NorthWestcopper

Kwanika-Stardust Project

NI 43-101 Technical Report and Preliminary Economic Assessment

British Columbia, Canada

Effective Date: January 04, 2023

Report Date: February 17, 2023

Prepared for:

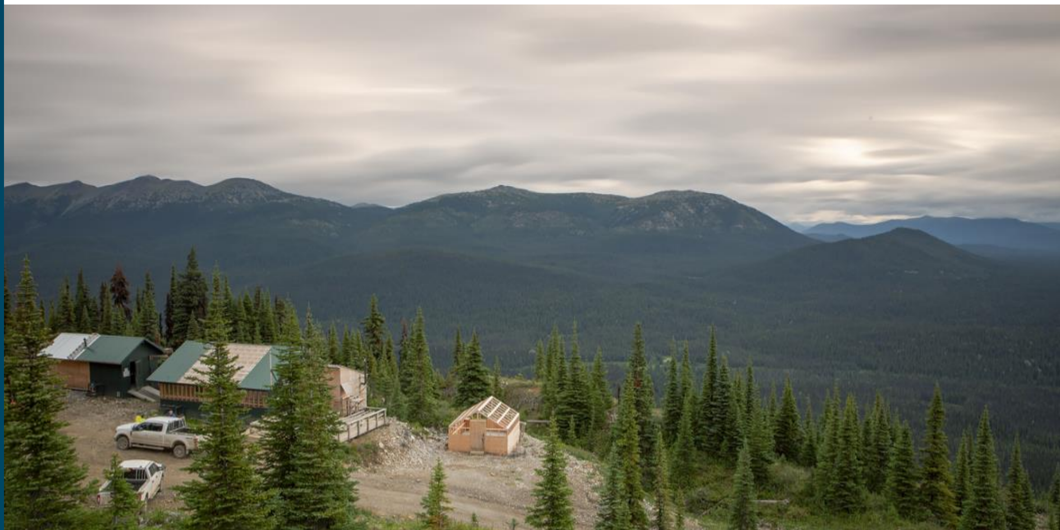
NorthWest Copper Corp.
1900-1055 West Hastings St.
Vancouver, BC, V6E 2E9, Canada

Prepared by:

Ausenco Engineering Canada Inc.
1050 West Pender, Suite 1200
Vancouver, BC, V6E 0C3, Canada

List of Qualified Persons:

Kevin Murray, P. Eng., Ausenco Engineering Canada Inc.
Jonathan Cooper, P. Eng., Ausenco Sustainability Inc.
Peter Mehrfert, P. Eng., Ausenco Engineering Canada Inc.
Scott C. Elfen, P. Eng., Ausenco Engineering Canada Inc.
Scott Weston, P. Geo., Ausenco Sustainability Inc.
Cale DuBois, P. Eng., Mining Plus Canada Consulting Ltd.
Jason Blais, P. Eng., Mining Plus Canada Consulting Ltd.
John Caldbick, P.Eng, Mining Plus Canada Consulting Ltd.
Brian S. Hartman, P. Geo., Ridge Geosciences LLC
Ronald G. Simpson, P. Geo., GeoSim Services Inc.



Ausenco

CERTIFICATE OF QUALIFIED PERSON

Kevin Murray, P. Eng.

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

1. I am a Professional Engineer, currently employed as a Manager Process Engineering with Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, B.C. Canada, V6E 3S7.
2. This certificate applies to the technical report titled, "Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment," (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023, with an effective date of January 4, 2023 (the "Effective Date").
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 the Association of Professional Engineers and Geoscientists of the Province of British Columbia, Registration number# 32350 and the Northwest Territories Association of Professional Engineers and Geoscientists' Registration# L4940.
4. I have practiced my profession continuously for 22 years. I have been directly involved in all levels of engineering studies from preliminary economic assessments (PEAs) to feasibility studies including being a Qualified Person for flotation projects such as Ero Copper Corp.'s Boa Esparança 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, located in Ontario, Porcupine Gold Mine located in Ontario, and Éléonore Gold Mine, located in Quebec, while working for Goldcorp Inc./Newmont Corporation.
5. I have read the definition of "Qualified Person" set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Kwanika-Stardust Project.
7. I am responsible for Sections 1.1, 1.11, 1.12.1-1.12.8, 1.14 - 1.16, 1.17.1, 1.17.6, 2, 3.1, 3.3, 3.4, 17, 18.1, 18.2, 18.3.1-18.3.6, 18.3.9, 19, 21.1, 21.2.1-21.2.3, 21.2.5-21.2.10, 21.2.11.1, 21.2.11.3, 21.2.11.4, 21.3.1, 21.3.2, 21.3.4, 21.3.5, 22, 24, 25.1, 25.7, 25.8, 25.10-25.13, 25.14.1.5, 25.14.1.9-25.14.1.13, 25.15, 26.1, 26.6, and 27 of the Technical Report.
8. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
9. I have had no previous involvement with the Kwanika-Stardust Project.
10. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
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: February 17, 2023

"*Signed and Sealed*"

Kevin Murray, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Jonathan Cooper M.Sc., P. Eng.

I, Jonathan Cooper, M.Sc., P. Eng., do hereby certify that:

1. I am a Professional Engineer, currently employed as a Senior Water Resources Engineer at Ausenco Sustainability Inc. ("Ausenco"), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7.

2. This certificate applies to the technical report titled, "Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment," (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023, with an effective date of January 4, 2023 (the "Effective Date").

3. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.

4. I am a Professional Engineer registered and in good standing with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number # 37864 and Professional Engineers Ontario licence number 100191626.

5. I have practiced my profession continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the Kwanika-Stardust Project are KSM for Seabridge Gold and Borden Advanced Exploration for Goldcorp, located in British Columbia and Ontario, respectively.

6. I have read the definition of "Qualified Person" set out in 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 the purpose of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

7. I have not visited the Kwanika-Stardust Project.

8. I am responsible for Sections 1.12.11, 1.17.8, 1.17.9, 16.3, 18.3.10, 25.14.1.7, 26.8, and 26.9 of the Technical Report.

9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.

10. I have had no previous involvement with Kwanika-Stardust Project.

11. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.

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: February 17, 2023.

"Signed and sealed"

Jonathan Cooper, M.Sc., P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Peter Mehrfert, P. Eng.

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

1. I am a Professional Engineer, currently employed as Principal Process Engineer, with Ausenco Engineering Canada Inc. (“Ausenco”), with an office at 1050 W Pender St, Suite 1200, Vancouver, B.C. V6E 3S7.
2. This certificate applies to the technical report titled, *“*Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment,*”* (the “Technical Report”) prepared for NorthWest Copper Corp. (the “Company”) dated February 17, 2023, with an effective date of January 4, 2023 (the “Effective Date”).
3. I graduated from 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 and in good standing with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number 24527.
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. Around 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 copper projects that I have worked on that have similar features to the Kwanika-Stardust Project are: Mt Milligan and Highland Valley Copper located in British Columbia, Canada, and the Josemaria Project located in Argentina.
6. I have read the definition of “Qualified Person” set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Kwanika-Stardust Project.
8. I am responsible for Sections 1.8, 1.17.5, 13, 25.4, 25.14.1.4, 25.14.2.2, and 26.5 of the Technical Report.
9. I am independent of NorthWest Copper Corp. as independence is described by Section 1.5 of the NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have been involved with the Kwanika-Stardust Project since September 2022, during which I have analyzed results from past metallurgical programs and technical reports.
11. I have read NI 43-101, Form 43-101F1 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: February 17, 2023

“*Signed and Sealed*”

Peter Mehrfert, P. Eng.
APEGBC # 100283

CERTIFICATE OF QUALIFIED PERSON
Scott C. Elfen, P.E.

I, Scott C. Elfen, P.E., do herby certify that:

1. I am a Professional Engineer, currently employed as the Global Lead Geotechnical and Civil Services of Ausenco Engineering Canada Inc. (“Ausenco”), with an office address of 855 Homer Street, Vancouver, BC V6B 2W2, Canada.
2. This certificate applies to the technical report titled, “Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment” (the “Technical Report”), prepared for NorthWest Copper Corp. (the “Company”), dated February 17, 2023, with an effective date of January 4, 2023 (the “Effective Date”).
3. I graduated from the University of California, Davis, CA, in 1991 with a Bachelor of Science degree in Civil Engineering (Geotechnical).
4. I am a Professional Engineer, Registered Civil Engineer in the State of California, licence number C56527 by exam since 1996 and I am also a member in good standing of the American Society of Civil Engineers , and the Society for Mining, Metallurgy & Exploration.
5. I have practiced my profession continuously for 28 years with experience in the development, design, construction and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities ranging from slurry to dry stack facilities, focusing on precious and base metals, both domestic and international. In addition, I have developed geotechnical and civil design parameters for pit slope design, plant foundation design, and other supporting infrastructure. Examples of projects I have worked on include the Eskay Creek Project PEA PFS and FS Tailings Storage Facility (TSF) by Skeena Resources Limited, the Marban Project PEA and PFS TSF by O3 Mining, the Springpole Project PEA and PFS TSF by First Mining Gold, the Puna Silver FS and Detailing Engineering In-Pit Tailings Disposal by SSR Mining, and the Cangrejos Project PEA and PFS Dry Stack Tailings Facility (DSTF) by Lumina Gold Corp.
6. I have read the definition of “Qualified Person” set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I have visited the Kwanika-Stardust Project on September 20 and 21, 2022.
8. I am responsible for Sections 1.12.10, 1.17.7, 1.17.10, 18.3.8, 21.2.12, 25.14.1.6, 26.7, and 26.10 of the Technical Report.
9. I am independent of Northwest Copper Corp. as independence is defined in Section 1.5 of NI 43-101.
10. I have had no previous involvement with Kwanika-Stardust Project.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: February 17, 2023.

“Signed and sealed”

Scott C. Elfen, P.E.

CERTIFICATE OF QUALIFIED PERSON
Scott Weston, P. Geo.

I, Scott Weston, P. Geo., do hereby certify that:

1. I am a Professional Geoscientist, currently employed as Vice President, Business Development with Ausenco Sustainability Inc. (“Ausenco”), with an office address of 4515 Central Boulevard, Burnaby, B.C., Canada.

2. This certificate applies to the technical report titled, “Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment,” (the “Technical Report”) prepared for NorthWest Copper Corp. (the “Company”), dated February 17, 2023, with an effective date of January 4, 2023 (the “Effective Date”).

3. I graduated from University of British Columbia, Vancouver, B.C., Canada, 1995 with a Bachelor of Science, Physical Geography, and Royal Roads University, Victoria, B.C., Canada, 2003 with a Master of Science, Environment and Management.

4. I am a Professional Geoscientist of Engineers and Geoscientists British Columbia; registration number 124888.

5. I have worked as a geoscientist continuously for 27 years, leading or working on teams advancing multidisciplinary environmental projects related to natural resource development. Example of projects I have been involved with include: Wasamac Project FS, Eskay Creek Mine PFS, Las Chispas Mine FS, and Casino Project FS.

6. I have read the definition of “Qualified Person” set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) for those sections of the Technical Report that I am responsible for preparing.

7. I have not visited the Kwanika-Stardust Project.

8. I am responsible for Sections 1.13, 1.17.11, 3.2, 20, 25.9, 25.14.1.8, and 26.11 of the Technical Report.

9. I am independent of the Company as independence is described by Section 1.5 of NI 43–101 and in Section 1.5 of the Companion Policy to NI 43-101.

10. I have not previously been involved with the Kwanika-Stardust Project.

11. I have read the NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.

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 the Technical Report not misleading.

