed to AACE Class 3. A summary of the project capital cost estimate is presented in Table 21-15.

Table 21-15: Capital Cost Estimate Summary

WBS	Description	Initial ($M)	Expansion Phase Total Cost ($M)	Sustaining Total Cost ($M)	LOM Total Cost ($M)
	Off-Site Infrastructure				
7100	Off-site Roads/Water Diversions	3.4	0.0	0.0	3.4
7200	Off-site Facilities	0.6	0.0	0.0	0.6
7600	Off-site Power Supply and Transmission	25.7	0.0	0.0	25.7
	Total Off-Site Infrastructure (excluding contingency)	**29.7**	**0.0**	**0.0**	**29.7**

21.1.6 Indirect Costs (WBS 9000)

Indirect costs are those that are required during the Project delivery period to enable and support the construction activities. Indirect costs include:

- Temporary facilities for the CM team
- Temporary services during construction, including but not limited to the following:
  - Bussing – from camp to workfront/return (Owner's costs)
  - Sub-contractors snow removal at local workfronts
  - Propane fuel cost during construction (Owner's costs)
- Construction equipment – to support CM team
- Temporary construction camp (rental basis) and camp operations and maintenance during construction period that includes renting 160 beds for Year -2 and rent an additional 50 beds for Year -1 (note: 227 beds are available at Forrest Kerr)
- First fills and spares
- Vendor representation support during installation and commissioning
- EPCM including the following:
  - home office engineering; site and home office expenses
  - commissioning services
  - sub-contractors bonding.

The indirect cost estimate was developed using a blend of first principles methods and budgetary pricing from contractors and recent historical costs.

EPCM (Ausenco and Third Parties) was estimated using first principles method. Temporary construction facilities, temporary services and construction equipment were developed using semi-detailed item list, Temporary construction camp bed count was developed by a built-up manning histogram and costs based on contractor's rental price submissions. Costs for freight services at KM 2 staging and transport to site was based on first principles. Spares and First Fills were based on percentages of direct equipment supply costs.

The indirect cost estimate is presented in Table 21-16.

Table 21-16: Indirect Costs Summary

WBS	Description	Initial ($M)	Expansion Phase Total Cost ($M)	Sustaining Total Cost ($M)	LOM Total Cost ($M)
	Project Indirects				
9100	EPCM & Commissioning	39.1	6.0	0.1	45.2
9200	Temporary Construction Facilities	0.6	0.0	0.0	0.6
9300	Temporary Construction Services	5.5	0.0	0.0	5.5
9400	Construction Equipment	0.6	0.0	0.0	0.6
9600	Temporary Camp and Catering	18.4	0.0	2.0	20.4
9700	Freight/traffic warehouse services at KM 2 staging – (Owner's costs)	2.2	0.0	0.0	2.2
9900	Spares, First Fills, Vendor Reps	7.2	0.6	2.8	10.6
	Total Project Indirects (excluding contingency)	**73.6**	**6.6**	**4.9**	**85.1**

21.1.7 Owner's Costs (WBS 8000)

Owner's costs were estimated by first principles. These costs include:

- General and administrative costs for the Owner's project team on and off-site;
- Security and First Aid;
- Pre-production operations;
- First Nations;
- Environmental;
- freight and logistics support;
- recruiting, training and site visits;
- IT and communications;
- insurance, finance, legal, and offices; and
- operational readiness.

Table 21-17: Owner's Costs – by Phase

WBS	Description	Initial ($M)	Expansion Phase Total Cost ($M)	Sustaining Total Cost ($M)	LOM Total Cost ($M)
8100	Owner's costs – operations readiness	30.3	0.0	0.0	30.3
	Total Owner's costs (excluding contingency)	**30.3**	**0.0**	**0.0**	**30.3**

21.1.8 Estimate Contingency (WBS 10000)

Estimate contingency is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost.

Contingency is defined as a monetary allowance that is included, over and above the base cost, to contribute to the success of the project by providing for the various cost uncertainties. The level of contingency varies depending on the nature of the contract and the Client's requirements. Due to uncertainties at the time the capital cost estimate was developed (in terms of the level of engineering definition, basis of the estimate, schedule development, etc.), it is essential that the estimate include a provision to cover the risk from these uncertainties.

The amount of risk was assessed with due consideration of the level of design work, the way pricing was derived, and the nature of the plan for project implementation.

A Probabilistic Contingency analysis was performed which consisted of a contingency ranging workshop taking place internally and evaluated the major cost components in terms of confidence of pricing and quantity basis and provided input ranges for potential underrun/overrun. The ranging inputs were applied as percentages to the base estimate and then ran in a Monte Carlo model using the @Risk program.

The estimate contingency does not allow for the following:

- abnormal weather conditions;
- changes to market conditions affecting the cost of labour or materials; and
- changes of scope within the general production and operating parameters effects of industrial disputes.

The following contingency percentages was applied:

- Process Plant and Site Infrastructure 9.9% (P80 confidence level based on @Risk simulation;
- Mining 5%;
- Power Supply 10%;
- Site Access Road 10%; and
- Owner 5%.

A summary of the contingency cost is shown in Table 21-18.

Table 21-18: Estimate Contingency

WBS	Party	Initial ($M)	Expansion Phase Total Cost ($M)	Sustaining Total Cost ($M)	LOM Total Cost ($M)
10000	Contingency – Process Plant and Site Infrastructure	34.96	2.15	3.42	40.53
10000	Contingency – Mining	6.15	0.0	3.26	9.41
10000	Contingency – Power Supply	2.71	0.0	0.0	2.71
10000	Contingency –Site Access Road	0.56	0.0	0.0	0.56
10000	Contingency - Owner	2.06	0.0	0.0	2.06
	Total Contingency	**46.44**	**2.15**	**6.68**	**55.27**

21.1.9 Growth Allowance

Each line item of the estimate is developed initially at base cost only. A growth allowance is then allocated to each element of those line item costs to reflect the level of definition of design and pricing strategy.

Estimate growth is:

- intended to account for items that cannot be quantified based on current engineering status, but which are empirically known to appear;
- accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at a feasibility study level; and
- pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalisation of commercial terms and conditions to be used on the project.

Where an allowance has been used that is the result of factoring, no growth has been applied, as the factor has been surmised from a total cost.

Growth has been calculated at the line item level by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The capital cost growth allowance is presented in Table 21-19.

Table 21-19: Growth Allowances

Commodity Code	Discipline	Growth Applied
A	Architectural	5%
B	Earthworks	5%
C	Concrete	5%
E	Electrical	5%
F	Platework	5%
I	Instrumentation	5%
M	Mechanical Equipment	3.5%
P	Pipework	5%
S	Structural Steel	5%

21.1.10 Exchange Rates

Vendors and contractors were requested to price in native currency. The estimate is prepared in the base currency of Canadian dollars (C$), where relevant exchange rates were used to convert to Canadian currency.

21.1.11 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- senior finance charges;
- residual value of temporary equipment and facilities;
- environmental approvals;
- this study or any further project studies;
- force majeure issues;
- future scope changes;
- special incentives (schedule, safety, or others);
- no allowance has been made for loss of productivity and/or disruption due to religious, union, social and/or cultural activities;
- Owner's escalation costs;
- Owner's foreign exchange exposure; and
- land acquisition.

21.1.12 Closure Costs

Closure Costs, based on the details provided in Section 20.4, are estimated to be C$138.3M. The following table shows the buildup of costs:

Table 21-20: Closure Costs (C$)

Closure Cost	Cost (C$M)
Site Preparation/Regrading/Water management	92.8
Revegetation	5.3
Maintenance	0.5
Facility Decommission	13.3
100 year Operational & Post-Closure Monitoring	8.4
Sub-total	120.3
Bond Premium Expense	18.0
Total	**138.3**

21.2 Operating Cost Estimate

21.2.1 Overview

The estimate conforms to Class 3 guidelines for a feasibility study level estimate with a ±15% accuracy according to the Association of the Advancement of Cost Engineering International (AACE International).

The operating cost estimate provided in Table 21-21 is based on a combination of first-principal calculations, experience, vendor quotes, reference projects and factors as appropriate for a FS.

Table 21-21: Operating Cost Estimate Summary (C$)

Tonnes Milled	Initial 3.0 Mt/a (typical)		Expansion 3.7 Mt/a (typical)		LOM *	
Operating Cost	C$M/a	C$/t milled	C$M/a	C$/t milled	C$M	C$/t milled
Mining	137	45.71	97	26.21	901	30.12
Process operations and maintenance	52	17.39	60	16.23	506	16.91
G&A	16	5.38	12	3.11	126	4.20
Total	**205**	**68.47**	**169**	**45.56**	**1,533**	**51.24**

3.0Mt/a costs represent a typical production year in the initial phase
3.7Mt/a costs represent a typical production year in the expansion phase. Mining declines and more material reclaimed from stockpiles after Y6 toward Y9.

21.2.2 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 $1.28/L delivered to the site. The mine fleet will be primarily diesel powered, except for the loading shovels and the large drills for which we are investigating electrified options in addition to diesel. We are also investigating the potential to electrify the pit dewatering pumps. A price of $0.06 per kilowatt was used for all electric equipment.

21.2.2.1 Labour

Labour costs for the various job classifications were obtained from salary surveys in British Columbia and other operations. A burden rate between 37% 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 two weeks on/two weeks off. Mine positions and salaries are shown in Table 21-22. 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 hourly labour requirements for Year 5 are shown in Table 21-23 below. 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 will 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 21-22: 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

Position	Employees	Annual Salary (C$/a)
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 21-23: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary (C$/a)
Mine General		
General Equipment Operator	8	100,900
Road/Pump Crew	8	97,900
General Mine Labourer	8	78,400
Light Duty Mechanic	4	130,200
Tire Technician	4	90,700
Lube Truck Driver	*32*	
Subtotal	8	100,900
Mine Operations		
Driller	16	105,600
Blaster	2	105,600
Blast Helper	4	78,400
Loader Operator	12	116,800
Hydraulic Shovel Operator	8	116,800
Haul Truck Driver	80	100,800
Dozer Operator	14	105,600
Grader Operator	7	105,600
Crusher Loader Operator	4	116,800
Snow plow/Water Truck	10	98,800
Subtotal	*157*	
Mine Maintenance		
Heavy/Light Duty Mechanics	45	130,200
Welder	23	130,200
Electrician	2	130,200
Apprentice	8	91,400
Subtotal	*78*	
Total Hourly	*235*	

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, and Lube Truck Drivers. The tire position is a contract. 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 Year 3 then drops to 16 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 20 in Year 2 and hold at that level until Year 5. Haulage Truck Drivers peak at 100 in Year 4 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.

21.2.2.2 Equipment Operating Costs

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Eskay Creek FS. 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 $30,700 so the cost per hour for tires is $36.83 /hr for the truck using 6 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 26 m/hr for the smaller drill and 27 m/hr for the electric drill for both mill feed and waste. The equipment costs used in the estimate are shown in Table 21-24.

Table 21-24: Major Equipment Operating Costs – No Labour ($/hr)

Equipment	Fuel/Power	Lube/ Oil	Tires Undercarriage	Repair Maintenance	GET/Consumables	Total
Production drill – 165 mm	83.20	8.32	3.00	131.94	96.14	322.60
Production drill (electric) – 165 mm	28.02	-	6.00	95.67	122.36	252.05
Production/crusher loader - 11.5 m^3	102.40	10.24	40.60	75.09	10.00	238.33
Hydraulic shovel – 22 m^3	68.22	-	-	187.14	30.00	285.36
Haulage truck – 40 t	38.40	3.84	18.00	51.75	5.00	116.99
Haulage truck – 91 t	96.00	9.60	18.80	75.07	3.00	202.47
Haulage truck – 144 t	121.60	12.16	36.83	89.67	4.00	264.26
Track dozer	89.60	8.96	10.00	62.93	5.00	176.49
Grader	28.16	2.82	3.20	15.66	5.00	54.84
Dragline	64.00	6.40	10.00	99.36	5.00	184.76
Support excavator – 6.7 m^3	76.80	15.36	5.00	59.06	8.00	164.22

21.2.2.3 Drilling

Drilling in the open pit will use down the hole hammer drill rigs. The preproduction drilling will be with the smaller diesel drills and 165 mm bits. The main production drills will use the same 165 mm bits and can potentially be electrified. 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 21-25.

Table 21-25: Drill Pattern Specifications

	Unit	Drill 165 mm		Drill 165 mm (electric)	
Specification	Unit	Mill Feed	Waste	Mill Feed	Waste
Bench height	m	10	10	10	10
Sub-drill	m	1.2	1.2	1.2	1.2
Blasthole diameter	mm	165	165	165	165
Pattern spacing - staggered	m	5.8	6.0	5.8	6.0
Pattern burden – staggered	m	5.0	5.2	5.0	5.2
Hole depth	m	11.2	11.2	11.2	11.2

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 parameters used to estimate drill productivity are shown in Table 21-26.

Table 21-26: Drill Productivity Criteria

		Drill 165 mm		Drill 165 mm electric	
Drill Activity	**Unit**	**Mill Feed**	**Waste**	**Mill Feed**	**Waste**
Pure penetration rate	m/min	0.55	0.55	0.55	0.55
Hole depth	m	11.2	11.2	11.2	11.2
Drill time	min	20.36	20.36	20.36	20.36
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	25.9	25.9	24.9	24.9
Drill productivity	m/hr	26.0	26.0	27.0	27.0

21.2.2.4 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 explosive calculation are shown in Table 21-27.

The blasting cost is estimated using quotations from a local explosives vendor. The emulsion price is $105.00/100 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 $119,500 per month.

Table 21-27: Design Powder Factor

	Unit	Drill 165 mm		Drill 165 mm	
		Mill Feed	**Waste**	**Mill Feed**	**Waste**
Powder Factor	kg/m^3	0.68	0.62	0.68	0.62
Powder Factor	kg/t	0.25	0.22	0.25	0.22

21.2.2.5 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 21-28, 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 21-28: Loading Parameters – Year 5

	Unit	Hydraulic Shovel	Front-End Loader
Bucket capacity	m^3	22	11.5
Truck capacity loaded	t	144	144
Waste tonnage loaded	%	70	30
Mill feed tonnage loaded	%	77	23
Bucket fill factor	%	89	85
Cycle time	sec	38	40
Trucks present at loading unit	%	80	80
Loading time	min	2.60	5.40

21.2.2.6 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 21-29.

Table 21-29: 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

21.2.2.7 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 21-30.

These factors resulted in the need for five track dozers, three graders, one dragline and two 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. Two self propelled snowblowers are also part of the support fleet and can throw the snow to the side or load into the smaller trucks to be hauled away. 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 21-30: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track dozer	25%	Of haulage hours to maximum of 4 dozers
Grader	15%	Of haulage hours to maximum of 3 graders
Crusher loader	25%	Of loading hours to maximum of 1 loader
Snowplow/water truck	10%	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
Snowblower	6	hours/day/unit
Road crew backhoe	4	hours/day/unit
Road crew dump truck	6	hours/day/unit
Road crew loader	8	hours/day/unit
Lube/fuel truck	8	hours/day/unit
Mechanics truck	12	hours/day/unit
Blasting loader	8	hours/day/unit
Blaster's truck	8	hours/day/unit
Integrated tool carrier	3	hours/day/unit
Light plants	12	hours/day/unit

21.2.2.8 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 5 m. These drill holes will be used to find undiscovered veinlets or pockets of mineralization. Over the life of the mine, a total of 229,000 m of drilling are expected to be completed for grade control work. A total of 252,000 samples are anticipated to be assayed from that drilling.

The grade control holes will serve three purposes:

- Definition of the mill feed grade and contacts;
- Location of previous underground infrastructure prior to blasthole rigs drilling;
- Identification of PAG/NAG material;
- Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.; and
- 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 $2M/a.

Additional costing for blasthole sampling has been included but only for PAG/NAG sampling. That cost is expected to be over $1M/a.

21.2.2.9 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 75,000 cubic metres. That climbs rapidly to 1.1 Mm^3 in Year -1 then levels at around 2.3 Mm^3 for the remainder of the mine life.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface.

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.

21.2.2.10 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; and

- Residual = $0.