Dated: February 17, 2023

“*Signed and sealed*”

Scott Weston, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

Cale DuBois, M.A.Sc., P. Eng.

I, Cale DuBois, M.A.Sc, P. Eng., do hereby certify that:

1. I am a Professional Engineer, currently employed as a Principal Engineer with Mining Plus Canada Consulting Ltd., with an office address of 77 King Street West, Suite 3740, Toronto Ontario, M5K 2A1.
2. This certificate applies to the technical report titled, "Kwanika-Stardust Project, NI 43-101 Technical Report and Preliminary Economic Assessment," (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023 with an effective date of January 4, 2023 (the "Effective Date").
3. I graduated from The University of British Columbia with a Bachelor of Applied Science in 2001 and a Master of Applied Science in 2009.
4. I am a Professional Engineer, registered and in good standing with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number 34168 and Professional Engineers of Ontario, licence number 100500088.
5. I have practiced my profession continuously for 19 years. I have been directly involved in mine design, geotechnical design, mining operations, mine construction projects and mining studies. My relevant experience for the purpose of the Technical Report includes the following:
   - Senior Geotechnical Engineer at the Cameco - Cigar Lake Project in Northern Saskatchewan;
   - Lead Geotechnical Engineering consultant for Glencore South American Mine Optimization Study Phase One located in Bolivia, South America. Caballo Blanco Operational Unit (Colquechquita, Tres Amigos, Reserva) located close to Potosi Bolivar Mine, close to Oruro, Bolivia;
   - Practicing Geotechnical Engineer for Evolution Red Lake operation located in Ontario, Canada;
   - Lead Geotechnical Engineering consultant for Thyssen – Cadomin Quarry Feasibility Study for the Cadomin Quarry located in Alberta, Canada;
   - Lead Geotechnical Engineer consultant for Tower Gold Consolidated Preliminary Economic Assessment Study for the Moneta Gold located in Ontario, Canada;
6. I have read the definition of "Qualified Person" set out in 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 the purposes of NI43-101 and in connection with those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Kwanika-Stardust Project location.
8. I am responsible for Sections 1.17.3, 16.2, 25.6.1, and 26.3 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no previous involvement with Kwanika-Stardust Project.
11. I have read NI 43-101, Form43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form43-101F1.
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: February 17, 2023

"Signed and sealed"

Cale DuBois, M.A.Sc, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Jason Blais, P. Eng.

I, Jason Blais, P. Eng., do hereby certify that:

1. I am a Professional Engineer, currently employed as a Manager, Plan Division with Mining Plus Canada Consulting Ltd., with an office address of Suite 504, 999 Canada Place, Vancouver, B.C. V6C 3E1.
2. This certificate applies to the technical report titled "Kwanika-Stardust Project, NI 43-101 Technical Report on Preliminary Economic Assessment" (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023 with an effective date of January 4, 2023 (the "Effective Date").
3. I graduated from the University of British Columbia in Vancouver B.C., in 2011 with a Bachelor of Applied Science Degree in the Mining Engineering Cooperative Program.
4. I am a Professional Engineer, registered and in good standing with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number 50105.
5. I have practiced my profession continuously for 11 years. I have been directly involved in mine design, mining operations, mine construction projects and mining studies. Previous projects that I have worked on that have similar features to the Kwanika-Stardust Project are the Skouries Feasibility Study for Eldorado Gold Corporation, the Pumpkin Hollow Feasibility study and Mine Construction for Nevada Copper in Greece and Nevada.
6. I have read the definition of "Qualified Person" set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I visited the Kwanika-Stardust Project on September 20, 2022.
8. I am responsible for Sections 1.10.1, 1.10.2, 12.1.1, 15, 16.4, 25.6.2, 25.6.3, 25.14.1.3.1, 25.14.1.3.2, 25.14.2.3.1, and 25.14.2.3.2 of the Technical Report.
9. I am independent of the Company, as independence is defined in Section 1.5 of the NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have no previous involvement with the Kwanika-Stardust Project.
11. I have read NI 43-101, Form43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form43-101F1.
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: February 17, 2023

"Signed and sealed"

Jason Blais, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
John Caldbick, P. Eng.

I, John Caldbick, P. Eng., certify that:

1. I am a Professional Engineer, currently employed as a Senior Principal Consultant with Mining Plus Canada Consulting Ltd., with an office address of Suite 3740, 77 King St West, Toronto, ON M5K 2A1, Canada.
2. This certificate applies to the technical report titled "Kwanika-Stardust Project, NI 43-101 Technical Report on Preliminary Economic Assessment" (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023 with an effective date of January 4, 2023 (the "Effective Date").
3. I graduated from the following institutions:
   - University of Toronto in Toronto, Ontario, in 1985 with a Bachelor of Arts Degree in Political Science.
   - Haileybury School of Mines in Haileybury, Ontario, in 1995 with a Mining Engineering Technician Diploma.
   - Queen's University in Kingston, Ontario, in 1999 with a Bachelor of Science Degree in Mining Engineering.
4. I am a Professional Engineer registered and in good standing with Professional Engineers Ontario; licence number 100042950.
5. I have practiced my profession continuously for 23 years. I have been directly involved in mine design, mining operations, mine construction projects and mining 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Kwanika-Stardust Project.
8. I am responsible for sections 1.10.3, 1.12.9, 1.17.4, 16.1, 16.5, 16.6, 18.3.7, 21.2.4, 21.2.11.2, 21.3.3, 25.6.4, 25.14.1.3.3, 25.14.2.3.3, and 26.4 of the Technical Report.
9. I am independent of the Company, as independence is defined in Section 1.5 of NI 43-101 and in Section1.5of the Companion Policy to NI 43-101.
10. I have had prior involvement with the Kwanika-Stardust Project. From March 26, 2021 to August 8, 2021 I served as Project Manager and author of the "MP 7055 – Kwanika Stardust Strategic Review," report dated July 28, 2021.
11. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
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: February 17, 2023.

"Signed and sealed"

John Caldbick, P. Eng.
PEO #100042950

CERTIFICATE OF QUALIFIED PERSON
Brian S. Hartman, M.S., P. Geo.

I, Brian S. Hartman, M.S., P. Geo., do hereby certify that:

1. I am a Professional Geologist, currently the owner and Principal Consultant with Ridge Geoscience LLC., with an office at 3152 Scanlon Farms Road, Coralville, IA 52241 and I work as a subcontractor to Mining Plus Consulting Ltd.
2. This certificate applies to the technical report titled "Kwanika-Stardust Project, NI 43-101 Technical Report on Preliminary Economic Assessment" (the "Technical Report"), prepared for NorthWest Copper Corp. (the "Company") dated February 17, 2023 with an effective date of January 4, 2023 (the "Effective Date").
3. I graduated from the University of Iowa in Iowa City, IA with a Bachelor of Science in Geoscience in 2001 and a Master of Science in Geoscience in 2004.
4. I am a Professional Geoscientist registered and in good standing with Professional Geoscientists Ontario, membership number 2413, and the Society for Mining, Metallurgy & Exploration, Registered Member #4175655.
5. I have practiced my profession continuously for 17 years. I have been directly involved in mineral exploration, mine geology and resource estimation.
6. I have read the definition of "Qualified Person" set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the Kwanika-Stardust Project.
8. I am responsible for portions of Sections 1.2.1, 1.3.1, 1.4.1, 1.5.1, 1.6.1, 1.7.1, 1.9.1, 1.9.2, 1.17.2.1, 4.1, 5, 6.1, 7.1, 7.2, 8.1, 9.1, 10.1, 11.1, 12.1.2, 12.1.3, 14.1, 14.2, 23, 25.2, 25.3, 25.5.1, 25.14.1.1, 25.14.1.2, and 26.2 of the Technical Report, as they pertain to the Kwanika Central and Kwanika South Deposits.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no previous involvement with the Kwanika-Stardust Project.
11. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
12. As of the Effective Date, 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: February 17, 2023

"Signed and sealed"

Brian S. Hartman, M.S., P. Geo.

CERTIFICATE OF QUALIFIED PERSON

Ronald G. Simpson, P. Geo.

I, Ronald G. Simpson, P. Geo., do hereby certify that:

1. I am a Professional Geoscientist, currently employed as a Professional Geoscientist with GeoSim Services Inc., with an office at 807 Geddes Road, Roberts Creek, B.C. V0N 2W6.
2. This certificate applies to the technical report titled "Kwanika-Stardust Project, NI 43-101 Technical Report on Preliminary Economic Assessment" (the "Technical Report") prepared for NorthWest Copper Corp. (the "Company"), dated February 17, 2023 with an effective date of January 4, 2023 (the "Effective Date").
3. I graduated with a Bachelor of Science in Geology from The University of British Columbia in May 1975.
4. I am a Professional Geoscientist, registered and in good standing with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number 19513.
5. I have practiced my profession [continuously] for 48 years. I have been directly involved in mineral exploration, mine geology and resource estimation with practical experience from feasibility studies I have past experience with and authored Technical Reports on CRD and other skarn related deposits located in Canada and Mexico. I have also authored three previous Technical Reports involving resource estimation on the Canyon Creek skarn deposit.

   Specific project details:

   - Nickel Plate Mine - B.C. (gold skarn) 1984-1998 Supervised exploration and development programs and co-authored publication
   - Springer MINE - NV (tungsten skarn) 2008 - technical report
   - Sierra Mojada -Mexico (CRD) 2011 - resource estimation
   - Mina La Negra - Mexico (CRD) 2008 & 2010 - resource estimations
   - Canyon Creek (This project) 2010, 2018 & 2021 - resource estimations
6. I have read the definition of "Qualified Person" set out in 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 the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.
7. I visited the Kwanika-Stardust Project on June 14, 2010, October 19, 2017 and September 23, 2020.
8. I am responsible for Sections 1.2.2, 1.3.2, 1.4.2, 1.5.2, 1.6.2, 1.7.2, 1.9.3, 1.17.2.2, 4.2, 6.2, 7.3, 8.2, 9.2, 10.2, 11.2, 12.2, 14.3, 21.2.11, 25.5.2, 25.14.2.1, and 26.2.2 of the Technical Report as they pertain to Stardust.
9. I am independent of the Company as independence is described in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had prior involvement with the Kwanika-Stardust Project. I was an author of the following previous technical reports on the Kwanika-Stardust Project:
    - "*Technical Report, Canyon Creek Copper-Gold Deposit, Lustdust Property, Omineca Mining Division, British Columbia, Canada*" with effective data of June 23, 2010;
    - "*Stardust Project NI43-101 Technical Report, Omineca Mining Division, British Columbia, Canada*" with effective data of January 8, 2018; and
    - "*Stardust Project – Updated Mineral Resource Estimate, Omineca Mining Division, British Columbia, Canada*" with effective data of July 2, 2021.
11. I have read NI 43-101, Form 43-101F1 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
12. As of the Effective Date, 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: February 17, 2023

"Signed and Sealed"

Ronald G. Simpson, P. Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for NorthWest Copper Corp. (NorthWest Copper) by Ausenco Engineering Canada Inc., Ausenco Sustainability Inc., Mining Plus Canada Consulting Ltd., Ridge Geosciences LLC., and GeoSim Services Inc., 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 NorthWest Copper subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Introduction

NorthWest Copper Corp. (NorthWest Copper) commissioned Ausenco Engineering Canada Inc. and Ausenco Sustainability Inc. (Ausenco) to compile a Preliminary Economic Assessment (PEA) of the Kwanika-Stardust project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 – Standards and Disclosure for Mineral Projects (NI 43-101) and the requirements of Form 43-101 F1.

The Kwanika-Stardust project involves the development of three copper-gold deposits known as "Kwanika Central," "Kwanika South," and "Stardust," all located around 195 kilometres (km) by road from Fort St. James, British Columbia, Canada.

The responsibilities of the engineering companies whom NorthWest Copper contracted to prepare this report are as follows:

- Ausenco managed and coordinated the work related to the report and developed PEA-level design, including cost estimates for the process plant, general site infrastructure, tailings storage facility, waste rock storage facility, environment and permitting, and economic analysis.
- Mining Plus Canada Consulting Ltd. (Mining Plus) designed the open pit and underground mining, mine production schedule, and mine capital and operating costs.
- Ridge Geosciences LLC. (Ridge Geosciences) developed the mineral resource estimate for the Kwanika project and completed the work related to property description, accessibility, local resources, geological setting, deposit type, drilling, exploration works, sample preparation and analysis, data verification and completed a review of the environmental studies.
- GeoSim Services Inc. (GeoSim) developed the mineral resource estimate for the Stardust project and completed the work related to property description, geological setting, deposit type, exploration work, drilling, sample preparation and analysis, and data verification.

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

1.2.1 Kwanika

NorthWest Copper, indirectly through its wholly owned subsidiary Kwanika Copper Corp. owns a 100% interest in the Kwanika property (including the Central and South zones), which is situated amongst a group of 59 unpatented mineral claims covering an area of 24,152.04 ha. There are no title encumbrances, surface rights issues or legal access obligations that must be met in order for NorthWest Copper to retain the property. The property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

1.2.2 Stardust

NorthWest Copper, indirectly through its wholly owned subsidiary Tsayta Resources Corp. (Tsayta), owns a 100% interest in the Stardust property. The Stardust property encompasses 26 mineral claims covering 12,932.39 ha. There are no title encumbrances, surface rights issues or legal access obligations that must be met in order for NorthWest Copper to retain the property. The Stardust property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

1.3 Geology and Mineralization

1.3.1 Kwanika

The Kwanika porphyry deposits are located at the western margin of the Quesnel terrane (Quesnellia). Quesnellia is a Late Paleozoic to Early Jurassic island arc that hosts numerous alkalic and calc-alkalic porphyry Cu ± Au ± Mo ± Ag deposits, and which extends north from the British Columbia-Washington State border for more than 1,000 km.

The Kwanika property consists of two mineralized areas: the Central Zone and the South Zone. Mineralization in the Central and South Zones at Kwanika occurs in the Quesnel Terrane, immediately east of the Pinchi fault which places Quesnellia against the Cache Creek Terrane and is associated with intrusive phases of the Hogem batholith. The mineralization is mostly covered by glacial sediments that average 25 m to 35 m in thickness and, as such, bedrock geology is interpreted from drill core and the few outcrops along Kwanika Creek in the South Zone.

The Central Zone is 1,400 m long by 400 m wide and extends more than 700 m below surface where it is open to depth on many drill sections. It is downfaulted in its middle on the west side of the Central fault and then cut off by the Pinchi fault further west. Mineralization is mainly hosted by a shallow to steeply-dipping plug and dyke complex of quartz monzonite porphyry. Hydrothermal alteration in the Central Zone comprises an inner potassic core surrounded by an outer potassic shell that yields to a peripheral propylitic zone, all of which are variably overprinted by patchy sericite alteration.

The South Zone is 2,200 m long by about 330 m wide, and locally extends more than 600 m below the surface. The highest copper grades occur in a steeply-dipping, 800 m long tabular body in the northwest part of the Zone, with an upper part extending to the east. The South Zone is ovoid in plan and is confined to a northerly trending corridor bounded by the West and East faults. Mineralization in the South Zone is mostly hosted by an equigranular quartz monzonite intrusion. Fine to medium-sized grains of pyrite, chalcopyrite, and minor molybdenite occur along micro-fractures and as disseminations within zones of fine-grained quartz that replace potassically-altered quartz monzonite.

1.3.2 Stardust

The Stardust property is located within the Cache Creek Terrane of the Intermontane Belt west of the Pinchi Fault. Once a major thrust fault, the Pinchi was later reactivated as a major right-lateral strike-slip fault which can now be traced roughly 600 km through north-central British Columbia. At Stardust, the Pinchi marks the contact between the Pennsylvanian-Permian Cache Creek Terrane to its southwest and Quesnellia to the northeast, which includes the Jurassic Hogem Batholith and Triassic Takla Group rocks.

Most of the property is underlain by very strongly deformed Pennsylvanian to Permian Cache Creek units. Much of the mapped regions of the property contains an assortment of intrusions that cut carbonate rocks interbedded with graphitic, siliceous, and calcareous phyllites, cherts, cherty argillites, and mafic flows. The most prominent intrusions form the

Eocene Glover stock but intrusions are found throughout the property, except in the far north of the claims, where they may lie buried beneath thick overburden.

Several styles of mineralization that are zonally related to each other and, apparently, to the Glover Stock, are present on the property. From most proximal to most distal from the Glover Stock, they are:

- Molybdenum-copper-gold porphyry consisting of quartz-K-spar, pyrite, molybdenite and/or chalcopyrite veinlets associated with potassic, sericitic, and propylitic alteration in intrusive rocks (Glover Stock).
- Multi-stage garnet-diopside skarn cut by Cu-Au-Ag-Zn bearing structures with surrounding dispersed Cu-Au mineralization (Canyon Creek Skarn).
- Structurally and stratigraphically controlled massive sulphide Zn, Au, Pb, Ag, Cu replacement bodies [CRD] (4b, 3, and 2 Zones) and their oxidized equivalents.
- Sulphosalt-rich veins (Zone 1) which follow faults and are associated with fine-grained, linear, felsic dykes containing high values of Au, Ag, Pb, Zn, Sb and Mn.
- Mercury mineralization in limestone proximal to the Pinchi Fault.
- Sediment-hosted gold mineralization in limestone.

1.4 History

1.4.1 Kwanika

Various claims in the area have been held by different operators since 1965. Serengeti Resources Inc. (Serengeti) staked the property starting in 2004 and operated throughout the years. In 2016 Serengeti formed a Joint Venture agreement with POSCO DAEWOO Corporation (POSCO) forming the private company Kwanika Copper Corporation (Kwanika Copper) which then owned the Kwanika tenure. POSCO had the option to earn in up to 35% of Kwanika Copper. In 2017, POSCO had met the requirements and had earned 35% of Kwanika Copper with Serengeti controlling the remaining 65%. From 2019 onward Serengeti started to earn back shares of Kwanika Copper. On March 5th of 2021, Serengeti and Sun Metals Corporation (Sun Metals) merged to form NorthWest Copper. This was followed by an announcement on December 29th, 2021 of NorthWest Copper's intent to purchase the remaining 31% share of Kwanika Copper from POSCO. As part of the Tranche 1 closing, the shareholder joint venture agreement was terminated and any interest or rights of POSCO with respect to the Kwanika project under the shareholder joint venture agreement, including offtake rights, were terminated. On September 7, 2022, NorthWest completed the closing of the Final Tranche of its consolidation of the Kwanika project, and now owns 100% of Kwanika Copper.

1.4.2 Stardust

The property has been explored since 1944 when the Takla silver vein (No. 1 Zone) was discovered. Alpha Gold Corporation (Alpha Gold) carried out exploration on the property between 1991 and 2012. In June of 2016, Lorraine Copper Corp. (Lorraine Copper) entered into an agreement to purchase a 100% interest in the property from Alpha Gold.

In September 2017, 1124245 B.C. Ltd. (subsequently renamed Sun Metals Corp. (Sun Metals)) was granted an option to acquire a 100% interest in the property subject to certain royalties and terms. Sun Metals fulfilled the 2017 expenditure requirement by completing an exploration program by year end.

In April 2019, Sun Metals acquired all outstanding shares of Lorraine Copper through its subsidiary Tsayta, following which in order to own a 100% interest in the property.

In March 2021, Sun Metals and Serengeti announced the completion of a merger and a name change to NorthWest Copper Corp.

1.5 Exploration

1.5.1 Kwanika

In 2005, Serengeti conducted a 530 line-km airborne magnetic/radiometric survey and collected rock samples on the Kwanika property. The airborne survey identified a small magnetic anomaly on the east side of the known South Zone porphyry copper-gold deposit, with similar anomalies trending to the north-northwest of the deposit, as well as to the south.

In 2006, the discovery holes into the Central Zone starting with K-06-09 were drilled by Serengeti Resources; historical drilling was concentrated at the known South Zone, so these were the first holes into the Central Zone. Over the season Serengeti drilled 10 diamond drillholes for 1,874 m.

Following the success of the drilling in the Central Zone in 2006, Serengeti followed up with 47 diamond drillholes in 2007, totalling 22,415.4 m. Concurrently, a regional airborne magnetic and electromagnetic (EM) survey, totalling 320 line-km over the Kwanika property was carried out by Serengeti. The survey identified multiple magnetic anomalies with varying resistivity throughout the property. The anomalies were coincident and demonstrated a north-northwest trend that is seen in the South and Central Zone deposit areas.

This was followed by another 49 diamond drillholes for 26,553 m in 2008. From 2009 to 2012, Serengeti drilled 17 (6,249 m), 28 (7,619 m), 5 (1,724 m), and 4 (2,446 m) diamond drillholes, respectively, with 3 line-km of induced polarization (IP) in 2012.

Drilling on the property resumed in 2016 with 5 diamond drillholes totalling 2,446 m and an additional 2.4 line-km of surface IP work.

During the 2018 field season, Serengeti drilled 21 drillholes for 7,411 m to support engineering and economic studies. This was followed by no work in 2019 and an additional 4,355 m in 9 drillholes during 2020 as well as 15 line-km of ground IP survey.

Following the creation of NorthWest Copper by the merger of Sun Metals and Serengeti in early 2021, NorthWest Copper drilled 20 holes for 8,696 m. Additionally, NorthWest Copper collected 385 soil samples, 238 silt samples, and conducted 12 line-km of ground IP.

1.5.2 Stardust

The earliest publicly available reports on exploration on the property date from 1944 with the discovery of Zone 1 (Zn-Pb-As-Sb veins). Later exploration programs resulted in the discovery of several targets that were drilled sporadically between 1966 and 1981.

Major drill programs began in 1991 when Alpha Gold was the operator. Most of the exploration carried out on the property since 1999 has focused on the Canyon Creek Skarn Zone and peripheral areas. Sun Metals conducted three drill programs on the property since 2018 to further delineate and explore for extensions of the Canyon Creek Skarn Zone. These programs resulted in the discovery of the 421 Zone in 2018. Follow-up drilling from this discovery established that the 421 Zone was part of the Canyon Creek Skarn Zone.

1.6 Drilling and Sampling

1.6.1 Kwanika

From July 2006 to September 2021, Serengeti and later Northwest carried out 95,255 m of diamond drilling in 226 holes on the Kwanika property. The results of this drilling have achieved three main goals:

- Measured, Indicated and Inferred Mineral Resources have been delineated on the Central Zone deposit, which was initially discovered by Serengeti in late 2006.
- An Inferred Mineral Resource was delineated on the South Zone deposit.
- Several geophysical anomalies on the Kwanika property were tested to explore for possible extensions of the Central Zone deposit.

Previously, the South Zone area at Kwanika was drill tested during the period 1965 to 1991 by 30 diamond and percussion drillholes. The historical data are not included in this data compilation and Resource Estimate, but they were confirmed by Serengeti and Northwest's drilling activities from 2006 to 2021.