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.16 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 1.16} / 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 $0.47/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it was $3.72/t mill feed.

21.2.2.11 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 21-31 and Table 21-32.

The General Mine Engineering includes the cost associated with an owner operated crushing plant to make stemming material and road crush. That cost is approximately $850,000 per year.

Table 21-31: 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	0.40	0.28	0.38	0.41
Drilling	$/t mined	0.19	0.19	0.20	0.21
Blasting	$/t mined	0.38	0.35	0.42	0.41
Loading	$/t mined	0.26	0.26	0.27	0.28
Hauling	$/t mined	0.73	1.14	1.35	1.26
Support	$/t mined	0.56	0.47	0.62	0.62
Grade control	$/t mined	0.10	0.08	0.11	0.11
Leasing costs	$/t mined	0.67	0.48	0.50	0.47
Dewatering	$/t mined	0.06	0.04	0.06	0.06
Total	**$/t mined**	**3.35**	**3.29**	**3.91**	**3.72**

Table 21-32: 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	4.46	3.95	3.91	3.22
Drilling	$/t mill feed	2.10	2.66	2.01	1.63
Blasting	$/t mill feed	4.30	4.95	4.31	3.26
Loading	$/t mill feed	2.88	3.62	2.79	2.20
Hauling	$/t mill feed	8.16	15.91	13.88	9.94
Support	$/t mill feed	6.25	6.54	6.35	4.87
Grade control	$/t mill feed	1.15	1.16	1.16	0.85
Leasing costs	$/t mill feed	7.54	6.75	5.08	3.68
Dewatering	$/t mill feed	0.69	0.56	0.65	0.48
Total	**$/t mill feed**	**37.53**	**46.09**	**40.15**	**30.12**

21.2.3 Processing

The operating costs were calculated to reflect the actual costs of developing a facility capable of processing 3 Mt/year between years one to five (Initial) and 3.7 Mt/y after six years (Expansion).

Operating costs include all regular, recurring costs of production, such as:

- Processing;
- Power consumption;
- Operating consumables;

- General maintenance; and
- General and administration (G&A).

The operating cost estimates were calculated based on the following assumptions:

- Plant capacity per day of 8,219 t/d for years 1 to 5 and 10,137 t/d for years 6+.
- Crushing and milling plant availabilities at 70% and 92%, respectively.
- Cost estimates were based on the first quarter (Q1) of 2022 pricing without allowances for inflation.
- The estimate is prepared in the base currency of Canadian dollars (C$), where relevant exchange rates were used to convert to Canadian currency.
- The propane cost was provided by Skeena Resources at C$0.60/L.
- Gasoline cost was estimated at C$1.44/L using a three-year trailing average from 2019 to 2022 StatCan data for the city of Victoria, BC.
- Diesel cost was estimated at C$1.28/L using a three-year trailing average from 2019 to 2022 Northern BC diesel prices.
- The power consumption cost was provided by Skeena Resources at C$0.06/kWh.
- Labour will mainly be sourced from British Columbia and locally such as Terrace, Stewart, and Smithers.
- Operating and maintenance consumables were provided by different vendors such as Univar, Molycorp, Metso-Outotec, Glencore, etc.
- Processing costs were determined based on labour, light vehicles and mobile equipment, operating and maintenance consumables, and processing power requirements.
- Off-site gold refining, insurance, and transportation costs were excluded
- Road maintenance costs were provided by Skeena.

Table 21-33: Operating Cost Summary

Cost Centre	Initial 3.0 Mt/a (typical)		Expansion 3.7 Mt/a (typical)	
	$M/a	$/t milled	$M/a	$/t milled
Processing	49.90	16.63	57.95	15.66
Road and Bridge Maintenance	2.26	0.75	2.11	0.57
G&A	16.13	5.38	11.51	3.11
TOTAL	**68.29**	**22.76**	**71.58**	**19.34**

3.0Mt/a costs represent a typical production year in the initial phase
3.7Mt/a costs represent a typical production year in the expansion phase.

21.2.3.1 Power

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh.

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. The British Columbia Hydro grid will supply power to service the facilities at the site. Costs associated with power consumption constitute about 20% of the total Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh.

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. The British Columbia Hydro grid will supply power to service the facilities at the site.

Annual energy consumption is estimated at 188,500 MWh/y, costing $11.3 M/y for years 1-5 and 224,100 MWh/y, costing $13.5M/y for years 6+.

21.2.3.2 Consumables

Processing reagent and consumable costs were estimated based on the throughput.

The operating consumables cost were developed with the following basis:

- Liner consumptions for the jaw crusher, SAG mill, ball mills, pebble crusher, regrind mill and Isamills were determined based on comminution and breakage data and Ausenco's calculations and in-house database.
- Grinding media consumption was based on the internal Ausenco calculations.
- Reagent consumption was estimated from metallurgical test work.
- Filter cloth consumption was established by benchmarking different Ausenco projects.
- All costs associated with operating consumables not stated above were determined using quotations and commercial proposals provided by vendors.

Table 21-34: Reagents and Consumables Summary

Operating Consumables (Variable Cost)	Initial 3.0 Mt/a (typical)		Expansion 3.7 Mt/a (typical)	
	$M/a	$/t milled	$M/a	$/t milled
Crushing & Conveying	0.08	0.03	0.08	0.02
Grinding/Milling/Classification	5.74	1.91	6.91	1.87
Flotation	20.77	6.92	25.01	6.76
Regrind	0.18	0.06	0.18	0.05
Process Utilities	0.54	0.18	0.63	0.17
Subtotal	**27.32**	**9.11**	**32.82**	**8.87**

3.0Mt/a costs represent a typical production year in the initial phase
3.7Mt/a costs represent a typical production year in the expansion phase.

21.2.3.3 Maintenance Consumables

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average factor from 3% to 4%. The factor was applied to mechanical equipment, plate work, and piping. For years 1 to 5, the total maintenance consumables operating cost was approximately $0.70 per tonne of feed while the cost for years 6+ is about $0.65 per tonne of feed.

This results in annual maintenance consumables cost estimate of $2.1M (Initial) and $2.4M (Expansion).

21.2.3.4 Labour

Labour includes all processing and maintenance labour costs.

Processing production labour was estimated by benchmarking against similar projects and includes plant operation departments such as metallurgy, mill operations, maintenance and the assay lab.

Each position was defined and classified as either a rotating shift of 14 days on and 14 days off or 7 days on and 7 days off. Table 21-36 presents the operations staffing. Costs including taxes and benefits. The annual cost estimate is $6.1M/y for process operations labour and $2.7M/y for process maintenance labour.

Table 21-35: Process Operations and Maintenance Labour

Processing Production Labour	Schedule (days on/off)	Roles per shift	No of employees
Mill Superintendent	7/7	1	1
Senior metallurgist	7/7	1	1
Metallurgist	14/14	1	2
Supervisors	14/14	1	4
Control Room Operator	14/14	1	4
Crushing Plant Operator	14/14	1	4
Grinding & Flotation Operator	14/14	2	8
Reagents Operator	14/14	1	2

Processing Production Labour	Schedule (days on/off)	Roles per shift	No of employees
Dewatering Operator	14/14	1	4
Tailings Operator	14/14	1	4
Helpers	14/14	1	4
Chief Assayer	7/7	1	1
Assayer	14/14	1	4
Lab Technician (Sample Prep)	14/14	2	8
Total Processing Production - 7/7 days on/off			3
Total Processing Production - 14/14 days on/off			48
Sub Total Processing Production		16	51
Maintenance Superintendent	7/7	1	1
Maintenance Supervisor	7/7	1	1
Electrical Supervisor	7/7	1	1
Instrumentation	14/14	1	2
Electrician	14/14	1	4
Mechanic	14/14	1	4
Welder	14/14	1	2
Electrical Apprentices	14/14	1	2
Mill Maintenance Apprentices	14/14	1	2
Planner/Clerk	14/14	1	2
Total Processing Maintenance - 7/7 days on/off		0	3
Total Processing Maintenance - 14/14 days on/off		0	18
Sub Total Processing Maintenance		**10**	**21**
Total Processing Labour		**26**	**72**

21.2.4 General and Administration

21.2.4.1 G&A Costs Overview

G&A costs are expenses not directly related to the production of gold and include expenses not considered in mining, processing, external refining, and transportation costs. These costs were developed with input from Ausenco's in-house database on existing Canadian operations., industry practice and feedback from Skeena Resources.

Some G&A areas have individual costs while some others consider several departments as follows:

- General including medical and first aid, environment, travel, training and safety, computer supplies, entertainment, and memberships;
- Contract and services comprising of insurance, consulting/External Assays, relocation expenses, recruitment, audit, and legal services departments;
- Travel and Camp costs include:
  - rotational return air from Vancouver to Terrace or Smithers and shuttling to site;

- all catering and camp operating costs for the life of mine; and
- first 3 years of operations there is also a camp rental costs and bussing from rental camp to process plant.

- Other costs included physical services for communications, expenses of the liaison committee/sustainability.

Table 21-36: G&A Costs Overview

Cost Center	Initial 3.0 Mt/a (typical)		Expansion 3.7 Mt/a (typical)	
	C$M/a	$/t milled	$M/a	$/t milled
Total G&A Wages	$3.98	$1.23	$3.68	$1.00
Total G&A expenses	$3.87	$1.19	$3.56	$0.96
Travel and Camp Costs	$9.58	$2.96	$4.27	$1.15
Total	**$17.43**	**$5.38**	**$11.51**	**$3.11**

Table 21-37: G&A Labour

G&A Labour	Schedule (days on/off)	Roles per Shift	No of employees
General Manager	7/7	1	1
Environmental Superintendent	7/7	1	1
Environmental Engineer	7/7	1	1
Environmental Technician	14/14	1	2
HR Superintendent	7/7	1	1
HR	7/7	1	1
Receptionist	7/7	1	1
Safety Superintendent	7/7	1	1
Safety Officer	14/14	1	2
IT Technician	14/14	1	2
Controller	7/7	1	1
Accountant	7/7	1	1
Buyer	14/14	1	2
Warehouse	14/14	2	4
Payroll Clerk	14/14	1	2
Accounts Payable/Receivable Clerk	14/14	1	2
Sub Total Management			**4**
Sub Total Staff			**24**
Total Staff			**30**

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

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; and
- 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 FS assumes that permits must 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 2022 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:
  - Technical sample begins middle of Year -3;
  - First gold concentrate produced in second half of Year -1;
- Mine life of 9 years;
- Base case gold price of US$1,700/oz and silver price of US$19/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.76 (US$/C$);
- Cost estimates in constant Q1 2022 C$ 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;
- Shipping costs disregard of escalation over the LOM;
- 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 FS.

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$983M over the LOM.

22.3.2 Working Capital

Working capital assumptions include Accounts Receivable (0 days), Inventories (30 days) and Accounts Payable (30 days). The effective sum of working capital over the life of mine is zero.

22.3.3 Royalty

A 2% NSR royalty has been assumed for the project, resulting in approximately C$100 million in royalty payments over life of mine.

22.3.4 Closure Costs

Closure costs include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude, including a period of 100-year post-closure monitoring. The closure costs total $138M.

22.4 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV 5%) is C$2,093.7 M, the internal rate of return IRR is 59.5%, and payback is 1.0 years. On an after-tax basis, the NPV 5% is C$1,412.1 M, the IRR is 50.2%, and the payback period is 1.0 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.

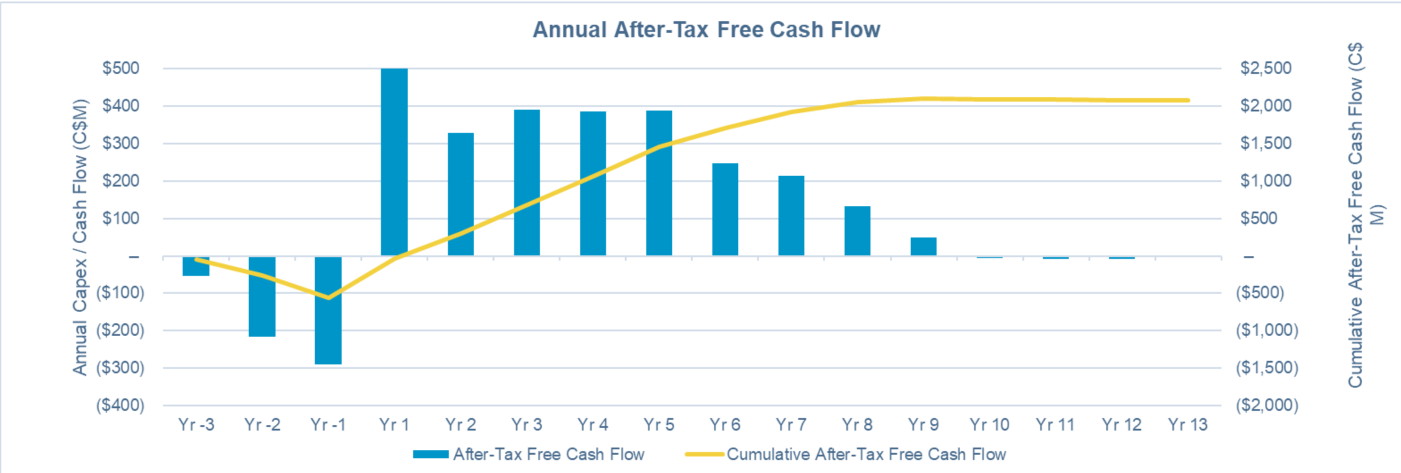
Table 22-1: Summary, Projected LOM Cashflow Assumptions and Results

Description	Units	Values
General Assumptions		
Gold price	(US$)	1,700
Silver price	(US$)	19
Exchange rate	(US$/C$)	0.76
Fuel cost	(C$/litre)	1.28
Power cost	(C$/kwh)	0.06
Discount rate	(%)	5%
Contained Metals		
Contained gold ounces	(koz)	2,874
Contained silver ounces	(koz)	75,538
Production		
Gold recovery	(%)	84.2%
Silver recovery	(%)	88.3%
LOM gold production	(koz)	2,419
LOM silver production	(koz)	66,707
LOM gold equiv. production	(koz)	3,164
LOM avg. annual gold production	(koz per annum)	269
LOM avg. annual silver production	(koz per annum)	7,412
LOM avg. annual gold equiv. production	(koz per annum)	352
Operating Costs Per Tonne		
Mining cost	(C$/t mined)	3.72
Mining cost	(C$/t milled)	30.12
Processing cost	(C$/t milled)	16.91
G&A cost	(C$/t milled)	4.20
Total operating costs	(C$/t milled)	51.24
NSR Parameters		
Net smelter royalty	(%)	2%

Description	Units	Values
Transport to smelter	(C$/wmt)	140
Cash Costs and All-in Sustaining Costs		
LOM cash cost net of silver by-product	(US$/oz Au)	253
LOM cash cost co-product	(US$/oz AuEq)	572
LOM AISC net of silver by-product	(US$/oz Au)	355
LOM AISC co-product	(US$/oz AuEq)	652
Capital Expenditures		
Pre-production capex (initial capital)	(C$M)	592
Expansion capex (year 5)	(C$M)	40
Sustaining capex	(C$M)	140
Closure capex	(C$M)	138
Economics		
Pre-tax NPV (5%)	(C$M)	2,094
Pre-tax IRR	(%)	59.5%
Pre-tax payback period	(years)	0.99
Pre-Tax NPV / Initial Capex	(x)	3.5 x
After-tax NPV (5%)	(C$M)	1,412
After-tax IRR	(%)	50.2%
After-tax payback period	(years)	1.0
After-Tax NPV / Initial Capex	(x)	2.4 x
Average annual after-tax free cash flow (Year 1–9)	(C$M)	293
LOM after-tax free cash flow	(C$M)	2,110

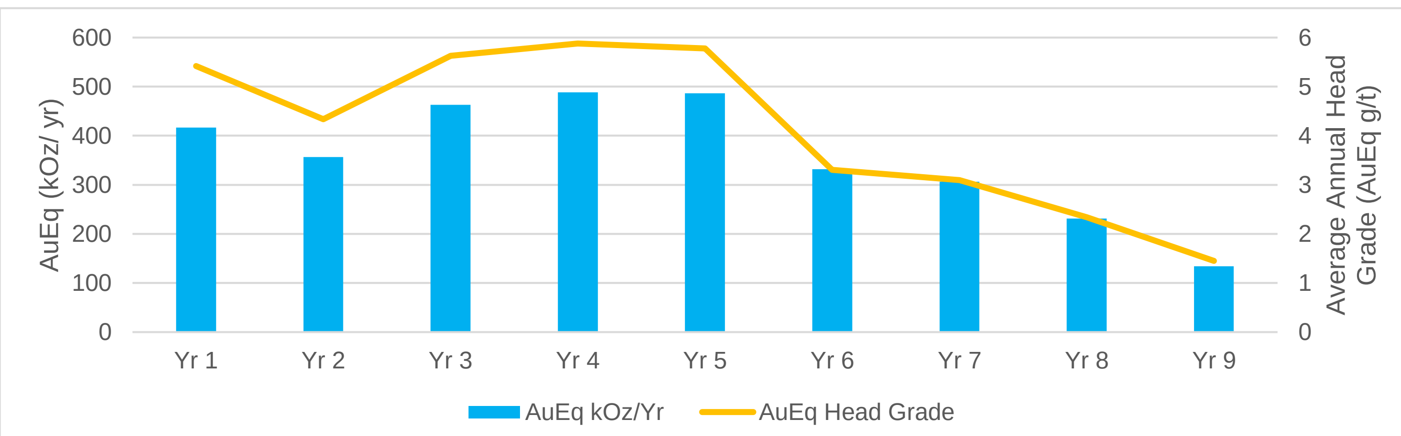
Notes: Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. AISC includes cash costs plus corporate G&A, sustaining capital and closure costs. Gold equivalent (AuEq) calculated using the formula: Au (g/t) + [Ag (g/t) / 89].