From 2018 to the end of 2021, Sun Metals completed 4 diamond drilling programs on the Stardust property, primarily in the Canyon Creek Skarn zone; these programs totalled 34,541.9 metres over 71 holes. Previously, at least 16 holes were drilled from 1965 to 1980 by Takla Silver Mines, Zapata Granby, and Noranda; additionally, 316 holes were drilled by Alpha Gold, Teck Cominco, and Lorraine Copper between 1991 and 2017.

1.6.2 Stardust

Between 2018 and the end of 2021, Sun Metals and Northwest carried out four drilling programs, primarily on the Canyon Creek Skarn zone. During this time, total of 34,522 m was completed in 70 diamond drillholes.

1.7 Data Verification

1.7.1 Kwanika

A site visit was completed to the Kwanika project area on September 20, 2022, by Mr. Jason Blais, P. Eng. of Mining Plus. All relevant data and procedures for measuring, capturing, recording, and storing were reviewed, and drill log and assay certificates were compared. Presentations of drill core and geological interpretation were made by senior site geologists.

1.7.2 Stardust

The author, Simpson has visited the Stardust project site on three occasions with the most recent visit being conducted on September 23, 2020. Previous visits were carried out on June 14, 2010, and September 17, 2017.

During the sites visits, the author visually identified copper-bearing sulphide mineralization in drill core and outcrop. A number of drill sites were checked by GPS and found to be accurate.

1.8 Metallurgical Testwork

Three metallurgical testwork programs have been performed on material from the Kwanika deposit, and the Stardust deposit has been the subject of its own metallurgical scoping study as described in Section 13.2. The full body of testwork is summarized in Table 1-1.

Table 1-1: Metallurgical Testwork Summary Table

Year	Test Programs	Laboratory
2008-2009	The Recovery of Copper and Gold from Kwanika Deposit	Vancouver Metallurgy, SGS Minerals Services
2009	Follow-up Testwork Summary on Kwanika Deposit	Vancouver Metallurgy, SGS Minerals Services
2018	Comminution Tests on Kwanika Deposit	ALS Metallurgy Kamloops
2018-2019	Prefeasibility Metallurgical Testing to Recover Copper and Gold on Kwanika Deposit	Metallurgical Division, Bureau Veritas Commodities Canada Ltd.
2020-2021	Metallurgical Scoping Study of the Stardust Project	Base Metallurgical Laboratories Ltd.
2022	Metallurgical Assessment of Samples from the Kwanika/Stardust Project	Base Metallurgical Laboratories Ltd.

The earliest SGS metallurgical testing utilized rougher and cleaner batch flotation, locked cycle flotation, and gravity testing. Gravity recovery results did not warrant the inclusion of gravity concentrators in the flowsheet. Rougher flotation was performed on P_{80} grind sizes of 133 µm, 87 µm, and 75 µm. The optimal conditions were considered to be a P_{80} of 75 µm, flotation time of 14 minutes, and a natural pH of 7.9. Cleaner flotation tests targeted a copper concentrate with a grade of 25-30% after three stages of cleaning. Rougher flotation concentrate was reground to 20 µm, 26 µm, and 32 µm to find the optimum regrind size. At 25 µm, the target grade was achieved with a copper recovery of 82-85%. Locked cycle tests attained a final concentrate grade of 27.7% copper with 88.5% copper recovery. At this concentrate grade, 65.2% of gold was recovered to the final concentrate.

The SGS follow-up testwork program was conducted to assess the potential of recovering gold from the rougher flotation tails. This was achieved by investigating the impact of regrind and scavenger flotation of rougher tails, cyanidation of rougher tails, and determination of gold distributions in different size fractions of rougher tails. Ultimately, this program determined that fine grinding can liberate more gold and improve gold recovery, but achieving this additional recovery is not necessarily economical.

The objectives of the Bureau Veritas test program were to establish optimized copper and gold recoveries for flotation design and determine grindability and head chemical and mineralogical characteristics of the material from the Kwanika deposit. The average of the BWi testwork results was 17.7 kWh/t. Mineralogical analysis showed low sulphide mineralization with only 3.5-4.7% sulphide minerals by weight. The copper in the deposit is fine-grained with most of the copper contained in chalcopyrite, and the remaining copper contained in chalcocite, covellite, and bornite. The non-sulphide mineralogy also supports the SGS findings of a relatively micaceous deposit.

The Bureau Veritas locked cycle flotation tests were able to achieve a saleable copper concentrate grade and reasonable copper and gold recoveries. The average copper recovery attained in these tests was 85.0% and the average gold recovery was 69.0%. Locked cycle tests with an additional cleaner scavenger stage achieved greater copper and gold recoveries than the previous locked cycle tests. Across these tests, the average copper recovery was 91.8% and the average gold recovery was 73.9%.

The first Base Metallurgical test program was performed to identify possible flowsheet options for processing material contained in the Stardust deposit. This test program was conducted separately of any results or flowsheet considerations from the earlier SGS and BV Minerals Kwanika test programs. Ultimately, this program proved that Stardust material can be processed by a similar flowsheet as the one established in earlier BV Minerals testwork.

The second Base Met test program was performed on composite samples made of material from both Kwanika and Stardust deposits. Similar metallurgical performance was measured, copper recoveries ranged from 81 to 92% to final concentrate grades of 26 to 35% copper. Gold recoveries to final concentrates ranged from 52 to 71%, with 14 to 16% of the gold reporting to gravity concentrates where applied.

From these testwork programs, a recovery estimate was generated for open pit and underground material. The primary grind size was increased from 75 µm, from the previous design, to 100 µm in the current PEA. These recovery equations are presented below in Table 1-2.

Table 1-2: Open Pit and Underground Recovery Equations

Metal	Open Pit	Underground
Copper	Rec = 6.04 * LN(Cu %) + 91.3	Rec = 4.75 * (Cu %) + 87.34
Gold	Rec = 9.24 * LN(Au g/t) + 76.5	Rec = 11.28 * LN(Au g/t) +77.5
Silver	Rec = 18.4 * LN(Ag g/t) + 52.37	Rec = 11.67 * LN(Ag g/t) + 57.5

1.9 Mineral Resource Estimation

1.9.1 Kwanika Central Mineral Resource Estimation

The Kwanika Central Mineral Resource is reported with an effective date of January 4, 2023 and using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus general and administrative (G&A) costs. The underground mineral resources are reported using an economic cut-off of US$ 16.41, which covers the additional underground mining and G&A costs of US$8.20/tonne. Additionally, the Mineral Resource is constrained by an open pit mining shell and underground block caving shape to satisfy reasonable prospects for eventual economic extraction. Table 1-3 shows the Kwanika Central Zone Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-3: Mineral Resource Statement - Kwanika Central Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit	8.21	Measured	30.7	0.31	0.31	1.05	210.8	310.5	1,041.7
		Indicated	35.9	0.22	0.19	0.80	174.9	222.0	923.9
		M&I	66.6	0.26	0.25	0.92	385.7	532.5	1,965.6
		Inferred	4.1	0.15	0.15	0.58	13.8	20.1	77.3
	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Underground	16.41	Measured	25.6	0.50	0.61	1.62	284.4	501.3	1,332.6
		Indicated	11.3	0.51	0.65	1.56	126.2	236.7	565.1
		M&I	36.8	0.51	0.62	1.60	410.6	738.0	1,897.8
		Inferred	-	-	-	-	-	-	-

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
3. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
4. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
5. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Underground Mineral Resources are reported at an economic cut-off of US$16.41 and constrained by a conceptual block cave shape. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne, and underground mining and G&A costs are assumed to be US$8.20/tonne.
6. Actual SG measurements were interpolated into the block model, with an average SG of 2.74.
7. The quantity and grade of reported Inferred Mineral Resources in this report are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
8. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

1.9.2 Kwanika South Zone Mineral Resource Estimation

The Kwanika South Mineral Resource is reported with an effective date of January 4, 2023 and using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus G&A costs. Additionally, the Mineral Resource is constrained by an open pit mining shell to satisfy reasonable prospects for eventual economic extraction.

Table 1-4 shows the Kwanika South Zone Mineral Resource. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-4: Mineral Resource Statement – Kwanika South Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit	8.21	Inferred	25.4	0.28	0.06	1.68	155.0	52.4	1,373.9

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral Resources are not mineral reserves and do not have demonstrated economic viability.
3. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
4. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
5. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 62% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne.
6. Actual SG measurements were interpolated into the block model, with an average SG of 2.68.
7. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
8. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

1.9.3 Stardust Mineral Resource Estimation

The updated Stardust mineral resource estimate for the Canyon Creek Skarn Zone has an effective date of January 4, 2023 and is presented in Table 1-5. It is based on a cut-off of US$88/t and 2-m minimum mining width. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-5: Mineral Resource Statement - Stardust Canyon Creek Skarn Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Underground	88.00	Indicated	1.6	1.49	1.63	30.1	52.2	83.1	1,536.4
		Inferred	4.1	1.00	1.38	22.8	90.0	181.1	3,004.3

Notes:

1. The Mineral Resources have been compiled by Mr. B Ronald G. Simpson of GeoSim Services Inc. Mr. Simpson has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
3. Mineral Resources are not mineral reserves and do not have demonstrated economic viability.
4. Reasonable prospects for economic extraction were determined by applying a minimum mining width of 2.0 m. and excluding isolated blocks and clusters of blocks that would likely not be mineable.
5. The base case cut-off of US$88/t was determined based on metal prices of $1,650/oz gold. $21.50/oz silver and $3.50/lb copper, underground mining cost of US$64/t, transportation cost of US$6/t, processing cost of US$8.25/t, and G&A cost of US$9.75/t. Recovery formulas were based on recent metallurgical test results. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag.

6. Block tonnes were estimated using a density of 3.4 g/cm^3 for mineralized material.
7. Six separate mineral domains models were used to constrain the estimate. Minimum width used for the wireframe models was 1.5 m.
8. For grade estimation, 2.0-metre composites were created within the zone boundaries using the best-fit method.
9. Capping values on composites were used to limit the impact of outliers. For Zone 102, gold was capped at 15 g/t, silver at 140 g/t and copper at 7.5%. For all other zones, gold was capped at 6 g/t, silver at 140 g/t and copper at 5%.
10. Grades were estimated using the inverse distance cubed method. Dynamic anisotropy was applied using trend surfaces from the vein models. A minimum of 3 and maximum of 12 composites were required for block grade estimation.
11. Blocks were classified based on drill spacing. Blocks falling within a drill spacing of 30 m within Zones 2, 3, and 6 were initially assigned to the Indicated category. All other estimated blocks within a maximum search distance of 100 m were assigned to the Inferred category. Blocks were reclassified to eliminate isolated Indicated resources within inferred resources.
12. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
13. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

1.10 Mining Methods

1.10.1 Kwanika Underground Mine Design

The Kwanika Central Block Cave mine produces 44 Mt mill feed with an average Net Smelter Return (NSR) value of 56.79 $/t based on 15% dilution at 38.41% of in-situ grade and 88.52% mining recovery. The operating cost assumed is 10.62 $/t inclusive of mining, transportation, and G&A. This mine commences production in year 3 of the project and is the predominant feed source for years 4 through 9 at a maximum throughput of 20,000 t/d. The Kwanika Central Block Cave mineral inventory is presented in Table 1-6.

Table 1-6: Kwanika Central Block Cave Mineral Inventory

Kwanika Central Block Cave	Total Mined (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	NSR ($/t)
Measured	31.0	0.50	1.35	0.42	55.58
Indicated	12.6	0.57	1.38	0.45	61.65
Unclassified Waste	0.4	0	0	0	0

1.10.2 Stardust Underground Mine Design

Stardust underground will produce 3.1 Mt mill feed with average NSR of 195.41 $/t. After 1 year of pre-production development the mine will be in steady state production for 6 years. The mine production schedule is presented in Table 1-7. Stardust underground mine is designed to be contractor-operated in this PEA.

Table 1-7: Stardust Underground Production Profile

	Total Mined (Mt)	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Mineral Tonnes (Mt)	3.11	0.00	0.49	0.60	0.61	0.61	0.59	0.20
Cu (%)	1.329	0.00	1.106	1.227	1.302	1.449	1.445	1.568
Au (g/t)	1.466	0.00	1.031	1.501	1.657	1.539	1.436	1.707
Ag (g/t)	27.793	0.00	20.358	28.990	33.043	27.372	26.397	31.842
NSR	195.41	0.00	148.58	189.93	207.53	208.77	201.83	230.40

1.10.3 Open Pit Mine Design

Both OP operations are designed as contractor-operated in this PEA.

1.10.3.1 Kwanika Central Pit

This mine produces 29.4 Mt mill feed with an average NSR value of 36.74 $/t and a strip ratio of 1.87. After 1-year pre-stripping at Year -1, this mine will be the only source of mill throughput in the production year of 1 to 3 and also the majority of mill feed in year 4.

The mine inventories in the designed Central Pit are presented in Table 1-8. This is calculated based on a 10x10x10 m regularized block model with 2% dilution and 5% losses, at an NSR Cut-off grade at $11.19/t.

Table 1-8: Central Pit Mineralized Material

	Total Mined (Mt)	Waste (Mt)	Overburden (Mt)	Mineralized Material (diluted & recovered) (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	NSR ($/t)
Central pit	84.62	25.18	29.99	29.45	0.29	1.16	0.32	36.74

Some amount of the resource encountered during production year 4 from Kwanika Central Pit is stockpiled. This resource will be reclaimed later in years 5 to12.

1.10.3.2 Kwanika South Pit

This mine will produce 19.1 Mt mill feed with an average NSR value of 23.4 $/t and a strip ratio of 1.66. This mine will supplement the Kwanika Block Cave in order to maintain mill feed capacity in years 9 to 12.

Kwanika South includes 3 separate pits which are mined in 4 phases from north to south. The mine contents in the designed South Pit are presented in Table 1-9. That is calculated based on 10x10x10 m regularized block model and applying 2% dilution and 5% losses to estimate the pit resource considering NSR Cut-off grade at $11.19/t.

Table 1-9: South Pit Mineralized Material

	Total Mined (Mt)	Waste (Mt)	Overburden (Mt)	Mineralized Material (diluted & recovered) (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	Mo (g/t)	NSR ($/t)
South pit	50.76	20.85	10.83	19.05	0.07	1.68	0.29	98.48	23.40

1.11 Recovery Methods

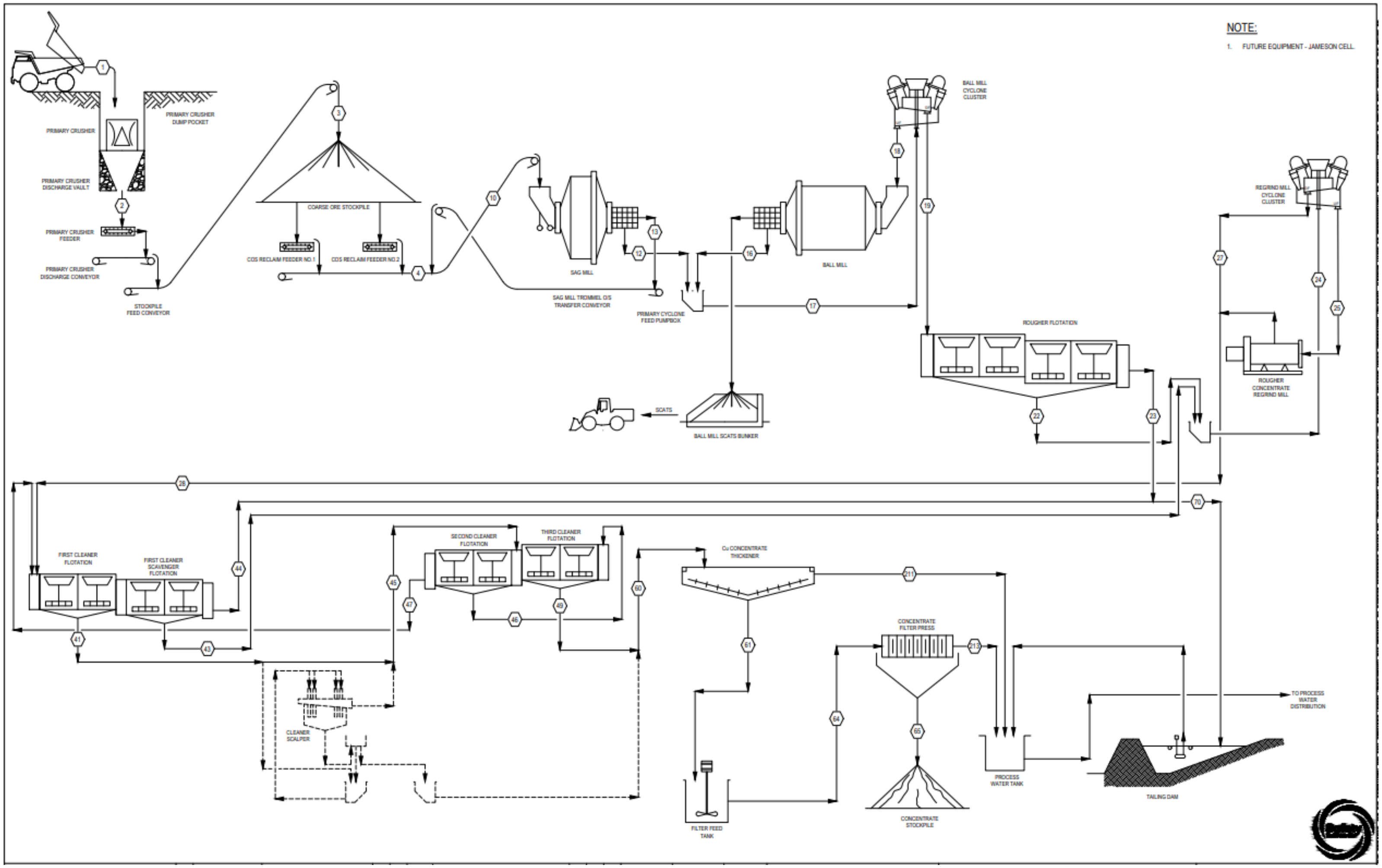
The process design is based on processing mineralized material from both the Kwanika and Stardust deposits through copper flotation to produce a saleable copper concentrate. The design is based previous testwork programs performed on the deposit, Ausenco's extensive database of reference projects, and in-house modelling programs. The plant is design for a throughput of 22,000 t/d at 92% availability. The crushing circuit is designed with an availability of 75%. The plant will operate with two 12-hour shifts per day, 365 days per year.

The process plant features the following:

- gyratory crushing of run-of-mine (ROM) material
- SAB grinding circuit followed by classification by cyclone
- rougher flotation, concentrate regrinding, and cleaning for copper flotation
- thickening, filtration, loadout, and shipping of copper concentrate
- tailings handling by pumping to tailings pond
- reagent handling and storage
- plant services, including air, water, and power.

The simplified process flow diagram for the project is shown below in Figure 1-1.

Figure 1-1: Simplified Process Flow Diagram



Source: Ausenco, 2022.

1.12 Project Infrastructure

1.12.1 Overview

Infrastructure at the Kwanika-Stardust project includes on-site infrastructure such as civil, structural, and earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, power supply, piping, camp and tailings storage facility. The site infrastructure will include:

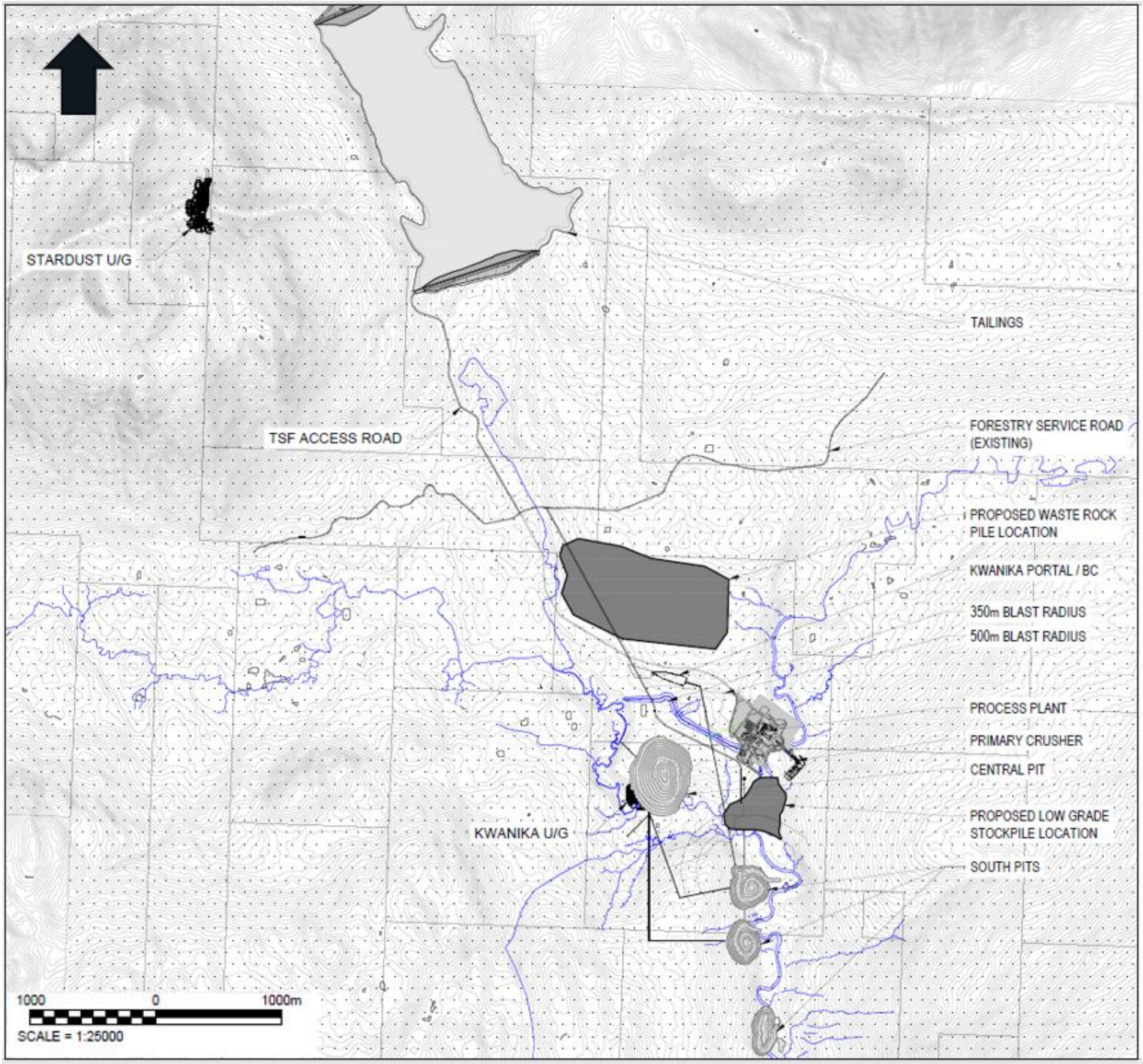
- mine facilities, including mining administration offices, a mine fleet truck shop and wash bays, a mine workshop, and a mine water treatment plant.
- common facilities, including an entrance/exit gatehouse, a security/medical office, overall site administration building, potable water and fire water distribution systems, compressed air, power generation and distribution facilities, diesel reception and combustion plants, communications area, and sanitation systems.
- process facilities, housed in the processing plant, including crushing, grinding and classification, flotation, product regrind, concentrate handling, thickening, dewatering, and filtration, reagent mixing and distribution, assay laboratory and process plant workshop and warehouse.
- other infrastructure includes the on-site man-camp, TSF and waste rock storage facility (WRSF).