Figure 22-1: Projected LOM Cashflow



Note: Figure prepared by Ausenco, 2022.

Figure 22-2: Projected LOM Production



Note: Figure prepared by Ausenco, 2022.

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
Free Cash Flow Valuation																	
Gross Revenue		$5,981	–	–	$50	$813	$670	$871	$910	$913	$619	$557	$375	$203.4	–	–	–
Penalties	C$mm	($50)	–	–	($0)	($41)	($3)	($2)	($1)	($2)	($1)	–	–	($0.5)	–	–	–
Transport	C$mm	($317)	–	–	($3)	($28)	($30)	($37)	($37)	($37)	($34)	($39)	($46)	($27.6)	–	–	–
Net Smelter Return		$5,614	–	–	$47	$745	$637	$833	$873	$874	$584	$518	$329	$175.3	–	–	–
Operating Expenses	C$mm	($1,531)	–	–	($14)	($164)	($205)	($207)	($203)	($186)	($187)	($169)	($111)	($85.0)	–	–	–
Royalties	C$mm	($112)	–	–	($1)	($15)	($13)	($17)	($17)	($17)	($12)	($10)	($7)	($3.5)	–	–	–
EBITDA	C$mm	$3,971	–	–	$33	$565	$418	$609	$653	$670	$385	$339	$211	$86.7	–	–	–
Initial & Expansion Capex	C$mm	($631)	($53)	($215)	($323)	–	–	–	–	($40)	–	–	–	–	–	–	–
Sustaining Capex	C$mm	($140)	–	–	–	($28)	($23)	($40)	($40)	($7)	($2)	($2)	($0)	–	–	–	–
Closure Capex	C$mm	($138)	–	–	–	($2)	($2)	($12)	($12)	($12)	($12)	($12)	($12)	($16.2)	($16)	($16)	($16)
Change in Working Capital	C$mm	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Pre-Tax Unlevered Free Cash Flow		$3,061	($53)	($215)	($291)	$536	$394	$558	$602	$611	$372	$326	$199	$70.5	($16)	($16)	($16)
Unlevered Cash Taxes	C$mm	($983)	–	–	($1)	($12)	($66)	($168)	($216)	($223)	($125)	($111)	($67)	($21.1)	$10	$9	$8
Post-Tax Unlevered Free Cash Flow		$2,077	($53)	($215)	($291)	$524	$328	$390	$385	$388	$247	$215	$132	$49.3	($7)	($8)	($8)
Production																	
Ore Mined	'000t	29,911	–	138	687	3,527	3,341	3,126	4,118	4,954	4,345	4,236	1,441	–	–	–	–
Stockpile Rehandle	'000t	9,111	–	138	487	1,677	804	683	1,118	1,954	991	1,036	224	–	–	–	–
Waste Mined	'000t	225,052	2,055	3,389	6,813	27,027	38,659	38,874	33,804	25,824	25,655	19,754	3,198	–	–	–	–
Total Material Mined (Includes Rehandle)	'000t	264,075	2,055	3,665	7,987	32,231	42,804	42,683	39,040	32,732	30,991	25,025	4,862	–	–	–	–
Total Material Mined (Excl. Rehandle)	'000t	254,964	2,055	3,527	7,500	30,553	42,000	42,000	37,922	30,778	30,000	23,989	4,638	–	–	–	–
Strip Ratio		7.52	–	–	18.67	9.91	12.89	12.96	11.27	8.61	6.93	5.34	0.86	–	–	–	–
Total Mill Feed	'000t	29,911	–	–	365	2,727	3,000	3,000	3,000	3,000	3,700	3,700	3,700	3,719	–	–	–
Beginning Stockpile Inventory	'000t		–	–	138	459	1,259	1,599	1,725	2,844	4,797	5,442	5,978	3,719	0	0	0
Add: Mine to Stockpile	'000t	9,111	–	138	487	1,677	804	683	1,118	1,954	991	1,036	224	–	–	–	–
Less: Stockpile to Mill	'000t	(9,111)	–	–	(165)	(878)	(463)	(557)	–	–	(346)	(501)	(2,483)	(3,719)	–	–	–
Ending Stockpile Inventory	'000t		–	138	459	1,259	1,599	1,725	2,844	4,797	5,442	5,978	3,719	0	0	0	0
Au Head Grade	g/t	2.99	–	–	1.89	4.47	3.28	4.21	4.12	4.26	2.50	2.45	1.72	1.12	–	–	–
Ag Head Grade	g/t	78.55	–	–	73.70	76.22	85.67	114.84	142.27	122.87	64.80	52.41	50.10	26.54	–	–	–
Contained Gold	kozs	2,873.8	–	–	22.2	392.4	316.4	406.3	397.5	411.2	297.9	291.1	204.6	134.2	–	–	–
Contained Silver	kozs	75,538.3	–	–	864.9	6,684.0	8,263.4	11,076.6	13,722.6	11,851.2	7,708.2	6,234.6	5,960.0	3,172.7	–	–	–
Au Recovery	%	84.2%	0%	0%	83%	87%	84%	84%	85%	86%	83%	82%	82%	76%	74%	0%	0%
Ag Recovery	%	88.3%	0%	0%	90%	89%	88%	89%	90%	90%	87%	86%	86%	82%	79%	0%	0%
		Total LOM															
Recovered Gold in Concentrate	kozs	2,418.6	–	–	18.4	343.0	266.3	342.0	336.5	354.1	248.7	240.1	168.0	101.6	–	–	–
Recovered Silver in Concentrate	kozs	66,707.3	–	–	780.9	5,959.2	7,305.5	9,813.7	12,317.4	10,704.9	6,712.6	5,360.3	5,148.3	2,604.5	–	–	–
Recovered Gold Equivalent in Concentrate	kozs	3,164.2	–	–	27.1	409.6	348.0	451.6	474.2	473.7	323.7	300.0	225.5	130.7	–	–	–
Total Payable Gold	kozs	2,083.3	–	–	15.4	310.4	234.1	301.6	296.8	312.3	216.6	201.2	121.5	73.5	–	–	–
Total Payable Silver	kozs	52,845.0	–	–	624.8	4,767.3	5,844.4	7,851.0	9,853.9	8,564.0	5,370.0	4,288.3	4,118.6	1,562.7	–	–	–
Total Payable Gold Equivalent	kozs Au Eq	2,673.9	–	–	22.4	363.7	299.4	389.4	406.9	408.0	276.7	249.1	167.5	90.9	–	–	–
Macro Assumptions																	
Gold Price	US$/oz	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700	$1,700
Silver Price	US$/oz	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00	$19.00
FX	C$:US$	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76
Revenue																	
Gold Revenue	C$mm	$4,660.1	–	–	$34.5	$694.3	$523.7	$674.7	$663.9	$698.6	$484.6	$450.0	$271.7	$164.3	–	–	–
Silver Revenue	C$mm	$1,321.1	–	–	$15.6	$119.2	$146.1	$196.3	$246.3	$214.1	$134.3	$107.2	$103.0	$39.1	–	–	–
Total Revenue	C$mm	$5,981.2	–	–	$50.1	$813.5	$669.8	$870.9	$910.2	$912.7	$618.8	$557.2	$374.6	$203.4	–	–	–
Total Mill Feed	'000t	29,911	–	–	365	2,727	3,000	3,000	3,000	3,000	3,700	3,700	3,700	3,719	–	–	–
Mass Pull		6.75%	–	–	4.58%	6.57%	6.39%	7.80%	7.75%	7.81%	5.81%	6.73%	7.85%	4.72%	–	–	
Concentrate Produced	000t (dmt)	2,018	–	–	17	179	192	234	233	234	215	249	290	176	–	–	–
Concentrate Au Grade	g/t	37			34	60	43	45	45	47	36	30	18	18	11	–	–
Concentrate Ag Grade	g/t	1,024			1,436	1,026	1,183	1,298	1,629	1,429	971	674	540	458	100	–	–
Penalties	C$mm	$49.8	–	–	$0.4	$40.8	$3.0	$1.7	$0.5	$2.3	$0.6	–	–	$0.5	–	–	–
Transport to Smelter	C$mm	$317.0	–	–	$2.63	$28.13	$30.13	$36.74	$36.53	$36.81	$33.75	$39.09	$45.59	$27.57	–	–	–
Net Smelter Return		$5,614.4		–	$47.1	$744.5	$636.6	$832.5	$873.2	$873.6	$584.5	$518.1	$329.0	$175.3	–	–	–
Royalty	%	2.00%	–	–	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	2.00%	–	–	–
Total Royalties	C$mm	$112.3	–	–	$0.9	$14.9	$12.7	$16.7	$17.5	$17.5	$11.7	$10.4	$6.6	$3.5	–	–	–
Penalties																	
Total Antimony (Sb) Penalty	C$mm	$3.2	–	–	–	$2.3	$0.9	–	–	–	–	–	–	–	–	–	–
Total Arsenic (As) Penalty	C$mm	$30.4	–	–	$0.4	$22.2	$2.1	$1.7	$0.5	$2.3	$0.6	–	–	$0.5	–	–	–
Total Mercury (Hg) Penalty	C$mm	$16.3	–	–	–	$16.3	–	–	–	–	–	–	–	–	–	–	–

Total Penalties	C$mm	$49.8	–	–	$0.4	$40.8	$3.0	$1.7	$0.5	$2.3	$0.6	–	–	$0.5	–	–	–
Operating Costs																	
Per Tonne Basis																	
Mining Cost - OP	C$/t mined OP	$3.72	–	–	–	$3.35	$3.26	$3.29	$3.61	$3.91	$3.86	$4.04	$8.60	–	–	–	–
Processing Cost	C$/t milled	$16.90	–	–	$22.40	$17.39	$17.39	$17.39	$17.49	$17.49	$16.23	$16.23	$16.23	$16.23	–	–	–
G&A Cost	C$/t milled	$4.18	–	–	$14.73	$5.38	$5.38	$5.38	$4.45	$4.45	$3.11	$3.11	$3.11	$3.11	–	–	–
Annual C$M Basis																	
Mining Cost - OP	C$mm	$901.0	–	–	–	$102.4	$137.1	$138.3	$136.9	$120.5	$115.9	$97.0	$39.9	$13.1	–	–	–
Processing Cost	C$mm	$505.4	–	–	$8.2	$47.4	$52.2	$52.2	$52.5	$52.5	$60.1	$60.1	$60.1	$60.4	–	–	–
G&A Cost	C$mm	$125.1	–	–	$5.4	$14.7	$16.1	$16.1	$13.3	$13.3	$11.5	$11.5	$11.5	$11.6	–	–	–
Total Operating Costs	C$mm	$1,531.5	–	–	$13.6	$164.4	$205.4	$206.5	$202.7	$186.3	$187.5	$168.6	$111.4	$85.0	–	–	–
Operating Costs per Tonne Milled - excl. smelter costs & royalties	C$/t milled	$51.2	–	–	$37.1	$60.3	$68.5	$68.8	$67.6	$62.1	$50.7	$45.6	$30.1	$22.9	–	–	–
Cash Costs																	
By-Product Basis																	
Cash Cost *	US$/oz Au	$251	–	–	$92	$316	$341	$165	$28	$70	$348	$419	$380	$803	–	–	–
All-in Sustaining Cost (AISC) **	US$/oz Au	$354	–	–	$240	$0	$430	$301	$167	$123	$406	$480	$472	$1,002	–	–	–
Co-Product Basis																	
Cash Cost *	US$/oz AuEq	$571	–	–	$593	$519	$638	$511	$480	$452	$642	$665	$742	$975	–	–	–
All-in Sustaining Cost (AISC) **	US$/oz AuEq	$651	–	–	$695	$586	$707	$617	$582	$493	$687	$715	$810	$1,136	–	–	–
* Cash costs consist of mining cost, processing cost, site G&A, treatment, and refining charges & royalties			** AISC includes cash costs plus corporate G&A, sustaining capital and closure costs														
Capital Expenditures																	
Initial & Expansion Capital																	
Mining Equipment	C$mm	$23.6	$14.9	$3.4	$5.1	–	–	–	–	$0.2	–	–	–	–	–	–	–
Mining Other	C$mm	$23.3	–	$9.3	$14.0	–	–	–	–	–	–	–	–	–	–	–	–
Pre-Production Stripping	C$mm	$78.2	–	$31.3	$46.9	–	–	–	–	–	–	–	–	–	–	–	–
Processing - Secondary Grinding	C$mm	$28.6	–	–	–	–	–	–	–	$28.6	–	–	–	–	–	–	–
Processing - Fines Flotation	C$mm	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Processing - Earth Works	C$mm	$19.2	$6.9	$4.9	$7.4	–	–	–	–	–	–	–	–	–	–	–	–
Processing - CAPEX (Remaining)	C$mm	$178.7	–	$71.5	$107.2	–	–	–	–	–	–	–	–	–	–	–	–
Onsite Infrastructure	C$mm	$70.6	$18.5	$20.1	$30.1	–	–	–	–	$2.0	–	–	–	–	–	–	–
Offsite Infrastructure (Access Road, Water, Power)	C$mm	$50.1	$3.4	$18.6	$28.0	–	–	–	–	–	–	–	–	–	–	–	–
Processing Indirects (Incl. EPCM)	C$mm	$80.2	$0.2	$29.3	$44.0	–	–	–	–	$6.7	–	–	–	–	–	–	–
Owners Cost	C$mm	$30.3	$7.4	$9.1	$13.7	–	–	–	–	–	–	–	–	–	–	–	–
Contingency	C$mm	$48.6	$1.8	$17.9	$26.8	–	–	–	–	$2.1	–	–	–	–	–	–	–
Sub-Total Initial & Expansion Capital		$631.4	$53.1	$215.4	$323.2	–	–	–	–	$39.7	–	–	–	–	–	–	–
Sustaining Capital																	
Mining	C$mm	$39.9				$13.4	$14.3	$4.4	$2.7	$1.9	$1.6	$1.4	$0.1	–	–	–	–
Processing	C$mm	$2.4				–	–	$2.4	–	–	–	–	–	–	–	–	–
Onsite Infrastructure	C$mm	$63.8				$1.3	$2.2	$28.6	$31.1	$0.7	–	–	–	–	–	–	–
Onsite Infrastructure (Tailings + Water)	C$mm	$22.7	–	–	–	$10.1	$4.0	–	$4.1	$4.3	–	$0.1	–	–	–	–	–
Indirects	C$mm	$4.9				$1.5	$1.2	$2.3	–	–	–	–	–	–	–	–	–
Contingency	C$mm	$6.7				$1.3	$1.1	$1.9	$1.9	$0.3	$0.1	$0.1	$0.0	–	–	–	–
Sub-Total Sustaining Capital	C$mm	$140.4				$27.6	$22.8	$39.6	$39.7	$7.2	$1.7	$1.6	$0.1	–	–	–	–
Closure Cost	C$mm	$138.3				$1.5	$1.5	$11.7	$11.7	$11.7	$11.7	$11.7	$11.7	$16.2	$16.2	$16.2	$16.2
Total Capital Expenditures	C$mm	$910.1	$53.1	$215.4	$323.2	$29.1	$24.3	$51.3	$51.4	$58.6	$13.5	$13.3	$11.8	$16.2	$16.2	$16.2	$16.2

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 22-3 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 exchange rates, 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.