The overall site layout was developed using the following criteria and factors:

- The facilities described above must be located on a site within the Kwanika-Stardust project boundary.
- The location of the process plant must be close to Kwanika open pit and underground mine which is the major source of feed, to reduce haul distance but outside of the 500 m blast radius.
- The location of the WRSF must be close to the open pits to reduce haul distance.
- The location of the primary crushing and ROM stockpile must be close to the Kwanika deposits to reduce haul distance.
- The TSF should be located at a site that takes advantage of sloped natural terrain to adequately drain entrained water and reduce earthworks, concrete, and structural development if possible.
- The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.

The Kwanika-Stardust project layout is shown in Figure 1-2.

Figure 1-2: Overall Site Plan



Source: Ausenco, 2022

1.12.2 Site Access

The Kwanika-Stardust project is located 140 kilometres northwest of Fort St. James. Surface access to the property is now provided via an existing forest service road (FSR) between Fort St. James and Tsayta Lake Road (see Figure 1-2). In addition, around a 30-kilometre-long Tsayta Lake Road will be improved to meet operational requirements and allow for the delivery of bulk freight by tractor-trailer units.

1.12.3 Water Supply

The Kwanika-Stardust project will source freshwater from wells and Kwanika Creek. Water will be pumped from the creek through a pipeline to the processing plants where storage tanks will be located. This water will be the source of potable water across the site.

1.12.4 Power Supply

Power will be provided from a connection to B.C. Hydro's electrical grid via a 230-kV transmission line from the Kemess Power Line to a substation at the site. The transmission line will be stepped down to the 25 kV at the substation for distribution to different power requirements across the project site.

1.12.5 Logistics

Copper-gold concentrate will be trucked from the site to Mackenzie, from which it will be transported by railcars to port storage facilities in North Vancouver. The concentrate will subsequently be transported by sea to clients.

1.12.6 On-site Roads

The Kwanika-Stardust project site has unpaved roads connecting the access road to the gatehouse. In addition to the existing roads on site, new roads will be constructed to link the gatehouse to the administration building, from the process plant to the TSF, and from the access road to the magazine.

1.12.7 Fuel Storage

Diesel storage facilities will consist of five bulk storage tanks with a total capacity of 500,000 L, which represents around two weeks of storage capacity for the site.

1.12.8 Buildings

The plant site consists of the necessary infrastructure to support the processing operations. All infrastructure buildings and structures will be built and constructed to all applicable codes and regulations. The project site will include administration building, plant maintenance shop and warehouse, and other buildings. The camp will have individual dormitory-style rooms for 525 camp personnel. In addition, the security and medical office will be part of the permanent on-site camp.

1.12.9 Waste Rock Storage Facility

The project will require a waste rock pile to store all non-mineralized material from the pits. This material will be deposited on a waste dump north of the process plant. The project will also have a low-grade stockpile used to blend mill feed with high-grade underground material.

1.12.10 Tailings Storage Facility

The project TSF was designed with consideration towards having upstream composite liner system (geomembrane liner-low permeability soil liner) for minimizing seepage through the embankment during staging the facility's development and maximizing the reuse of process water from the facility. The TSF is designed to hold a total of 96,360 kt of tailings (66,456,000 m^3) of material. The dam will be constructed in three stages. All the tailings will be pumped overland from the process plant.

1.12.11 Site Water Management

Based on the rainfall frequency at the project site, the proposed water management structures include diversion channel, diversion ditches, collection ditches and collection ponds. The source of runoff water is from stockpile, excess from process plant, groundwater inflow to mining pit, surface runoff from precipitation, and the waste rock storage facilities. A preliminary site-wide water balance analysis was performed for the Kwanika mine site and the average condition is shown below.

1.13 Environmental, Permitting, and Social Considerations

The Kwanika-Stardust project involves the development of the Kwanika and the nearby Stardust copper-gold deposits. The site is accessible by forest service roads and Tsayta Lake Road and contains several kilometres of excavated trails. The project site is within a broad valley bordered by mountains of the Omineca Mountain range. Presently, the area is characterized by wilderness, forestry and mineral exploration land use. The nearest community is Takla Landing which encompasses the part of the reserve lands of the Takla First Nation. The access roads to Takla Landing are used for community access as well as logging and mining/mineral exploration work. The project site neighbours Kwanika Creek and West Kwanika Creek, which are tributaries flowing south to the larger Nation River. The proposed project site is contained entirely within the Kwanika Creek subwatershed. The property is located within the traditional territory of Takla First Nation (Takla).

1.13.1 Environmental Considerations

A number of limited field and screening environmental baseline studies and reports were completed in 2018 and 2019. The programs involved the collection of baseline data within the proposed project footprint area (as of 2018) and commenced the process of identifying potential environmental constraints and opportunities related to the proposed development of the project, including engineering designs and management plans for the construction, operation, and closure phases of the project. The reports also outlined recommended next steps for baseline data collection.

The 2018-2019 environmental programs covered a range of valued ecosystem components (VECs) and involved the following activities:

- hydrometric and climatic monitoring
- surface water quality monitoring
- hydrogeological monitoring and testing
- fish and fish habitat monitoring
- soils, vegetation and wildlife monitoring
- socio-economic and cultural baseline studies.

From a study area perspective, the baseline environmental studies were focused mainly on the areas potentially impacted by the Kwanika deposits and little information is available for the Stardust deposit area where underground development is proposed. In addition, there have been no baseline studies completed to date on air quality, noise, greenhouse gases and climate change, and groundwater quality, Ongoing and expanded baseline studies will be required to support the project through the feasibility and environmental assessment (EA)/permitting phases of the project. The results of baseline studies will be used to minimize impact of the project on valued ecosystem components and to optimize the location and operation of project infrastructure.

In addition to the above studies, a screening level tailings and waste rock facilities alternatives assessment was completed that included environmental criteria as part of the screening methodology and ratings. A preliminary geochemistry study was completed that assessed the potential for metal leaching and acid generation from tailings, mineralized material, and waste rock materials.

In terms of water management, the main consideration for the project is related to changes to the flow regime of Kwanika and West Kwanika Creeks which will require diversion around open pits and loss of fish habitat which will require fisheries authorization and habitat compensation measures. Mine contact water around all surface facilities will be managed in accordance with regulatory requirements and tested/treated as required prior to discharge to downstream receivers.

As the project progresses through the PFS and EA/permitting stage a number of environmental management and monitoring plans will be required for the purpose of guiding the development and operation of the project and mitigating and limiting environmental impacts. These plans will be complementary to the engineered designs that will be required for the storage of tailings, waste rock, mineralized material, and conveyance/storage/treatment of mine contact water (refer to Chapter 18 of this report). series of mine and environmental management and monitoring.

There are three provincial parks within 30 km of the project including: Nation Lakes Provincial Park located around 10 km south and downstream of the project; Omineca Provincial Park and Protected Area located around 20 km north and 25 km northeast of the project; and Mt. Blanchet Provincial Park around 30 km southeast of the project. The historic Bralorne Takla Mercury Mine is located within the property boundaries; however, this historic mine site is under the jurisdiction of the Crown Contaminated Sites Program (CCSP). A full cleanup program was completed on this site through CCSP in 2018. At this point, only ongoing monitoring through CCSP and their contractors is required. NorthWest Copper is not involved with or responsible for any of the ongoing monitoring programs.

1.13.2 Permitting Considerations

The major Federal legislation and associated authorizations related to the anticipated for the project include an Impact Assessment, issued under the Impact Assessment Act (IAA); and a Fisheries Act Authorization, issued under the Fisheries Act. When a project falls under both provincial and federal environmental assessment responsibility, there is an agreement in place between B.C. and Canada which enables the two governments to carry out a single, cooperative environmental assessment while retaining their respective decision-making powers. Provincial and federal ministers make independent decisions on whether to issue an Environmental Assessment Certificate (EAC) from a single report. The project as envisioned in this report will require a Fisheries Authorization and Fish Habitat Compensation Plan. A Schedule 2 amendment to the MDMER may also be required subject to further fish and fish habitat surveys required for areas where mine waste will be stored (waste rock, tailings, mineralized material, and untreated contact water / mine effluent).

The major provincial legislation and associated authorizations anticipated for the project include the following: a B.C. Mines Act Permit, issued under the Mines Act (MA); an Effluent and Air Emissions Permit, issued under the B.C. Environmental Management Act (EMA); and B.C. EAC, issued under the B.C. Environmental Assessment Act (EAA).

1.13.3 Closure and Reclamation Considerations

Under the B.C. Mines Act, anyone who engages in mining exploration work or mining operations determined by regulation must submit a reclamation plan. A conceptual reclamation and closure plan and a closure security estimate will need to be developed to support the submission of an EA report to the province or to the federal agency. The reclamation security will need to be posted to B.C. government prior to the commencement of construction the construction phase.

The current Conceptual Closure and Reclamation Plan for the project includes the following measures:

- Partial backfilling of open pits with waste rock, and flooding of the remaining open pit, and in the case of the Kwanika Central Pit, the underlying block cave mine, likely achieved by breaching the diversion dam and channel.
- The mineralized material stockpile will be reclaimed, once depleted.
- The mine portals will be decommissioned, plugged and backfilled.
- The plant and infrastructure pad will be dismantled, removed, and re-contoured and revegetated.
- The tailings dam will be vegetated to establish an erosion resistant surface.
- The tailings beach will be capped with soil and vegetated.
- Water treatment will be continued until the Tailing Management Facility (TMF) water quality meets discharge criteria.
- Once TMF water quality meets discharge criteria, water treatment will be stopped, diversions will be decommissioned, and the TFF will be allowed to discharge naturally via a closure spillway.
- At closure, Potentially Acid Generating (PAG) rock will be managed by: rehandling into the pit to keep it permanently submerged in the pit lake or capping it with low permeability glacial till to reduce seepage and oxygen infiltration. Non-Potentially Acid Generating (NPAG) waste rock stored on the surface will be capped with soil and revegetated.

1.13.4 Social Considerations

The project is located with the traditional territory of Takla. As of 2019, the traditional territory of Takla is in North-Central British Columbia and totalled around 27,250 km^2 including 17 reserves totalling 809 hectares. The closest community to the project is Takla Landing located around 30 km west of the project. Other First Nations potentially affected by the project include the Nak'azdli Whut'en First Nation and the McLeod Lake Indian Band (MLIB). The MLIB's traditional territories overlap or border those of Takla at or near the project; and include the downstream receiving waters including Tchentlo Lake, Nation River and Williston Reservoir.

Results from an Archaeology Overview Assessment (AOA) indicated no archeological sites are in direct conflict with the proposed development area, as identified in 2018. Culturally modified trees (pre-AD 1846) or other protected cultural remains may be present inside proposed development boundaries. A documented aboriginal and probable historic pack train trail (Kwanika Trail) is located in this study area. Overall, archaeological potential is rated moderate to high for the project site.

In early 2018 the Kwanika Copper Corporation (KCC) signed an agreement with Takla. The agreement outlined requirements for communication, information sharing, capacity funding and collaboration. The known traditional land and resource use within traditional territories include hunting, fishing, plant gathering, habitation, gathering places, sacred sites, trails and travel ways, and trapping. Within the 2018 exploration agreement with Takla it was understood that Takla's interests regarding traditional land use areas must be addressed through the Takla Nation Lands and Stewardship Department who are responsible for facilitating meetings for impacted Takla families and the community as a whole. Several seasonal cabins and campsites are reported to be within the local study area of the project site.

Community engagement activities during 2018 and 2019 included numerous meetings, community updates, employment and training opportunities, and sponsorship of a Takla career fair held in Takla Landing in March 2019. In 2018 and 2019 the exploration programs involved significant participation by the Takla community members and Takla contractors. Between 2020 and 2022, direct community engagement was limited due to COVID restrictions, however agreement implementation and involvement of Takla community members and contractors in the project continued. In 2021 and 2022, NorthWest Copper increased its direct interaction with the Takla Nation Lands and Stewardship Department to update the Wildlife Management and Mitigation Plan (WMMP), conduct an Archaeological Overview Assessment, update the Chance Find Procedure for the project, implement agreements and improve ongoing communications and reporting on project activities. Continued engagement and collaboration with Takla and other potentially impacted and affected communities is a top priority for NorthWest Copper going forward.

1.14 Capital and Operating Cost Estimates

The preliminary economics of the project can be assessed using the capital and operating cost estimates presented in this PEA. These calculations have been developed for an open pit and underground mining operation with a processing plant, supporting infrastructure, TSF, and owner's expenses and provisions.

1.14.1 Capital Cost Estimates

The capital costs provided in this PEA are reported in Canadian dollars (C$) with no allowance for escalation or exchange rate fluctuations. The capital cost estimate follows Class 5 guidelines for a PEA-level estimate with +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International).

The total initial capital cost for the project is C$567.9 million, the life of mine (LOM) sustaining cost including financing is C$282.5 million, and the LOM growth capital cost is C$493.3.

Table 1-10: Summary of Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Mining	1000	65.8	151.4	393.3	610.5
Process Plant	2000	198.0	0.0	0.0	198.0
Additional Process Facilities	3000	6.4	5.6	0.0	12.0
On-site Infrastructure	4000	21.6	4.9	0.0	26.5
Off-site Infrastructure	5000	82.5	78.5	0.0	161.0
Total Directs		**374.3**	**240.4**	**393.3**	**1008.0**
Project Preliminaries	6000	28.4	2.1	0.0	30.5
Project Delivery	7000	50.4	2.1	0.0	52.5
Owner's Costs	8000	33.7	27.3	100.0	161.0
Provisions	9000	81.1	10.5	0.0	91.6
Total Indirect		**193.6**	**42.0**	**100.0**	**335.6**
Project Total		**567.9**	**282.4**	**493.3**	**1343.6**

1.14.2 Operating Cost Estimates

Operating costs for the project include those related to mining, processing, infrastructure, tailings disposal, and general and administrative activities. The estimate for these operating costs conforms to Class 5 guidelines for a PEA-level estimate with +50%/-30% accuracy, according to the Association for the Advancement of Cost Engineering International (AACE International).

The LOM average unit operating cost is C$23.04/t milled, including an annual G&A cost of C$18.3 million.

Table 1-11: Operating Cost Summary

Cost Area	Total ($M)	C$/t Milled	% of Total
Mining	1,207.8	12.63	54.8
Process	776.9	8.13	35.3
G&A	218.1	2.28	9.9
Total	**2,202.9**	**23.04**	**100**

1.15 Economic Analysis

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

The economic analysis was performed assuming a 7% discount rate. Cash flows have been discounted to the start of construction, assuming that the project execution will be made, and major project financing will be carried at this time.

The pre-tax NPV discounted at 7% is C$440.1 million; the internal rate of return (IRR) is 17.1%; and payback period is 5.99 years. On a post-tax basis, the NPV discounted at 7% is C$215.0 million; the IRR is 12.7%; and payback period is 6.37 years. A summary of project economics is listed in Table 1-12.

Table 1-12: Economic Analysis Summary

Description	Unit	LOM Total / Avg.
General		
Copper Price	US$/lb	$3.63
Gold Price	US$/oz	$1,650
Silver Price	US$/oz	21.5
Exchange Rate	CAD:USD	0.77
Mine Life	Years	11.9
Total Mineralized Material Processed	kt	95,607
Total Waste	kt	86,926
Strip Ratio – Kwanika Central OP	waste tonnes:ore tonnes	1.9
Strip Ratio – Kwanika South OP	waste tonnes:ore tonnes	1.7
Production		
Average Feed Grade, Cu	%	0.39
Average Feed Grade, Au	g/t	0.39
Average Feed Grade, Ag	g/t	2.21
Average Open pit Mill Recovery Rate, Cu	%	84.3
Average Open pit Mill Recovery Rate, Au	%	60.0
Average Open pit Mill Recovery Rate, Ag	%	57.8
Average Underground Mill Recovery Rate, Cu	%	89.7
Average Underground Mill Recovery Rate, Au	%	71.4
Average Underground Mill Recovery Rate, Ag	%	70.3
Total Payable Copper	mlbs	694
Total Payable Gold	koz	803
Total Payable Silver	koz	3,204
Total Payable Copper Equivalent	mlbs	1,078
Operating Costs		
Mining Cost	C$/t Mined	$6.62
Mining Cost	C$/t Milled	$12.63
Processing Cost	C$/t Milled	$8.13
G&A Cost	C$/t Milled	$2.28
Refining and Transport Cost	C$/lb of Cu Eq.	$0.27
Cash Cost	C$/lb of Cu Eq.	$2.6
Ail-in Sustaining Costs	C$/lb of Cu Eq.	$1.5
Capital Costs		
Initial Capital	C$M	$567.9
Sustaining Capital	C$M	$282.5
Growth Capital	C$M	$493.3
Closure Costs	C$M	$41.9
Salvage Costs	C$M	($2.5)
Financials		
Pre-Tax NPV (7%)	C$M	$440.1
Pre-Tax IRR	%	17.1%
Pre-Tax Payback (Years)	Years	5.99
Post-Tax NPV (7%)	C$M	$215.0
Post-Tax IRR	%	12.7%
Post-Tax Payback (Years)	Years	6.37

1.15.1 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the project, using the following variables: commodity price, discount rate, foreign exchange, operating cost, and initial capital cost.

Table 1-13: Post-Tax NPV Sensitivity Analysis

Commodity Price Change (%)	Post-Tax NPV (7%) Base Case	Initial Capital Cost		Operating Cost		Foreign Exchange	
		(-20%)	20%	(-20%)	20%	(-20%)	20%
(20%)	($187)	($103)	($275)	($87)	($298)	$215	($541)
(10%)	$18	$97	($62)	$112	($77)	$456	($305)
--	$215	$292	$136	$307	$122	$694	($116)
10%	$408	$483	$332	$498	$317	$931	$51
20%	$598	$673	$524	$689	$508	$1,168	$215

Table 1-14: Post-Tax IRR Sensitivity Analysis

Commodity Price Change (%)	Post-Tax IRR Base Case	Initial Capital Cost		Operating Cost		Foreign Exchange	
		(-20%)	20%	(-20%)	20%	(-20%)	20%
(20%)	1.9%	3.7%	0.3%	4.6%	0.0%	12.7%	0.0%
(10%)	7.5%	10.0%	5.5%	10.0%	4.9%	18.6%	0.0%
--	12.7%	15.9%	10.2%	15.0%	10.3%	24.2%	3.8%
10%	17.5%	21.4%	14.5%	19.6%	15.3%	29.7%	8.4%
20%	22.0%	26.7%	18.6%	24.0%	19.9%	34.9%	12.7%

1.15.2 Markets and Contracts

There are currently no sales contracts or refining agreements in place for the Project. The marketing terms used in this study and addressed in this part were based on a preliminary marketing study.

The metal payables used in the marketing study are given below in Table 1-15. A summary of the treatment, refining, and transportation costs is provided in Table 1-16 and Table 1-17. There are no known deleterious elements that could significantly affect a potential future economic extraction. The QP is of the opinion that the information presented here is suitable for use in cashflow analyses to support this assessment.

Table 1-15: Metals Payables

Metal	Unit	Concentrate
Copper	%	96.0%
Less Deductible	%	0
Silver	%	70%
Less Deductible	g/t	0
Gold	%	97.5%
Less Deductible	g/t	0

Table 1-16: Transportation and Treatment Cost

Concept	Value	Unit
Transportation Cost	$ 100.00	Per wmt
Treatment Charge	$ 75.00	Per dmt

Table 1-17: Refining Charge

Refining Charge	Value	Unit
Copper	$ 0.075	per lb
Gold	$ 5.00	per oz
Silver	$ 0.40	per oz

1.16 Interpretation and Conclusions

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics (i.e., $215 M post-tax $NPV_{7\%}$ and 12.7% post-tax IRR). The PEA supports a decision to carry out additional detailed studies to progress the project further into detailed assessment.

1.17 Recommendations

1.17.1 Overall Recommendations

The Kwanika-Stardust project demonstrates positive economics, as shown by the results presented in this Technical Report. It is recommended to continue developing the project through prefeasibility study. Table 1-18 summarizes the estimated cost for the recommended future work on the Kwanika-Stardust Project.

Table 1-18: Cost Summary for the Recommended Future Work

Recommendation	Cost (C$)
Project Management	200,000
Mineral Resource (Including drilling)	6,000,000
Geotechnical & Hydrological Studies	2,300,000
Mine Engineering	450,000
Metallurgical Testwork	620,000
Process and Infrastructure Engineering	500,000
Geotechnical Studies (Infrastructure)	840,000
Topography	100,000
Geochemical Assessment	110,000
Water Management Studies	100,000
Environmental Studies and Permitting	300,000
Total	**11,520,000**

1.17.2 Mineral Resource Estimate

1.17.2.1 Kwanika Central and Kwanika South

The results of the recent exploration work clearly demonstrate that additional exploration is warranted. Drilling should continue at both Kwanika Central and Kwanika South to potentially extend and upgrade the mineral resources. Estimated cost for this task is $2,500,000.

1.17.2.2 Stardust

The results of the recent exploration programs clearly demonstrate that additional exploration is warranted. The program should continue to focus on expanding the Canyon Creek Skarn zone as well as testing for additional skarn lenses along the siliciclastic sedimentary – carbonate contact. Infill drilling should be carried out to upgrade Inferred resources to Measured or Indicated categories. Advanced metallurgical testing should also be carried out. Specific recommendations for a first phase program include:

- resource expansion drilling to potentially expand the mineral resources within the Canyon Creek Skarn Zone
- infill drilling to potentially upgrade inferred mineral resources to measured or indicated
- further metallurgical testing including comminution testing, locked cycle tests on main rock types, variability testing and detailed concentrate analysis to identify any potential deleterious elements that might impact marketability of the final concentrates.

The cost to implement the above recommendations is estimated to be $3,500,000.

1.17.3 Geotechnical and Hydrogeologic Studies

1.17.3.1 Kwanika

A review of the available geotechnical dataset in support of the Kwanika underground block cave mine plan indicates a need for further geotechnical and hydrogeologic information to advance to a PFS-level study. Specific recommendations include:

- expand the rock mass classification and geotechnical domains to encompass the projected mine development area
- complete drilling investigations for the proposed portal and decline route including sterilization drilling
- deepen drilling investigations to investigate opportunities to deepen or split production elevations
- complete in-situ stress measurements
- update the structural interpretations and model facilitated by acoustic televiewer surveys on existing and future drillholes
- complete hydrogeological modelling to increase confidence in water inflows from the various geological units during mine development and production.