Table 22-3: Sensitivity Analysis Summary

Sensitivity Summary	Unit	Even Lower Case	Lower Case	Base Case	Higher Case	Upside Case
Gold Price (US$/oz)	(US$/oz)	$1,500	$1,600	$1,700	$1,800	$1,900
Silver Price (US$/oz)	(US$/oz)	$15.00	$17.00	$19.00	$21.00	$23.00
After-Tax NPV (5%) (C$M)	(C$M)	$1,044	$1,228	$1,412	$1,596	$1,780
After-Tax IRR (%)	(%)	41.0%	45.7%	50.2%	54.6%	58.7%
After-Tax Payback (yrs)	(yrs)	1.29	1.14	1.01	0.93	0.83
After-Tax NPV / Initial Capex		1.8 x	2.1 x	2.4 x	2.7 x	3.0 x
Average Annual After-Tax Free Cash Flow (Yr 1-9) (C$M)	(C$M)	$237	$265	$293	$321	$350

Figure 22-3: Pre-Tax NPV & IRR Sensitivity Results

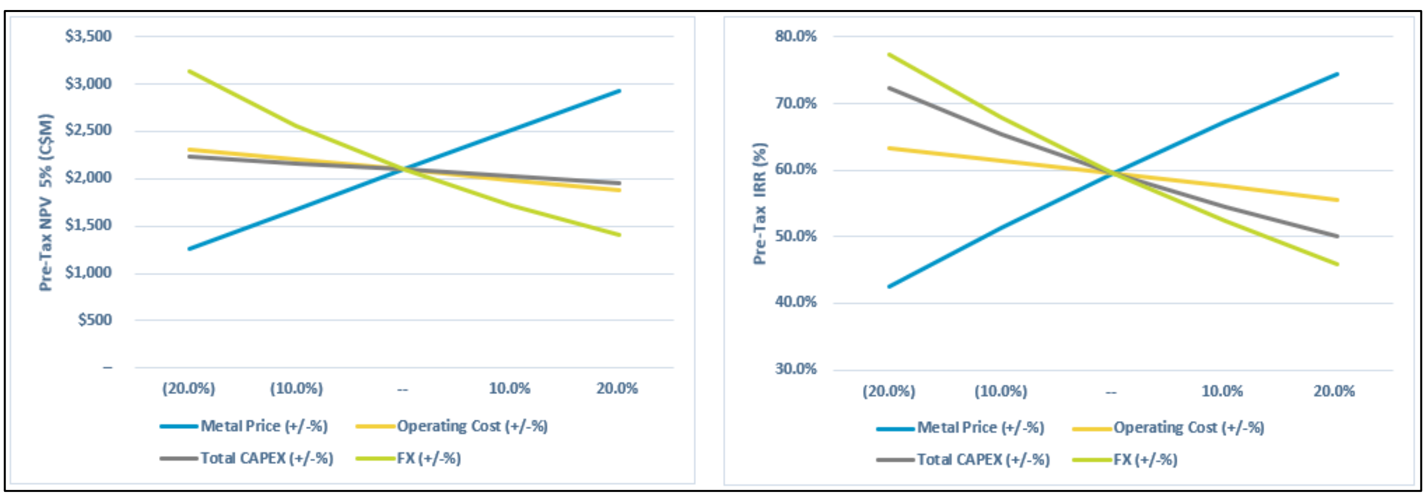


Figure prepared by Ausenco, 2022.

Table 22-4: Pre-Tax Sensitivity

Pre-Tax NPV Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,500	$1,600	$1,700	$1,800	$1,900
$15.00	$1,517	$1,709	$1,901	$2,093	$2,285
$17.00	$1,613	$1,805	$1,997	$2,189	$2,382
$19.00	$1,709	$1,902	$2,094	$2,286	$2,478
$21.00	$1,806	$1,998	$2,190	$2,382	$2,574
$23.00	$1,902	$2,094	$2,286	$2,478	$2,670

Pre-Tax IRR Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,500	$1,600	$1,700	$1,800	$1,900
$15.00	48.2%	52.3%	56.2%	60.0%	63.6%
$17.00	50.1%	54.0%	57.9%	61.6%	65.2%
$19.00	51.9%	55.8%	59.5%	63.2%	66.8%
$21.00	53.6%	57.4%	61.2%	64.8%	68.3%
$23.00	55.3%	59.1%	62.8%	66.3%	69.8%

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.0%	$2,253	$2,658	$3,063	$3,469	$3,874
3.0%	$1,774	$2,103	$2,433	$2,762	$3,091
5.0%	$1,517	$1,805	$2,094	$2,382	$2,670
8.0%	$1,203	$1,441	$1,679	$1,918	$2,156
10.0%	$1,031	$1,243	$1,454	$1,665	$1,876

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.0%	48.2%	54.0%	59.5%	64.8%	69.8%
3.0%	48.2%	54.0%	59.5%	64.8%	69.8%
5.0%	48.2%	54.0%	59.5%	64.8%	69.8%
8.0%	48.2%	54.0%	59.5%	64.8%	69.8%
10.0%	48.2%	54.0%	59.5%	64.8%	69.8%

Pre-Tax NPV Sensitivity To FX (CAD:USD)

FX (CAD:USD)	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.68	$1,936	$2,258	$2,580	$2,903	$3,225
0.73	$1,663	$1,963	$2,264	$2,564	$2,864
0.76	$1,517	$1,805	$2,094	$2,382	$2,670
0.83	$1,217	$1,481	$1,745	$2,009	$2,273
0.88	$1,031	$1,280	$1,529	$1,778	$2,028

Pre-Tax IRR Sensitivity To FX (CAD:USD)

FX (CAD:USD)	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.68	56.6%	62.6%	68.3%	73.7%	79.0%
0.73	51.2%	57.1%	62.7%	68.0%	73.1%
0.76	48.2%	54.0%	59.5%	64.8%	69.8%
0.83	41.7%	47.4%	52.8%	57.9%	62.8%
0.88	37.5%	43.1%	48.5%	53.5%	58.3%

Pre-Tax NPV Sensitivity To Opex

Opex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	$1,733	$2,021	$2,309	$2,598	$2,886
(10.0%)	$1,625	$1,913	$2,202	$2,490	$2,778
--	$1,517	$1,805	$2,094	$2,382	$2,670
10.0%	$1,409	$1,697	$1,986	$2,274	$2,563
20.0%	$1,301	$1,590	$1,878	$2,166	$2,455

Pre-Tax IRR Sensitivity To Opex

Opex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	52.4%	58.0%	63.3%	68.4%	73.3%
(10.0%)	50.3%	56.0%	61.4%	66.6%	71.6%
--	48.2%	54.0%	59.5%	64.8%	69.8%
10.0%	46.0%	52.0%	57.6%	62.9%	68.0%
20.0%	43.8%	49.9%	55.6%	61.0%	66.2%

Pre-Tax NPV Sensitivity To Total Capex

Total Capex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	$1,651	$1,939	$2,228	$2,516	$2,804
(10.0%)	$1,584	$1,872	$2,161	$2,449	$2,737
--	$1,517	$1,805	$2,094	$2,382	$2,670
10.0%	$1,450	$1,738	$2,027	$2,315	$2,603
20.0%	$1,383	$1,671	$1,960	$2,248	$2,536

Pre-Tax IRR Sensitivity To Total Capex

Total Capex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	59.6%	66.1%	72.4%	78.3%	83.9%
(10.0%)	53.4%	59.6%	65.4%	70.9%	76.3%
--	48.2%	54.0%	59.5%	64.8%	69.8%
10.0%	43.7%	49.3%	54.5%	59.5%	64.3%
20.0%	39.9%	45.2%	50.2%	54.9%	59.5%

Figure 22-4: Post-Tax NPV & IRR Sensitivity Results

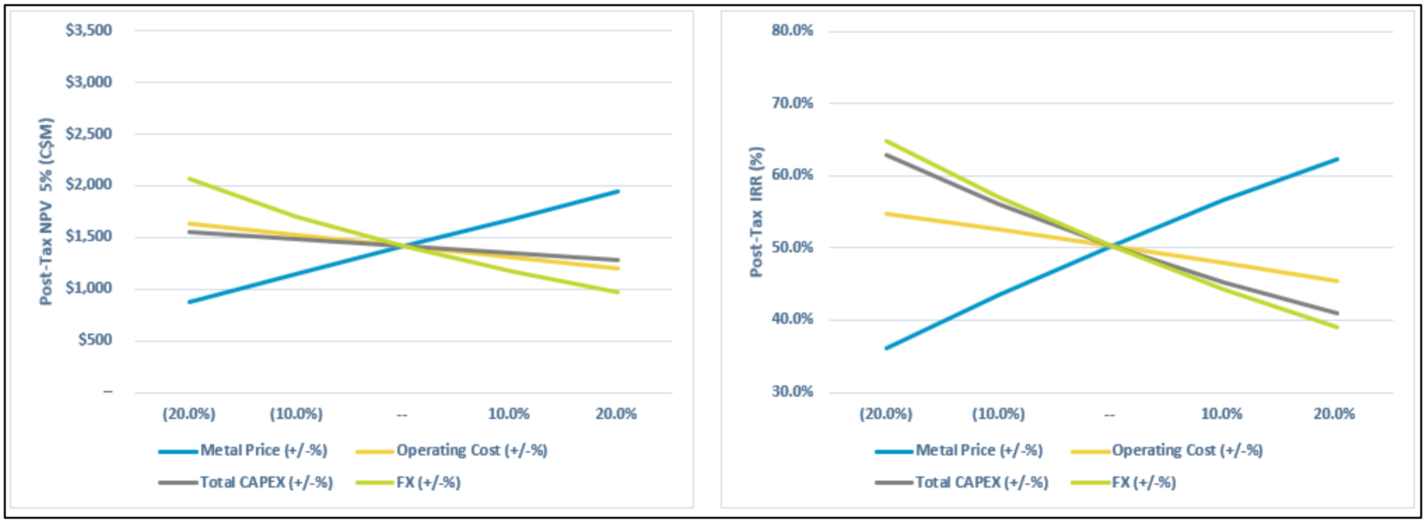


Figure prepared by Ausenco, 2022.

Table 22-5: Post-Tax Sensitivity

Post-Tax NPV Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz): $1,500	$1,600	$1,700	$1,800	$1,900
$15.00	$1,044	$1,167	$1,290	$1,412	$1,535
$17.00	$1,106	$1,228	$1,351	$1,473	$1,596
$19.00	$1,167	$1,290	$1,412	$1,535	$1,657
$21.00	$1,228	$1,351	$1,473	$1,596	$1,718
$23.00	$1,289	$1,412	$1,535	$1,657	$1,780

Post-Tax IRR Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz): $1,500	$1,600	$1,700	$1,800	$1,900
$15.00	41.0%	44.3%	47.5%	50.6%	53.7%
$17.00	42.5%	45.7%	48.9%	51.9%	55.0%
$19.00	44.0%	47.1%	50.2%	53.3%	56.3%
$21.00	45.4%	48.5%	51.6%	54.6%	57.5%
$23.00	46.8%	49.9%	52.9%	55.9%	58.7%

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.0%	$1,566	$1,823	$2,080	$2,337	$2,593
3.0%	$1,227	$1,436	$1,646	$1,855	$2,064
5.0%	$1,044	$1,228	$1,412	$1,596	$1,780
8.0%	$820	$974	$1,126	$1,279	$1,431
10.0%	$698	$835	$970	$1,106	$1,241

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.0%	41.0%	45.7%	50.2%	54.6%	58.7%
3.0%	41.0%	45.7%	50.2%	54.6%	58.7%
5.0%	41.0%	45.7%	50.2%	54.6%	58.7%
8.0%	41.0%	45.7%	50.2%	54.6%	58.7%
10.0%	41.0%	45.7%	50.2%	54.6%	58.7%

Post-Tax NPV Sensitivity To FX (USD:CAD)

FX (CAD:USD)	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.68	$1,311	$1,517	$1,722	$1,927	$2,132
0.73	$1,138	$1,329	$1,520	$1,712	$1,903
0.76	$1,044	$1,228	$1,412	$1,596	$1,780
0.83	$849	$1,020	$1,190	$1,358	$1,526
0.88	$729	$891	$1,052	$1,211	$1,370

Post-Tax IRR Sensitivity To FX (USD:CAD)

FX (CAD:USD)	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
0.68	47.8%	52.7%	57.5%	61.8%	65.9%
0.73	43.4%	48.2%	52.8%	57.3%	61.3%
0.76	41.0%	45.7%	50.2%	54.6%	58.7%
0.83	35.5%	40.3%	44.8%	48.9%	53.0%
0.88	31.9%	36.7%	41.2%	45.3%	49.2%

Post-Tax NPV Sensitivity To Opex

Opex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	$1,260	$1,444	$1,628	$1,812	$1,995
(10.0%)	$1,152	$1,336	$1,520	$1,704	$1,887
--	$1,044	$1,228	$1,412	$1,596	$1,780
10.0%	$936	$1,120	$1,304	$1,488	$1,672
20.0%	$828	$1,013	$1,196	$1,380	$1,564

Post-Tax IRR Sensitivity To Opex

Opex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	45.9%	50.4%	54.7%	58.9%	62.9%
(10.0%)	43.5%	48.1%	52.5%	56.8%	60.8%
--	41.0%	45.7%	50.2%	54.6%	58.7%
10.0%	38.3%	43.3%	47.9%	52.4%	56.6%
20.0%	35.6%	40.7%	45.5%	50.1%	54.4%

Post-Tax NPV Sensitivity To Total Capex

Total Capex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	$1,178	$1,362	$1,546	$1,730	$1,914
(10.0%)	$1,111	$1,295	$1,479	$1,663	$1,847
--	$1,044	$1,228	$1,412	$1,596	$1,780
10.0%	$977	$1,161	$1,345	$1,529	$1,713
20.0%	$910	$1,094	$1,278	$1,462	$1,646

Post-Tax IRR Sensitivity To Total Capex

Total Capex	Metal Price (US$/oz)				
Au:	$1,500	$1,600	$1,700	$1,800	$1,900
Ag:	$15.00	$17.00	$19.00	$21.00	$23.00
(20.0%)	52.4%	57.8%	62.9%	67.9%	72.6%
(10.0%)	46.2%	51.3%	56.0%	60.7%	65.1%
--	41.0%	45.7%	50.2%	54.6%	58.7%
10.0%	36.5%	41.0%	45.3%	49.4%	53.3%
20.0%	32.6%	36.9%	41.0%	44.9%	48.7%

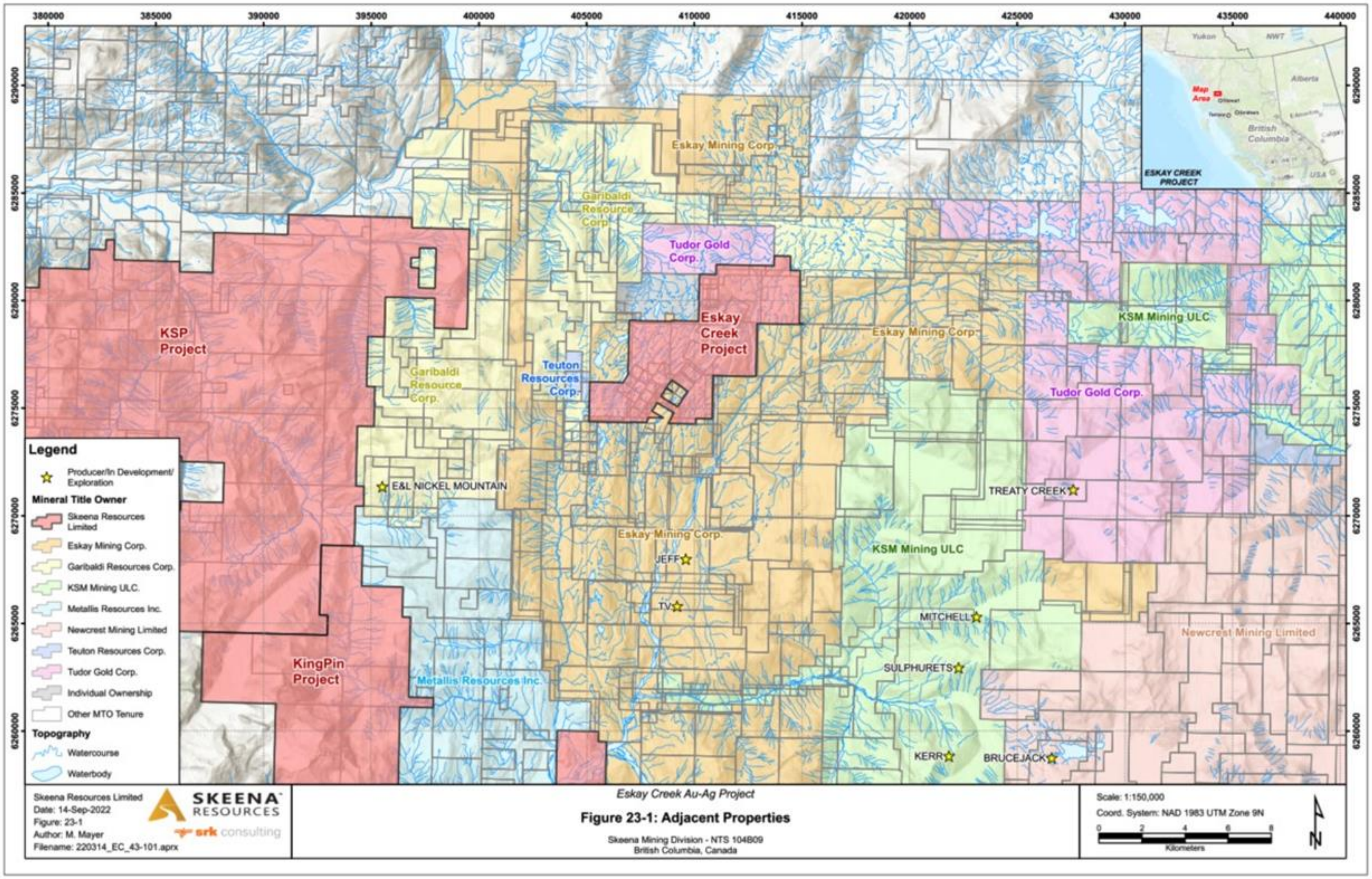
23 ADJACENT PROPERTIES

The adjacent properties to the Eskay Creek Project are shown in Figure 23-1 and Notable third-party properties are summarized in Table 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 Iskut River Region

<table>
<tr><th rowspan="2">Project Name</th><th rowspan="2">Owner/Operator</th><th rowspan="2">Status</th><th rowspan="2">Date</th><th rowspan="2" colspan="2">Classification</th><th rowspan="2">Tonnes (000)</th><th colspan="4">Average Grades</th><th colspan="4">Contained Metal</th><th rowspan="2">Source</th></tr>
<tr><th>Au (g/t)</th><th>Ag (g/t)</th><th>Cu (%)</th><th>Mo (%)</th><th>Au (MOz)</th><th>Ag (MOz)</th><th>Cu (Mlbs)</th><th>Mo (Mlbs)</th></tr>
<tr><td>Brucejack</td><td>Newcrest Mining Ltd.</td><td>Producer</td><td>March, 2020</td><td>Reserves</td><td>Proven & Probable</td><td>15.7</td><td>8.4</td><td>59.6</td><td></td><td></td><td>4.2</td><td>30.1</td><td></td><td></td><td>43-101 Technical Report. Shaw et al. (2020)</td></tr>
<tr><td>KSM</td><td>Seabridge Gold Inc.</td><td>Development Project</td><td>June, 2022</td><td>Reserves</td><td>Proven & Probable</td><td>2,292</td><td>0.64</td><td>0.14</td><td>2.2</td><td>76</td><td>47.3</td><td>160</td><td>7,320</td><td>385</td><td>Company website</td></tr>
<tr><td>Treaty Creek</td><td>Tudor Gold Corp. (60%)/American Creek Resources Ltd. (20%)/Teuton Resources Corp.(20%)</td><td>Exploration/ Development Project</td><td>March, 2021</td><td>Resources</td><td>Measured & Indicated
Inferred</td><td>609.8
139.4</td><td>0.65
0.72</td><td>3.2
3.6</td><td>0.06
0.04</td><td></td><td>12.7
3.2</td><td>63.2
16.29</td><td>770.5
113.7</td><td></td><td>Company website</td></tr>
<tr><th>Project Name</th><th>Owner/ Operator</th><th>Status</th><th colspan="12">Current Exploration Work</th><th></th></tr>
<tr><td>E & L Nickel Mountain</td><td>Garibaldi Resources Corp.</td><td>Exploration</td><td colspan="12">Significant exploration interest in 2017 on reports of nickel-copper rich massive sulphide mineralization intersected in 14 drill holes. In 2021 Garibaldi conducted deep penetration ZTEM survey across the entire Nickel Mountain - PSP Claim. The large ZTEM response expands the potential for further discovery beaneath and around E&L, as well as showed anomolous responses throughout the rest of the property, many of which correspond to surface mineralization.</td><td>Company website</td></tr>
<tr><td>Jeff/TV</td><td>Eskay Mining Corp.</td><td>Exploration</td><td colspan="12">In 2020, exploration consisted of a property-wide Skytem and IP surveys. In 2020 and 2021, approximately 28,000m were drilled at TV and Jeff. At least three significant sulphide-mineralized horizons were identified and drilling confirms these projects are stacked VMS systems. Post-season interpretation and petrography work carried out at the Colorado School of Mines has indicated that both the TV and Jeff zones are consistent with sub-seafloor replacement, and VMS feeder style mineralization, with significant Au and Ag enrichment. Exploration in 2022 includes soil sampling, refined geological models and target generation building off the 2020 and 2021 programs.</td><td>Company website and 43-101 report</td></tr>
</table>

Figure 23-1: Adjacent Properties



24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution and Organization

The Project Execution Plan (PEP) is a governing document that establishes the means to execute, monitor, and control the execution phase of the Eskay Creek Project. The plan will serve as the main communication tool to ensure the project team is aware and knowledgeable of project objectives and how they will be accomplished.

The following subsections summarise the contents of the Eskay Creek PEP.

24.1.1 Summary

The PEP includes, but is not limited to, the following:

- an overview of the project
- the scope of work and services
- execution strategy
- the project schedule with key activities and target dates identified
- an organizational chart

The PEP is supported by various sub-plans including, but not limited to, the following:

- Health, Safety and Environment Management Plan
- Procurement Management Plan
- Contracting Strategy Plan
- Construction Execution Plan
- Project Quality Plan
- Logistics Plan

24.1.2 Objectives

Skeena Resources aims to bring the Project into operation while satisfying the following objectives:

- zero harm to personnel involved with construction, operation, and maintenance of the facilities, and zero unintended environmental impact or incidents
- preserve or improve the project value through effective control of project costs and completion of construction and commissioning on or ahead of schedule
- satisfy quality and performance targets
- comply with company policies and legislative requirements, negotiated benefits agreements
- maintain positive community relations

24.1.3 Execution Strategy

Three contract strategies will be employed to deliver the detailed engineering and execution phases of the project:

1. Engineering, Procurement and Construction Management (EPCM) contract(s), led by an engineering consultant nominated by Skeena Resources , that generally encompasses the process plant and select on-site infrastructure
2. EPCM scope, led by an engineering consultant nominated by Skeena Resources, that generally encompasses site bulk earthworks
3. EPCM scope, led by Skeena Resources, that generally encompasses the development of the mining pits, off-site infrastructure, and permanent camp

These are described in more detail in the following subsections.

24.1.3.1 EPCM Scope Led by Engineering Consultant

This delivery strategy can be summarised as follows:

- Engineering and design for construction will be completed by the engineering consultant. Detailed design will start in early 2023 and be completed in early 2024.
- Procurement of equipment and services, expediting and contract management will be performed by Skeena Resources. The engineering consultant will advise Skeena Resources on vendor and contractor selection through production of specification and contractor packages and performing technical and commercial bid evaluations.
- Skeena Resources will continue to perform commercial management of contractors during construction. The engineering consultant will provide technical supervision and support on-site as required. The engineering consultant's site team will report to Skeena Resources' project manager.

24.1.3.2 EPCM Scope Led by Skeena Resources

Skeena Resources will manage select scope areas and engage delivery contractors as required to execute fixed scopes. Notable scope inclusions are as follows:

- mobile mining equipment selection and procurement

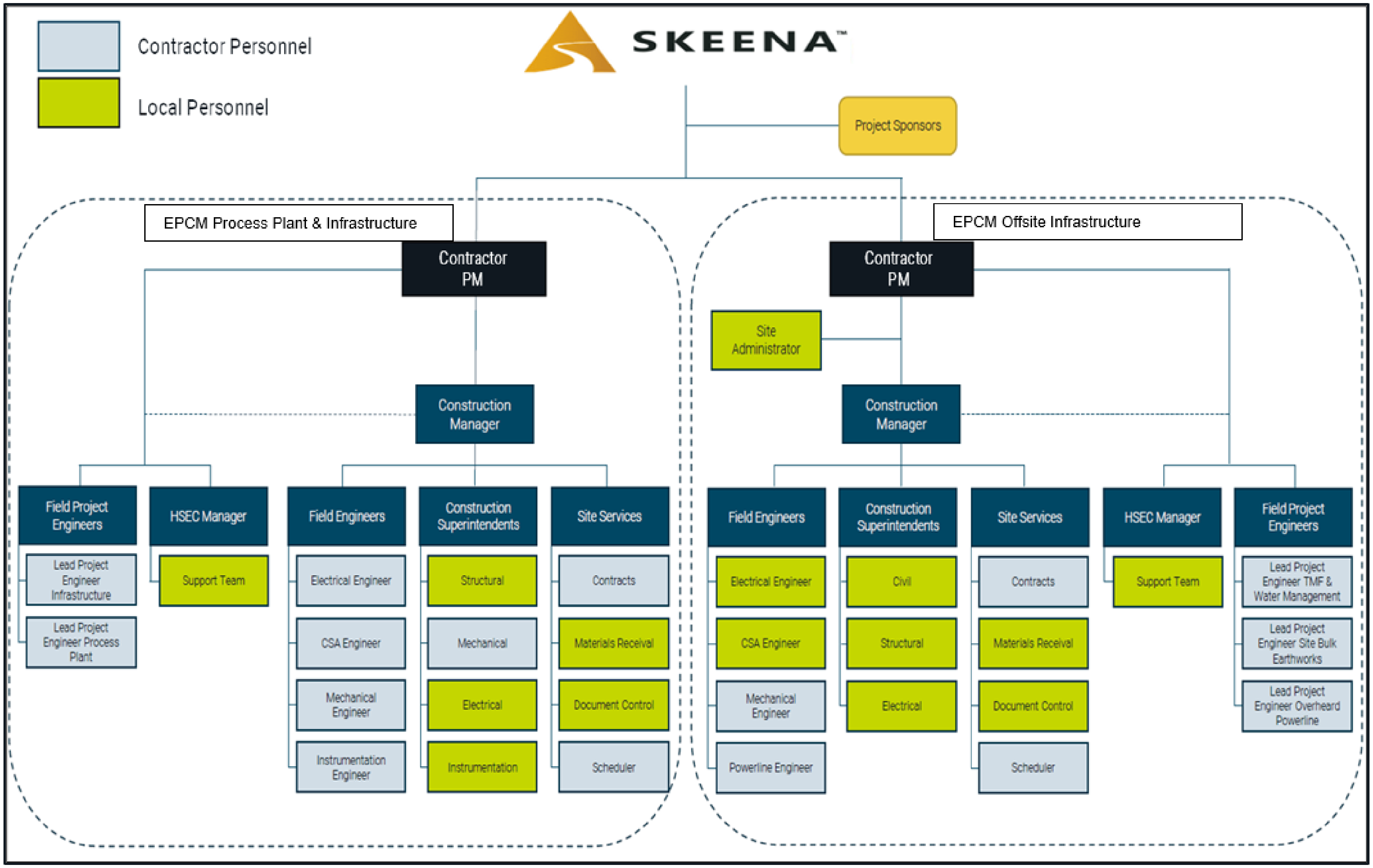
- mining pit detailed design and development and haul roads
- permanent camp design and procurement
- access road upgrades scoping and development
- high-voltage powerline to site permitting, engineering and development

24.1.4 Project Organization

24.1.4.1 Organization & Resourcing

The project team is organised based on an integrated team approach, minimising the duplication of roles and activities between the Owner's Team and their major delivery partners. A project organisation chart is shown in Figure 24-1 on the following page.

Figure 24-1: Project Organisation Chart



Skeena Resources will be performing or managing a considerable portion of the project scope, including the mine design, power transmission line, pit pre-stripping and delivery of certain construction materials to designated work sites. Key persons will be established on both teams at site to ensure efficient coordination.

24.1.4.2 Alignment Strategy

The project alignment strategy aims to create shared understanding of the project vision and strategy to enable Skeena Resources and its internal and external stakeholders to achieve the project objectives. The project delivery team will operate as one team with defined responsibilities, accountabilities, and authorities. The team will be established and supported to deliver "Best for Project" outcomes in-line with Skeena Resources' expectations and critical success factors.

Establishment of the delivery team working relationships and agreeing acceptable desired outcomes will be done in facilitated alignment sessions.

The alignment effort will be concentrated at the front-end of the project, although ongoing activities will be planned throughout to increase overall effectiveness, commitment, and cohesiveness of project team members.

24.1.4.3 Sponsor Group

A Sponsor Group will be formed to reinforce corporate commitment to the project as it passes through its various phases.

Key activities include:

- directing the business objectives for the participants to achieve 'best for project' outcomes
- providing corporate commitment to achieving the desired outcomes for the project
- reinforcing common purpose in achieving the project goals
- managing third party events outside of the control of the project team
- providing corporate recognition and reward for performance
- supporting the project in resolution of issues

The Sponsor Group will comprise senior executives from the EPC(M) contractors and Skeena Resources. The Sponsor Group will stay abreast of events and issues on and around the project. The principal responsibility of each member in their role on the Sponsor Group is directed at ensuring that the project is guided, supported, and encouraged to achieve the project objectives. Each member's association with their own organisation is secondary to their responsibility to support the project.

24.1.5 Construction Execution Strategy

An overall master execution schedule is included in Figure 24-3; however, this section will outline the high-level execution sequencing constraints that were evaluated in order to determine the execution schedule baseline for the feasibility study.

24.1.5.1 Construction Sequencing – Early Works and Technical Sample Phase

After completion of the Feasibility Study and with the start of working season, there will be a period of early works that will need to be completed prior to the first mobilization to site. These early works include the following main tasks;

- Environmental and construction permitting activities.
- Tree Clearing
- Access Road upgrades and bridge repairs.
- Award of key construction contracts (Camp Construction, Site Civil Works)

These early works activities will all be completed prior to first mobilization to site, which at this stage is anticipated to be April 2023. This date is predicated on Skeena Resources receiving their Joint Application No.1 (JA1) permit approval and filing and receiving any other appropriate environmental/construction permits to allow breaking ground to occur.

Once the permits are in hand, the first contractors to mobilize will be the Technical Sample civil works contractors responsible for the clearing and grubbing of the specific site works boundaries. It is critical that the clearing and grubbing contractors drop the trees in the specific site boundaries in the Winter before the migratory bird nesting window opens.

As the clearing and grubbing activities continue, the heavy civil work will follow behind to strip the topsoil and organics and stockpile in designated areas for future remediation works. Temporary water management catchments and ditches will also be developed as the civil works continue in constructing the main water management ponds, haul roads to access TMSF area, Technical Sample Pit and material quarries, temporary PAG stockpile, and the Process Plant pad development. The Technical Sample phase will also complete the construction of 69kV Right of Way (ROW) as well as completion of main laydown area for the project at KM2 laydown yard.

After the Technical Sample civil works are completed there will be four main work-fronts on the project property, which will be governed under the Joint Application No.2 (JA2) permit, governed by an Environmental Assessment (EA) process:

- The mining works will continue the pit development, generating and stockpiling waste rock material that will be crushed/screened via a contract crushing/screening plant and used for construction materials.
- The construction of MSE wall and ROM Pad will commence in preparation for construction of crusher plant and main conveyance system.
- The process plant and mining infrastructure area (MIA) concrete works will begin in preparation for pre-engineered building and SMP contractors to arrive for building/major equipment foundations, and
- The TMSF works which includes placing and compacting hauled waste rock to raise the starter dam wall and finishing with crushed/screened material and installing the geomembrane liner.as well as expansion and completion of the tailings and reclaim water pipelines along the TMSF haul road.

24.1.5.1 Construction Sequencing – Revitalization Project Construction Phase

The construction phase of the project will consist of:

- Mobilization and site establishment to the project site for the project execution team
- Staged mobilization and site establishment of Contractor Execution team and sub-contractor teams
- Establishment of site facilities such as accommodation camp, security gates, infrastructure upgrades, utility connections
- Completion of engineering design to the level of detail agreed upon in the contract
- Procurement of long lead items (mining fleet, SAG/secondary mills, ADR Circuit)
- Award of transport and logistics contract to transport materials from KM 2 laydown to site
- Award of remaining construction contracts
- Stakeholder engagement

The timing of the construction phase is dictated by Skeena Resources receiving their permit approvals and filing and receiving the appropriate environmental/construction permits.

24.1.5.1 Winter Construction

The construction duration for this project will see works continue through the 2023/2025 winter periods. To mitigate downtime and loss or productivity the following considerations were included in the execution schedule.

The concrete works for the process plant are, for the most part, scheduled to be within the summer months. The construction sequence for the process plant is such that the Process Plant and Truckshop pre-engineered buildings will be fully constructed and cladded prior to the winter weather period. This will allow installation works to continue within the buildings, sheltered from any inclement weather. Priority will also be given to erect the contractors' temporary warehouse and fabrication buildings for additional all-weather storage for the winter months.

24.1.5.1 Site Laydown Requirements

An early priority for site construction should be the assembly of temporary and permanent storage warehouse facilities with sufficient space to store any goods with indoor storage requirements.

The primary Project marshalling yard and primary staging/ laydown for construction materials, will be the KM 2 Laydown area, approximately 2.0 km from in from Highway 37 junction with main access road and approximately 50km away from site. This location is near the existing KM 2 security checkpoint allowing for continuous security coverage. The area requires grading and leveling to make functional but provides a transition point for offsite deliveries located near highway 37.

Any goods or equipment that can be stored outdoors may be placed in the outdoor lay down area at KM 2. The outdoor lay down area will have to be on level ground, with all snow removed prior to arrival of goods and equipment.

Contractor and vendor offsite deliveries will stop at the security checkpoint, confirm delivery and proceed to the KM 2 laydown, unload and depart. This limits unfamiliar personnel and transports on the main access road. Contractor personnel familiar with the access road and the site will attend the KM 2 laydown, reload and deliver materials to site in accordance with execution schedule.

The existing buildings in the historical camp site will be utilised as a storage warehouse during construction as well as the new warehouse within the process plant area when it is constructed as part of the erection of the process plant pre-engineered building.

The Plant site plateau and specifically the pad for mining infrastructure area provides a laydown area (approximately 1.5 Ha) near the Plant site. This area will be required for plant site and site services contractor facilities and equipment.

Both the site lay down and storage warehouse will need to obtain the necessary authorizations for the storage of any hazardous materials. The required security, protective and handling equipment should be on hand to allow for the temporary storage of hazardous materials whenever necessary.

A designated laydown area will be developed and prepared for installation and operation of crushed aggregate plant. This laydown area will be in proximity to the NAG waste rock storage area, and the NAG quarries. The same laydown area is designated for storage of processed aggregates, including crushed granular material and concrete aggregates, as well as installation of the future batching plant to supply the required concrete for project construction. This is based on the strategy that the crushed aggregates plant will be able to produce the required processed material during year (-3), and then be decommissioned (or relocated) to clear enough space for batch-plant installation at the start of year (-2).

24.1.5.2 Existing Facilities (Historical Site)

The execution strategy is based on maximizing the utilisation of the existing infrastructure and facilities at the historical site. This includes:

- Offices and Lunchroom: the existing admin building will be utilised for Skeena/EPC(M) project management teams; the contractors will provide their own offices and lunchrooms.
- Camp: the existing camp and ancillary facilities (kitchen, dining room, sewage treatment plant, incinerator) will be used initially and a new camp and facilities will be established at KM 37 (near the existing Forrest Kerr Infrastructure) as the workforce levels ramp up (as shown in the camp histogram in Figure 24-2). The existing camp will be relocated during operations to supplement camp requirements during operation.
- Warehousing: some of the existing buildings in the historical camp will be used for heated and unheated warehousing during project execution
- Water Treatment Plant and Polishing Ponds: the existing facilities will be used (with some minor upgrades) and continue to support the dewatering of underground mining area, as well as act as the permitted discharge point during the coarse of the construction.

24.2 Construction Facilities

Temporary construction infrastructure will be minimized by utilizing 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 camp/accommodation.

24.2.1 Camp Requirements

The accommodation strategy has accounted for the labour requirements over the period of construction, existing camp capacity and additional camp capacity to be added during the construction period. Peak levels and surge capacity has been also accounted for.

There is currently an existing operating camp on the project property (at KM58) with a total capacity of 230 beds, which is currently being used (in part) for the drilling and exploration workforce. A single 210 bed temporary camp will be built (at KM 37) and utilized for both the construction phase and first three years of operations, thereby providing a total of 440 beds available. To further derisk the project, additional beds will be available at the existing Coast Mountain Hydro Forest Kerr camp located at KM 37, near the new camp (up to 80 beds) to account for any additional beds (on a temporary basis) that are required on a short term rental basis – similar to what is indicated during Year -2 as shown in Figure 24-2.

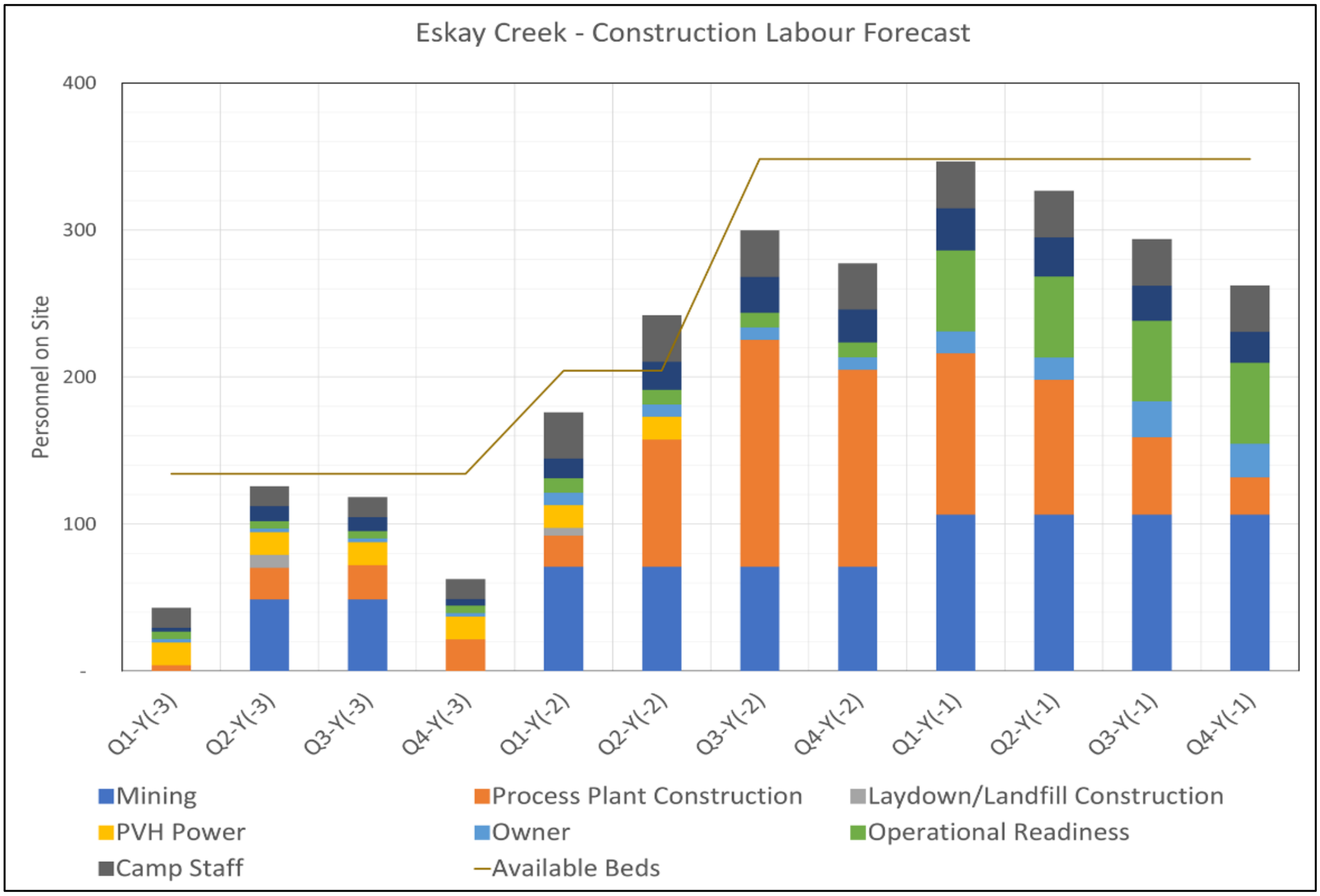
The existing camp will be used for Early Works and Technical Sample Phase, as well as providing additional capacity during construction to the new 210-bed temporary camp. During the initial phase of construction, the exploration activities will cease for a few months to allow the temporary camp construction crew and clearing and grubbing workforce to lodge, while the temporary camp is being built. The temporary camp will be operational at its full 210-person capacity in two phases: Phase 1 with 160 beds prior to the ramp up of construction personnel in Year (-2), and after receipt of the EA Certificate; Phase 2 will be an additional 50 beds which will be added after construction is over, during the first year of operation, to allow for gradual transfer of existing facilities from the historical site, to the process plant area.

24.2.1.1 Construction Staffing

A labour loading forecast was developed for the construction phase (see Figure 24-2 on the following page) and operations phase. The forecast was developed utilising labour hours received from contractors who provided budgetary pricing for the feasibility study, as well as from organisation charts for the construction management teams from both the owner and the engineering firms.

During the peak of the construction labour requirements the persons on-site may exceed the capacity of the permanent camp. In this case, the burden may be reduced by utilizing additional space in the Forest Kerr camp, renting additional temporary trailers, or re-directing indirect personnel to off-site facilities.

Figure 24-2: Camp Requirements During Construction Period (Ausenco 2022)



24.2.2 Lunchrooms and Offices

All contractors will be responsible to provide and maintain their own facilities to support their personnel such as offices, lunchrooms, etc. The Owner will provide space for each contractor to setup their own facilities. The Owner will also provide camp accommodation facilities for their personnel.

24.2.3 Maintenance Facilities

The existing maintenance facilities at the historical site will be used for mobile equipment maintenance until the permanent truckshop is operational.

24.2.4 Security

The common security at KM2 of the access road is in place and will remain throughout the Project. The permanent Skeena Resources specific security facility (at 54.5KM) will be installed early in the construction program.

24.2.5 Emergency Response

The existing emergency response facilities at the historical site will be used until the permanent facilities are available. Existing buildings will be used to house the ambulance and fire truck during construction.

24.2.6 Medical Facilities

The existing medical facilities at the historical site, supplemented with temporary medical facilities at the plant site area, will be utilized until the permanent medical facilities are operational.

24.2.7 Warehousing and Temporary Shelters

The existing facilities at the historical site will be used during construction and the permanent warehouse will be used when erected in Summer 2024, which is part of the process plant building located at the process plant site

24.2.8 Water supply

The construction freshwater wells for the site will be installed during Yr-3 to provide replacement water source for the Historical site and construction water source.

24.2.9 Fuel supply and storage

The existing fuel storage facility on site which can be augmented with a fuel truck and individual double walled storage tanks until the permanent fuel storage and dispensing facility is operational. Bulk fuel will be delivered by truck from local suppliers in Terrace, Smithers or Prince George. Fuel will be supplied to the contractors and managed through a FOB system.

24.2.10 Waste management and sewage treatment

The waste management and sewage treatment systems will be installed during early stages in the project and will become operational with the commissioning of the accommodations facility.

Construction waste products will be separated, which can be disposed of in the site landfill and wastes that must be disposed of off-site in regulated disposal or recycling sites. Products requiring special off-site waste handling facilities will be temporarily stored on site at a lined waste transfer station until transported off-site.

The sewage treatment facility will have an external connection for offloading the sewage vacuum truck. Maintenance and operation of the sewage treatment plants will be the responsibility of the site services contractor. Sewage from process plant area will be stored in sewage storage tanks, and transported to the sewage treatment facility for treatment, as required.

24.2.11 Communications

The permanent site communications system will be installed in the first year of construction.

24.3 COVID-19 Considerations

Provincial guidelines and best practices have been followed to develop the working procedures pertaining to COVID-19. At the time of writing, there are no mandatory testing or isolation requirements for COVID-19, therefore there is no allowance for COVID-19 related costs.

24.4 Shared Site Services

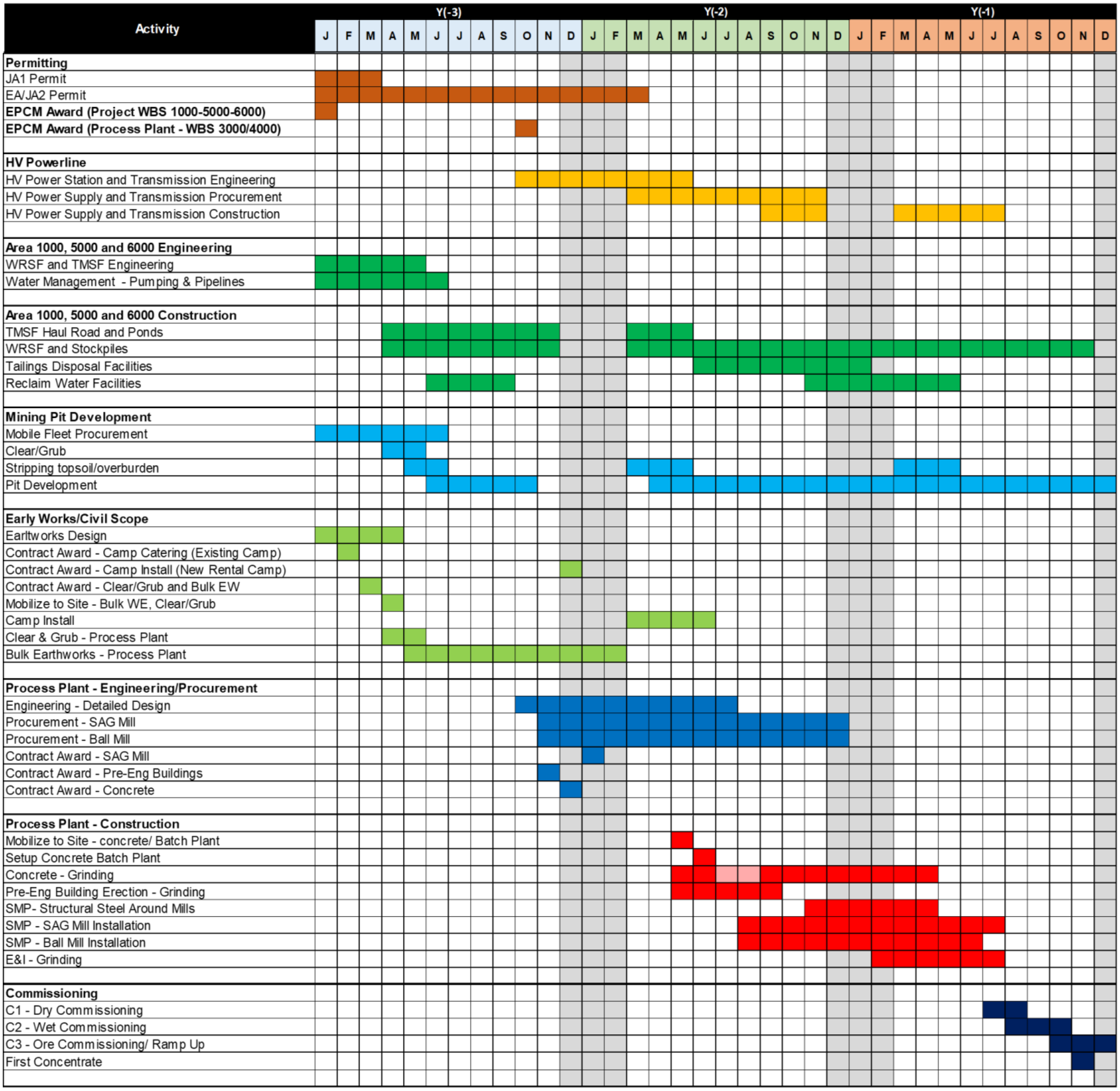
A number of services were identified during the feasibility study that were common across the work fronts during construction. It may be advantageous to offer these common services to the contractors both from a cost perspective, as well as to allow site service contracts to local businesses. These services include:

- diesel fuel supply
- temporary power supply
- road maintenance/snow clearing
- garbage removal
- bussing workforce to/from the camp each day
- upfront purchase or lease of mobile equipment that will be required by operations that can be free issued to the construction contractors for use during construction

24.5 Execution Schedule

The preliminary project execution schedule is shown in Figure 24-3.

Figure 24-3: Eskay Creek Project Execution Schedule



Note: figure prepared by Ausenco, 2022.

25 INTERPRETATION AND CONCLUSIONS

25.1 Property Description & Location

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 several 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 additional surface rights in support of planned 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.

25.2 Geological Setting & Mineralisation

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.

25.3 Exploration, Drilling, Sample Preparation Analyses & Security

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 QA/QC 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.

25.4 Mineral Processing & Metallurgical Testing

Recent testwork by Base Met was conducted on a wide range of samples, collected from every ore zone and separately from Rhyolite and Hanging wall/Mudstone lithologies. The variability sample program confirmed the suitability of the MF2 flowsheet design and revealed opportunities for a coarser primary grind as well as secondary grind of the deslimed rougher tailings material.

Comminution tests have now been completed on a reasonable number of samples of each ore zone and provide confidence in the grinding circuit power requirements for the initial and expansion plant design capacities. Regrind mill testing has continued on both rougher concentrate and deslimed rougher tailings samples; this work will continue in the next phase.

Hanging wall/Mudstone samples showed poor cleaner flotation response that appeared to be somewhat mitigated with additional collector and reductions in applied grinding. Although the Mudstone samples showed levels of organic carbon (C_{org}), this was not in the form of graphite.

Dewatering of the final concentrate to below TML levels of moisture now includes the option of supplementary drying, as some samples have shown slow filtration rates and tendency to generate thin filter cakes.

Results from the 2022 FS testwork program were used to develop separate recovery equations for Rhyolite and Hanging wall/Mudstone material.

Additional testwork is planned to further investigate flotation conditions for Hanging wall/Mudstone samples (particularly high in C_{org}), and flowsheet configurations that may improve concentrate quality and dewatering performance. Testing is required to confirm regrind mill specific energy requirements and larger scale filtration performance of representative final concentrate. Additional samples will be collected to increase confidence in the current recovery equations and confirm the response to a blend of Rhyolite and Mudstone material. At least one bulk sample will be pilot tested to generate large sample masses of regrind mill feed and final concentrate for downstream testing.

25.5 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.

25.6 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.

Ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries, the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

25.7 Mining Methods

25.7.1 Geotechnical Considerations

The current geotechnical dataset is considered adequate for feasibility study 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 26–51º based on kinematic sectors. 20 m high double benches are likely achievable in all sectors, with recommended catch bench widths ranging from 12.7 m to 37.5 m. The slope design criteria assume that controlled blasting will be implemented. Scaling bench faces and cleaning accumulated material from bench toes is recommended. Slope depressurization will be required in the north, east and south walls of the North pit to meet the design acceptance criteria in these 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.

25.7.2 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 650 masl, and therefore only the upper flooded workings are 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.

25.7.3 Mine Plan

Each pit phase was designed to accommodate the proposed mining fleet. Mining will occur on 10 m benches with catch benches spaced 20 m vertically. Berm widths will vary depending on lithology type and kinematic sector. The haul roads will be 30.2 m in width with a road grade of 10%.

The mine schedule plans to deliver 29.9 Mt of mill feed grading 2.99 g/t gold and 78.5 g/t silver over a mine life of eight years. Waste tonnage from the pits totalling 223 Mt will be placed into either NAG or PAG waste destinations. The overall strip ratio is 7.5:1.

The mine schedule initially assumes a maximum of 3.0 Mt/a of feed will be sent to the process facility using a suitable ramp-up in year 1. The mill throughput is increased to 3.7 Mt/y in year 6 and continues until year 9. To maintain these throughput tonnages, a minimum proportion of 55% rhyolite tonnes in the mill feed was maintained in the schedule periods. 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 proposed mine life includes three years of pre-stripping and 8 years of mining. Mill feed will be stockpiled during the pre-production years. A technical sample and two small quarries will be mined in pre-production so that process performance of the mill can be evaluated with a large representative feed sample of approximately 10 kt.

The northern end of the open pit will intersect Tom MacKay Creek requiring the provision of a water diversion tunnel to re-route flowing water around the open pit before re-entering the existing river downstream. This water diversion is currently required to be operational in year 5 of the production schedule. Minimum tunnel dimensions have been selected as 4.7 metres wide by 4.7 metres high to accommodate the expected water flows. The full length of the tunnel is 1214 metres. Starting from the tunnel inlet, 802 metres are at -2% gradient, 362 metres at -18.5% gradient, and 50 metres at -2% nearest the outlet.

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.

25.8 Recovery Methods

The plant will process material at a nominal rate of 3.0 Mt/a for years 1 to 5 and 3.7 Mt/a for the remaining years with an average head grade of 3.0 g/t Au and 79 g/t Ag. The ore becomes harder and more competent after the first three years of operation therefore a pebble crusher is installed to start operations at the beginning of year 4 to maintain 3.0 Mt/a capacity prior to the expansion that will begins operation at the beginning of year 6.

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.

25.9 Project Infrastructure

25.9.1 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.

Bulk transportation of concentrate will be done by tandem, side dump tractor-trailers at 72,000 kg GVW to SBT, then bulk load out onto ocean-going vessels using shiploading infrastructure.

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.

25.9.2 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 has sufficient capacity to contain 109.4 Mt of NAG and PAG tailings and PAG waste rock and will be constructed in three phases over the LOM based on storage and operating criteria. TMSF has been design in accordance with both CDA guidelines (2013) and Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (2016).

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 berm approach, depositing waste rock across the facility from west to east then the upper 5 metre will be removed and deposited on the south side of the berm for the PAG waste rock is submerged a minimum of 3 m below the water surface. Floating turbidity fences will be utilized to prevent the migration of temporarily fine grain suspended fine grain particles suspends solids downstream.

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 using a manifold to promote settling of the tailings by reducing the discharge velocity. In addition, at the edge of the TMSF, a flocculant dosing station will be installed and inject flocculant from the family polyacrylamide (anionic) into the tailings pipeline to promote settlement along with a floating turbidity fence around the discharge point.

25.9.3 Water Supply

Potable water supplies can undoubtedly be met by groundwater; however, sourcing all the freshwater makeup (27 to 33 L/s) from bedrock aquifers may require establishing wellfields at significant distance from the processing plant; from 10 to 15 wells would potentially be required. Pumping the bedrock wells at high flow rates may not be sustainable over the life of the mine and require additional wells in addition to regular maintenance to clear biofilm or other flow impedances. The TMSF could provide an alternative freshwater makeup source; however, the water would potentially require treatment to make it suitable for reagent mixing, gland service and other uses in the process plant.

25.9.4 Snow Management

The purpose of the snow management plan is to provide technical directives for handling, removal, and management of snow for the Eskay Creek Project. Snow volumes that need to be managed would depend on the active areas of each facility and climate conditions. Based on climate condition and facilities' active areas (operational schedule), the following general recommendations should be considered for snow removal practice:

- In extreme snow conditions with significant 24 hours snow depth, the top clean snow may directly be removed and deposited to the nearby creeks for each facility (Tom MacKay Creek, Eskay Creek and Argillite Creek). However, this option may require an environmental study to assess the impact of bulk clean snow disposal into a waterbody.
- Where practical, snow should be removed in the same direction as the prevailing winds of the area. This will minimize the amount of drifting that occurs across laydowns and roads during the winter months.
- Pit benches that advance slowly or are inactive can be used to store snow as much as possible.
- It is assumed that snow during WRSF construction can be managed on-site by using a leap-frog method (moving snow from active areas and depositing it on inactive areas).
- Snow on the stockpile and the primary crusher & ROM Pad is considered in contact with PAG material. The first option to manage its snow is to use inactive areas of low-grade stockpile for snow deposition, and its snowmelt should be collected in pond 5.
- In normal operational conditions, any snow on Haul Road should be removed frequently to maintain the flow of traffic. It should also be noted that all the road snow cannot be stored along the Haul Road to the TMSF during the snow season since it is not wide enough. Therefore, its snow could either be transported to the TMSF or deposited on the WRSF's inactive area.

25.9.5 Power

The power supply for the Project will be provided from the 287kV Volcano Creek interconnection point, where a new 287/69kV substation will be installed and a 17km, 69kV overhead power line will be run to the mine to feed an estimated operating load of 26.5MW at the base case.

25.10 Environmental Studies, Permitting & Social or Community Impact

25.10.1 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; and monitor and manage water quality to meet discharge standards prior to discharge.

25.10.2 Water Management

The objective of surface water management is to protect groundwater and surface water resources. Feasibility Study infrastructure and upstream catchments for the Project were delineated based on topography data and footprints of facilities provided. Contact and non-contact water are managed separately for the Project. The site water management has been designed in accordance with BC regulations.

Water management at the waste rock storage facility (WRSF) includes both contact and non-contact structures. Non-contact water will pass underneath the facility in a rock drain into the tail end of Argillite Creek and then into Tom MacKay Creek. Runoff from the WRSF will be captured in collection ditches and discharged into Contact Water Pond 5. The WRSF contact water management system was designed for 1:100-year storm event with 0.3m freeboard and pass the 1:200 year storm event within the confines of the structures. and the non-contact water management system for 1:2,475-year storm event.

The contact water management for the plant site include diversion ditches that direct runoff to Contact Water Pond 6. Water from Pond 6 will be pumped to Pond 5. Contact water and dewatering from the open pits will be pumped to Pond 5.

Pond 5 is design to capture the 1:100 year storm will pumping at the maximum pumping rate during the storm event. Contact water from Pond 5 will be recycled to the process plant and any excess water would be discharged into TMSF with the slurry tailing.

There are no diversion works at the TMSF and there will be runoff from the surrounding catchment into the TMSF. A penstock will be used to maintain flow in Tom MacKay Creek and a 3 m water cover over waste rock and tailings.

All mine roads will have collections ditches to capture contact surface runoff from the roads. The ditches are design to convey the 1:100 year storm event with 0.3 m of freeboard. In addition, they are designed to convey within the limits of the ditch the 1:200 year storm event. These channel discharge into Sediment ponds located strategically long the roads. The ponds are design to capture 10 micron or greater from the 1:10 year storm event and safely pass the 1:200 year storm event.

25.10.3 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.

25.10.4 Permitting Considerations

The proposed Project is anticipated to undergo a substituted EA process to address both provincial EA and federal IA requirements as well as Tahltan Nation requirements and consent. Since the Eskay Creek Mine has two existing Certificates, one or both will be amended through a EA/IA process or a new EAC issued. 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 and consent requirements. In addition to obtaining the EAC, the Project will require permits and authorisations in accordance with provincial and federal legislation and regulations prior to construction and operation. Permits for the technical bulk sample do not require the EAC and will proceed in advance of the EAC review. The issuance of the EAC will enable obtaining new and/or amended construction and operation permits for the open pit mine operation in collaboration with Tahltan Nation review. Consequently, Skeena will apply for permits during the EA/IA review process for subsequent new and/or amended construction and operation of the open pit operation 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.

25.10.5 Stakeholder Considerations

Community and socio-economic effects of the Project can potentially be very favourable for the region, particularly with effective mitigation and management of the social and community challenges endemic to development of an industrial project in northwestern BC, 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 implementing a thorough Engagement Plan with communities and Indigenous Nations for the Project as required by the provincial and federal EA processes and considering their preferences. Engagement and consideration of the preferences, interests, concerns and unique interaction of the communities and Nations will address potential risks and help develop collaborative approaches. Ongoing and future engagement and consultation measures by Skeena are driven by best practices as well as Skeena's internal company policies and Indigenous Nation preferences and requirements. These measures will achieve federal and provincial information needs and regulations as well as Indigenous requirements and interests.

Skeena recognises 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 will continue engagement with local Indigenous Nations to gain that support, yet also recognises 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 treaty) 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 government groups, affected communities and Nations. 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.

25.11 Capital & Operating Costs

AACE Class 3 costs have been developed for this feasibility study with an accuracy of ±15%. The cost estimates were derived from first principles bulk material take-offs and equipment sizing calculations, with supporting quotations for major equipment, and contractor supply/installation rates to the value of 94% of the cost estimate, with the remaining cost items benchmarked against recent Canadian mining projects.

The estimate is prepared in the base currency of Canadian dollars (C$), where relevant exchange rates were used to convert to Canadian currency, is expressed in Canadian dollars, has a base date of Q1, 2022, and has an accuracy range of +/-15%.

LOM Project capital costs total $911 M which can be broken down as follows:

- Initial capital cost includes the costs required to construct all the surface facilities, and open pit development to commence a 3.0 Mt/a operation. The initial capital cost is estimated to be $ 592 M.
- Expansion and sustaining capital costs are estimated to be $180 M.
- Closure costs: include all the costs required to close, reclaim, and complete ongoing monitoring of the mine once operations conclude. Closure costs total $138 M.

25.12 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, operating costs will average $170.2 M/a, and $51.24/t processed.

25.13 Economic Analysis

The economic analysis was performed assuming a 5% discount rate, $1,700 / oz gold price, $19 / oz silver price. The pre-tax NPV discounted at 5% is $2,094 million; the internal rate of return IRR is 59.5%; and payback period is 0.99 years. On an after-tax basis, the NPV discounted at 5% is C$1,412 million; the IRR is 50.2%; and the payback period is 1.0 year. The sensitivity analysis revealed that the Project is most sensitive to changes in gold prices and less sensitive to operating costs, discount rate and initial capital costs.

25.14 Other Relevant Data

25.14.1 Risks

25.14.1.1 Overview

Risk identification and mitigation was ongoing throughout the feasibility study, and will continue through value/detailed engineering, construction, operations and closure. Risks were identified and qualitatively ranked in the Eskay Creek Project Risk Register. As the Project moves from feasibility into the execution phase, it will be necessary to update the project risk register.

The objective of this process was to undertake a risk analysis in a workshop environment utilising expert input from consultants, engineering firms and Skeena Resources representatives. The purpose was to capture the results in a Risk Register that can be utilised for ongoing project risk management.

Risks identified were grouped into the following categories:

- Health & Safety
- Project Schedule
- Financial
- Environmental
- Reputation
- Legal

Table 25-1 summarizes the risk criteria and Table 25-2 summarizes the risk likelihood used to evaluate the project risks.

Table 25-1: Risk Criteria

Consequence	1: Minor	2: Moderate	3: Serious	4: Major	5: Critical
Health and Safety	Low level symptoms requiring first aid treatment only	Medical treatment injury	Serious injury/permanent disability or impairment to one or more persons	Single fatality or severe permanent impacts to >10 persons	Multiple fatalities as a result of short or long-term health effects or irreversible impacts
Project Schedule	< 1 day	1 - 7 days	1 - 4 weeks	1 - 3 months	> 3 months
Financial	< $400k US	$400k – $2.5 Million US	$2.5 – 10 Million US	$10 – 20 Million US	>$20 Million US
Environmental	Limited damage to a localised area. No lasting effects	Localised short to medium term damage to an area of minor local significance	Localised medium term damage to an area of local value	Wide spread long to medium term damage to valued area	Significant, extensive detrimental long-term impact
Reputation	Local public concern/complaints. Minor technical noncompliance	Negative publicity and attention from local media. Moderate breach of regulations	Attention from media, negative regional publicity. Serious breach of regulations with fine.	Significant negative attention, national publicity. Major breach of regulation. Reputation tarnished	Negative international publicity. Very serious litigation. Reputation severely tarnished. Share price may be affected
Legal	Minor noncompliances and breaches of regulations	Minor legal issues, moderate noncompliances and breaches of regulations	Serious breach of regulation with prosecution or moderate fine possible	Major breach of regulation. Major litigation	Significant prosecution and fines. Very serious litigation including class action

Table 25-2: Risk Likelihood

Likelihood	1: Rare	2: Unlikely	3: Possible	4: Likely	5: Almost Certain
Description	The event is not expected to occur in most circumstance 1% – 10%	The event may occur in exceptional circumstances 10% - 25%	The event could occur at some time 25% - 75%	The event will probably occur at some time 75% - 95%	The event is expected to occur in most circumstances 95% - 100%
Anticipated interval between events	Have never heard of this happening	The event might occur once in your career	The event or similar has occurred elsewhere	The event has occurred several times or more in your career	Occurs more than once per year

The methodology adopted for this risk analysis was in accordance with the best practices of risk management standards. Risk identification is the most important part of the process by which risks are identified based heavily on "expert judgement". Quantified evaluations of likelihood and consequences are captured in the workshop environment under the guidance of the risk facilitator. From these likelihood and consequence scores, a risk score was assigned. Following this, mitigation strategies were identified to reduce the likelihood or impact of the risk, thus reducing the risk score to a more acceptable level.

25.14.1.2 Geology and Resource Estimates

The current understanding of the distribution and variability of the suite 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 more information obtained from future 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 2022 FS mine plan and concentrate marketability assumptions are based on.

25.14.1.3 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 mineralised 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.

25.14.1.4 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.

Further data collection and interpretation tasks are recommended to fill in geotechnical data gaps to support future stages of design. A numerical groundwater flow model should be calibrated and developed under transient conditions to inform subsequent geotechnical evaluations and depressurization assumptions. Supplementary laboratory strength data, particularly in the Hanging Wall Mudstone, Contact Mudstone, Footwall Sediments and Bowser Sediments units where existing laboratory data is limited. Additionally, discontinuities were not systematically tagged by structure type during the 2020 and 2021 drilling programs, so it was not possible to develop relationships between discontinuity type and shear strength. There is a risk that pit slope angles may need to be shallower in some sectors of the pits.

Limited field data was available near the diversion tunnel area. An in-situ stress study in the diversion tunnel area and an analysis of borehole breakouts from televiewer surveys may provide information on in-situ stresses and lower the risk of tunnel or portal failure.

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.

Grade control and mining near the ore contacts present a risk of potentially mining too much dilution material or losing high-grade material. Performance of grade control methods and mining techniques should be continually evaluated to manage this risk.

The WRSF design assumes that no geotextile liner will be required as primarily NAG rock will be sent to the storage facilities. If, with further data, such a liner is required, this will affect the mining capital cost estimate.

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.

25.14.1.5 Mineral Processing

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.

The process design assumed for the 2022 FS 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:

- Performance of specific equipment such as filters and thickeners performing less than proven testwork results will be mitigated by further pilot level testing.
- Gold and silver recoveries not achieving proven testwork levels which will be mitigated by further geometallurgical testwork.

25.14.1.6 Infrastructure

Presently, there is insufficient water quality data for TMSF to determine the water quality discharging into Tom MacKay Creek during operations and post closure. Therefore, a water treatment plant at the outlet of the TMSF to treat water before discharging into Tom MacKay Creek has not been included in the scope of the 2022 feasibility study. Furthermore, on going testing will be done in detail design to confirm there is no requirement for water treatment. If required, this would affect the water-related capital and operating cost assumptions in the 2022 feasibility study.

If the bedrock from the 2022 Geotechnical Investigation shows the bedrock to be highly fractured requiring grouting, this would affect capital cost and sustaining capital cost assumptions in the 2022 feasibility study.

A PAG waste rock deposition plan into the TMSF was developed for the 2022 feasibility study. A detailed operating procedure will need to be established in the detail design 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 increased capital and operating costs.

Additional testing is required to understand settling times of tailings and fine particle material from the waste rock placed into TMSF, i.e. a large scale tailings deposition model that will also look at different discharge manifold configurations and flocculant dosing requirements. 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 fences will be used to reduce the potential for turbid water to discharge from the facility. If the large scale deposition simulation indicates 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 berm 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.

25.14.1.7 Environmental, Permitting and Social

The provincial and federal regulatory processes under recent legislative changes may influence overall timelines to amend the existing permits, address Indigenous consent and collaboration needs, and obtain new permits for the Project, including the EAC as well as construction and operating permits. 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 pre-feasibility study or for the feasibility study scale open pit project. 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 and access routes pass through lands subject to the Nisga'a Final Agreement treaty. 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.

25.14.1.8 Marketability

Smelter terms are market dependent and may be less favorable in the future negatively affecting the economics.

25.14.2 Opportunities

25.14.2.1 Exploration

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

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 mineralisation. 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. The recently discovered 23 Zone hosted in the Dacite of the Lower Package also has the potential to be expanded with further drilling.

25.14.2.2 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 test work.

25.14.2.3 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) discovery of additional mineralization that may support Mineral Resource estimation.

The 2022 drill program focuses on expanding mineralisation in the recently discovered that is proximal and lateral to currently defined mineralization domains, to generate maximum ounces with minimal cost. In addition, there is potential to increase the Mineral Resources in the 23 Zone with additional infill drilling and mineralisation 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.

25.14.2.4 Mining

With additional detailed metallurgical test work information on lithologies and zones, the mining sequence and blending strategy may be altered to provide concentrate higher value. The south pit is generally considered to have harder ores, but additional hardness testing may provide further basis for more detailed throughput management and potentially higher value.

There is potential for steeper slope design, when additional geotechnical data such as waste rock strength and joint orientations, are available from drill testing. Development of a numerical groundwater flow model should be calibrated and developed under transient conditions to also inform subsequent geotechnical evaluations and depressurization assumptions. Steeper pit slopes would reduce the cost associated with waste stripping and provide an opportunity to improve economics.

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 should be monitored to see if a portion of the PAG waste material can be effectively neutralised 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. Effective definition of PAG areas during mine operations will provide better PAG material management destination options. It will also improve the confidence in segregation when assigning more NAG waste to waste facilities other than the TMSF.

25.14.2.5 Mineral Processing

Improvements in concentrate quality and dewatering performance may be obtained by modifications to the flowsheet that optimize the application of regrinding energy.

Some rock types, particularly mudstones, may produce better metallurgical performance at coarser grind sizes, positively affecting the economics.

A more comprehensive understanding of grinding requirements as a function of feed characteristics may present opportunities to optimize the installed comminution power, positively affecting the economics.

Further investigations into geo-metallurgical relationships may provide a greater accuracy in forecasting metal recoveries, providing better support for process operations leading to increased recoveries, positively affecting the economics.

Initial drilling of Albino Lake, subaqueous repository for mine waste rock and tailings used by the previous operators, has indicated elevated gold values in this material. Test work 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.

25.14.2.6 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.

The potential for additional modularization of infrastructure items such as pre-engineered and modular buildings, over and above what has been nominated to this point, should be considered to reduce construction durations and costs.

25.14.2.7 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 NAG/PAG classifications and material segregation may help optimize waste management costs, design, and complexity. Ongoing geochemical studies will 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 the 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.

25.14.2.8 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 through ore blending or improved process rejection to below smelter penalty thresholds, while maintaining a payable range for Gold and Silver.

26 RECOMMENDATIONS

26.1 Overall

The financial analysis of this feasibility study demonstrates that the Eskay Creek Project has robust economics, and it is recommended to continue developing the project through engineering and de-risking, and into a construction decision.

Analysis of the results and findings from each major area of investigation completed as part of this feasibility study suggests numerous recommendations for further investigations to mitigate risks and/or improve the base case designs. The following sections summarize the key recommendations arising from this feasibility study. Each recommendation is not contingent to a subsequent one. Table 26-1on the following page presents a summary of recommended tasks, budget and detailed in the subsections that follow.

Table 26-1: Proposed Budget Summary

Description	Cost ($)
Drilling	10,000,000
Metallurgical Testwork	665,000
Mine Geotechnical	500,000
Mine Studies	400,000
Hydrological	75,000
Water Treatment	300,000
Meteorological Update	N/A
Infrastructure Geotechnical and Construction Borrow Material	1,300,000
Environmental, Permitting and Social	4,600,000
Water Sampling and PAG/NAG Evaluation	5,000,000
Total	**22,840,000**

26.2 Drilling

A total of 67 surface drill holes for a total of 12,536.9 m have been drilled at Eskay Creek since the database supporting the Mineral Resource estimate was closed out.

Skeena planned to drill 60,000 m using skid-mounted and heli-portable drills during 2022 of which 42,000 m has been completed. This program is estimated with all-in drilling costs of $555/m, to be approximately $10.0 M. At program completion, the intent is to update the block model and resource estimate.

26.2.1 Sampling and QA/QC

The QA/QC measures implemented in the initial 2018–2019 drill programs should be retained for future drill campaigns.

Lithological, alteration, mineralization and structural data captured during future drilling 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.

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.

26.3 Mineral Processing & Metallurgical Testing

For the flowsheet selected in the feasibility study, additional testwork is proposed to further refine metallurgical performance estimates and equipment sizing. Specifically, additional testwork should include:

- Variability testing on samples with selected feed characteristics, to improve the recovery and concentrate grade models and confirm metallurgical performance
- Comminution testing on new samples to increase the database of results to optimize comminution power
- Investigate opportunities to apply coarser grinds to samples that have higher work indices and likely higher SiO2 contents
- Investigate cleaner circuit flowsheet modifications to optimize regrind energy application, improve concentrate quality and improve dewatering performance
- Optimisation of HW/Mudstone flowsheet conditions and confirm the impact of blending at 20 to 50% of plant feed;
- Generation of additional regrind mill feed samples for vendor testing (both IsaMill and HIGmill), also complete bench scale regrinding testing where applicable
- Generation of additional final concentrate samples for filter/dryer vendor testing
- Material handling test work is recommended on crushed material and concentrate for design of bins, chutes, conveyors, and stockpile drawdown.

The last two items will require bulk samples and pilot plant runs to generate sufficient mass for testing.

This will require approximately 1.5 t of half core samples. It is expected the next phase of testwork will cost approximately $565 k with pilot plant work to cost an additional $100 k.

26.4 Mining Methods

26.4.1 Mine Geotechnical

Further data collection and interpretation tasks are recommended to fill in data gaps to support future stages of design. These include:

- Improvements to geotechnical core logging methodology, including the addition of Joint Roughness Coefficient (JRC), degree of alteration/weathering, fracture spacing, number of discontinuity sets, identification of faults/shears, logging of both "worst-case" and "representative" discontinuities (if not feasible to log all discontinuities), logging of joint roughness number (Jr) and joint alteration number (Ja) for every discontinuity, and the use of geotechnical intervals instead of runs for the main delineation of logging units.

- The collection of supplementary structural data in areas of the open pit and diversion tunnel where existing data is sparse or where additional data it is required to validate design inputs. Surface mapping is recommended to obtain information on discontinuity persistence and waviness across the open pit area and at the diversion tunnel portal locations. Additional characterization of the location, orientation and geotechnical characteristics of major structures (i.e., fault and shear zones) is also recommended.

- Supplementary laboratory strength data, particularly in the Hanging Wall Mudstone, Contact Mudstone, Footwall Sediments and Bowser Sediments units where existing laboratory data is limited. Additionally, discontinuities were not systematically tagged by structure type during the 2020 and 2021 drilling programs, so it was not possible to develop relationships between discontinuity type and shear strength. This is recommended for future studies.

- Calibration of rock fall analyses at portal locations based on ongoing observations of rock fall activity along the Tom Mackay Creek valley.

- An in-situ stress study in the diversion tunnel area. An analysis of borehole breakouts from televiewer surveys may provide information on in-situ stresses.

A budget of approximately $500 K is recommended.

26.4.2 Mine Studies

The following areas should be addressed during more detailed studies. These studies are collectively estimated at $400 k.

26.4.2.1 Grade Control

The 2022 feasibility study 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.

26.4.2.2 Geology Model Improvement

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.

26.4.2.3 Dewatering Requirements

A proper understanding of pumping requirements and the hydrogeology is critical. Further work assessing this is recommended. Additional hydrogeological testing including packer testing, piezometer installations, pumping well construction and long-term aquifer testing is recommended. A numerical groundwater flow model should be calibrated and developed under transient conditions to inform subsequent geotechnical evaluations and depressurization assumptions.

26.4.2.4 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.

26.4.2.5 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.

26.5 Recovery Methods

The following activities are recommended to support the design of the processing plant beyond the feasibility study:

- Incorporate the abovementioned materials handling testwork (material flowability testwork results) into the crushing and stockpile circuit detailed design
- Incorporate the abovementioned metallurgical testwork to refine the comminution, flotation and concentrate handling equipment sizing.

26.6 Project Infrastructure

The following activities are recommended to support the detailed design of the project infrastructure beyond the feasibility study:

- Confirmatory geotechnical site investigations should be carried out at the preferred surface infrastructure site locations to characterise the foundation conditions associated with the proposed buildings, identify borrow material sources for construction activities, and provide information for support of the WRSF, plant site, ancillary facilities locations, and the TMSF designs. This program is estimated at $1.3 M.
- Further logistics planning and route surveys. This program is estimated at $100 k.
- The design of the 69 kV high-voltage powerline and substation should be further refined by BC Hydro and consultants.
- Additional wells will need to be installed and pumping tests carried out to establish sustainable yield and to support licensing. This program is estimated at $700 k.

- Additional 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.

26.7 Environmental Studies, Permitting, & Social or Community Impact

These recommendations will focus on project environmental, permitting, and social de-risking activities, which will include:

- continuation of a suite of monitoring and baseline environmental studies, some of which have been ongoing since 2020 for documentation of current conditions since 2020. Much of the baseline data collection since 2020 has established new or re-used historic baseline/permit compliance/monitoring sampling locations and future monitoring programs of a suite (e.g. climate, hydrology, surface and groundwater quality and quantity) will be useful for long-term monitoring and ongoing permit compliance as well as collaboration with Indigenous Nations.
- proceeding through permitting and EA/IA and Indigenous processes and relationship building
- documenting the required data to support applications for operating permits and completion of such applications
- consultation, engagement 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.

Additionally, to characterize the waste rock material and minimize PAG, a lab could be established on the site to support PAG/NAG evaluation. This same lab could also support water quality testing and modelling to further validate the removal of the water treatment plant. A budget of approximately $5 M is recommended for this lab.

26.7.1 Meteorological Update

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 was installed at the Project in 2020 to provide a correlation between the Eskay Creek and KSM project data sets and continued monitoring will inform future project design and modelling as well as tracking of potential changes in site specific data trends.

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