The cost to implement the above recommendations is estimated to be $1,100,000.

1.17.3.2 Stardust

A review of the available geotechnical dataset in support of the Stardust underground mine plan indicates a need for further geotechnical and hydrogeologic information to advance to a PFS-level study. Specific recommendations include:

- complete geotechnical drilling investigations, guided by the location of major ramps, capital infrastructure and production areas to collect rock mass classification data, structural data and samples for rock material property testing. Structural information collection should be augmented with the application of geophysical acoustic televiewer surveys
- complete Matthew's Stability graph stope size optimization studies as well as Equivalent Linear Overbreak Slough (ELOS) assessments to refine production scheduling and stope dilution estimations
- develop geotechnical domains based on and updated structural model and geologic data
- complete in-situ stress measurements
- complete empirical and numerical modelling analyses to refine pillar sizing estimates for the mine plan
- complete hydrogeologic investigations to evaluate the impact of water flows on the development and production areas.

The cost to implement the above recommendations is estimated to be $1,200,000.

1.17.4 Mine Engineering

The following work is recommended:

- a study to better understand and quantify the pit internal dilution
- a material handling trade-off study should be conducted to determine the optimal plant location and optimal waste storage locations based on haul route cycle time analysis for the Kwanika deposits
- study mine automation such as automated truck hauling and mucking between shifts
- geotechnical and hydrogeological study for Kwanika Underground
- further block cave footprint and production schedule optimization for Kwanika underground in advanced stages of study
- first principles study for Stardust Mining and G&A costs
- geotechnical and hydrogeological study for Stardust

The recommended budget for these works is $450,000.

1.17.5 Metallurgical Testwork

The metallurgical work outlined below is recommended for the next project phase and should be completed on samples originating from ½ drill core.

- sample selection for future mining studies should reflect mineralization that would be treated throughout the mine life; variability samples are required to understand the responses of the various mineralized zones
- feed mineralogy tests

- additional comminution tests—including include SMC, RWi, BWi and Ai tests—to further expand the comminution database is recommended to develop a robust comminution model and grinding circuit design; this will improve the future analysis of power requirements and equipment selection.
- bench scale flotation tests including rougher, cleaner, and locked cycle tests to should investigate the potential to apply a primary grind sizing that is coarser than 75 µm P_{80}
- bench scale tests for assessment of preconcentration technologies such as coarse particle flotation and sensor sorting
- expected Mo levels in the resource should be confirmed and appropriately represented in future test samples; if adequate Mo levels are present, bulk concentrate generation and subsequent Cu-Mo separation tests should be arranged.

The estimated cost of the above recommended metallurgical testwork and the cost of testwork management is $620,000.

1.17.6 Process and Infrastructure Engineering

The estimated cost for process and infrastructure engineering for the PFS is $500,000. Engineering deliverables would include:

- process trade-off studies (comminution, cyanidation options and preconcentration studies)
- flow diagrams (comminution, recovery processes, tails)
- detailed equipment list
- power listing and consumption estimate
- architectural (building sizes) to estimate steel and concrete quantities
- detailed material and water balance
- detailed process design criteria
- GA and Elevation drawings (for crushing/overland conveying, comminution, leaching, recovery, reagents)
- electrical single line drawing
- equipment and supply quotations updated, and sources determined
- estimate of equipment and materials freight quantities
- capital cost estimate
- operating cost estimate
- major equipment spares and warehouse inventory cost estimate
- construction manpower estimate
- construction schedule

The following activities are recommended to support infrastructure design for the PFS phase.

1.17.7 Site-Wide Assessment and Tailings Storage Facility Studies

Due to the conceptual nature of this study and the paucity of information available at the time of writing, assumptions have been made regarding the layout, MTOs, and construction of the proposed TSF. Construction material geotechnical properties are required to perform slope stability analyses and other geotechnical assessments to confirm that the TSF can be built as designed. A tailings deposition plan will be required which may lead to the conceptual staging requiring adjustment to contain the given capacities.

Additional studies and data collection will be required to advance project development beyond the conceptual level. Some, but not necessarily all, of the current data gaps that would need to be addressed in future studies include the following:

- Geological and geotechnical site investigations and laboratory program should be carried out for infrastructure, Process plant, WRSF and TSF, including drilling and in-situ and laboratory testing, to understand subsurface soil and rock characteristics, construction material properties, and existing groundwater levels.
- Seepage analysis for the TSF needs to be investigated.
- Additional geotechnical testing of the anticipated tailings, waste rock, and other associated construction materials, (e.g., horizontal drain gravel and sand and candidate geomembranes) should be carried out.
- Hydrological information should be gathered from site-specific climate studies to detail ponds and channels.
- Hydrogeological information from desktop studies and site investigations should be gathered to better understand subsurface flow regimes.
- A trade-off study between dry stacking of tailings vs conventional disposal of tailings.

As additional information is obtained, assumptions made in this study can be verified or updated to advance the project to the next level of design. The cost of implementing the above recommendations is estimated at CAD $840,000.

1.17.8 Water Management Studies

- A detailed site-wide stochastic water balance modelling using GoldSIM (Monte-Carlo) should be completed for the next phase of the study: This would inform of potential needs for water treatment and support future financial analysis. Estimated cost for this item is $50,000.
- Packer testing should be conducted to determine pit hydrogeology, hydraulic conductivity and refine pit water inflow estimates.
- Further hydrogeological and hydrological characterization are required in the pit areas.
- Updating the water management structure design based on the updated site layout. Estimated cost for this item is $50,000.

The cost of carrying out the above work is estimated at CAD $100,000.

1.17.9 Geochemical Assessment

For proceeding to a PFS-level study, the general level of effort required to establish the ARD/ML risk for a typical project would generally comprise:

- around 200 – 300 waste rock samples

- six to 12 tailings samples (if composition different)
- six to 12 mineralized material samples
- several overburden samples
- range of tests to include:
  - elemental analysis
  - acid base accounting
  - shake flask extraction (short term leach)
  - net acid generation (NAG) pH
  - mineralogy
  - humidity cell testing (minimum 40 weeks)

The estimated cost for the recommended lab testwork is $80,000.

- To better assess the ARD/ML risk from tailings, confirmation of the type of tailings streams (i.e., spiral / flotation / cyanidation) and the percentage ratios of each type that will be deposited in the tailings storage facility.
- If available, the results of testing of historical mine wastes and site water quality data should be reviewed as this can provide useful supporting information to aid in assessing the existing geochemistry data.

The estimated cost of the assessment is $30,000. The total cost for geochemical assessment is $110,000.

1.17.10 Topography

A site-wide LiDAR survey is recommended to define the site topography at higher accuracy. The current topography is based on SRTM which is sufficient for PEA, however, higher definition will be required in the PFS. The estimated cost for this task is $100,000.

1.17.11 Environmental, Permitting, Social and Community Recommendations

The following recommendations are made regarding future studies and activities related to areas of environment, permitting and community engagement. These studies and activities will be necessary to support the project to the PFS stage and provide a strong basis for future EA preparation and permitting.

1.17.11.1 Water Resources

- Development and implementation of s hydrological and meteorological monitoring plan for key areas within the study area will be required to further characterize the hydrological conditions and to develop a future water balance model.
- Development and implementation of a surface water and groundwater monitoring, sampling, and testing plan and focusing on areas that will be potentially affected by mine infrastructure based on current infrastructure plans (refer to Chapter 18) and should meet the requirements of an Environmental Assessment application (ENV 2016). The surface and groundwater baseline and testing data will need to support the development of a future integrated numerical 3-D groundwater predictive model and overall water balance model for the site.

- Hydrogeological testing of monitoring wells to support groundwater inflow estimates for pits and underground workings.
- Development of a conceptual groundwater model.
- Estimated cost for these tasks is $150,000.

1.17.11.2 Geochemistry

- The geochemical testing results do not include all current deposits/pits being considered for development and further work is required for those areas, specifically the Kwanika South deposit and the Stardust deposit areas. Additional sample selection and analyses are recommended including for mine waste rock/tailings and mineralized materials. Detailed description and costs for the proposed geochemical program is provided in Section 26.8, above.

1.17.11.3 Fish and Fish Habitat

- Additional fish and fish habitat sampling and assessments are recommended for the areas of proposed project disturbance.
- Estimated cost for this task is $50,000.

1.17.11.4 Terrestrial and Wildlife Monitoring

- Additional surveys will need to be completed related to the areas of vegetation/ecosystem and wildlife/wildlife habitat for the mine infrastructure presented in Chapter 18. The results of those surveys should be used to develop plans that will eliminate or mitigate environmental risk for the purpose of PFS.
- Takla and other land users should be closely involved in the development and execution of wildlife baseline studies, especially in relation to traditional and current use of the land for harvesting.
- Estimated cost for this task is $40,000.

1.17.11.5 Socio-Economic, Cultural Baseline Studies and Community Engagement

- Archaeological Overview or Impact Assessment to be completed on locations of proposed project infrastructure.
- It is important that NorthWest Copper carry on with commitments previously made to Takla including:
    - continuing to meet as a group for follow-up discussion on project plans
    - developing Terms of Reference (TOR) for defining collaboration process and procedures
    - working towards defining communications, processes and procedures to guide the project through the next stages.
- Estimated cost for this task is $50,000.

1.17.11.6 Environmental Constraints Mapping

- To assist in the development of the project at the PFS stage, environmental constraint mapping should be developed and continuously updated, based on the results of historical and future baseline environmental and land use studies. This mapping should be utilized to limit risks at the design stages of the project.
- The estimated cost for this task is $10,000.

2 INTRODUCTION

2.1 Introduction

NorthWest Copper commissioned Ausenco Engineering Canada Inc. and Ausenco Sustainability Inc. (Ausenco) to compile a Preliminary Economic Assessment (PEA) of the Kwanika-Stardust project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and the requirements of Form 43-101 F1.

The Kwanika-Stardust project involves the development of the Kwanika deposit and the Stardust deposit. The Kwanika-Stardust Project consists of a copper-gold deposit located around 195 kilometres by road from Fort St. James, British Columbia, Canada and 40 km from the community of Takla Landing.

The responsibilities of the engineering companies whom NorthWest Copper contracted to prepare this report are as follows:

- Ausenco managed and coordinated the work related to the report and developed PEA-level design, including cost estimates for the process plant, general site infrastructure, tailings storage facility, waste rock storage facility, environment and permitting, and economic analysis.
- Mining Plus designed the open pit and underground mining, mine production schedule, and mine capital and operating costs.
- Apex Geosciences Ltd. developed the mineral resource estimate for the project and completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification and completed a review of the environmental studies.

2.2 Terms of Reference

The report supports disclosures by NorthWest Copper in a news release dated January 5, 2023, titled, "NorthWest Copper Announces Positive PEA for the Kwanika-Stardust Copper-Gold Project, Describing a Low Capex Project with Scale."

Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

2.3 Qualified Persons

The Qualified Persons for the report are listed in Table 2-1. By virtue of their education, experience, and professional association membership, they are considered Qualified Person as defined by NI 43-101.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Issuer Name(s)	Report Section
Kevin Murray	P. Eng.	Manager, Process Engineering	Ausenco Engineering Canada Inc.	Yes	1.1, 1.11, 1.12.1-1.12.8, 1.14 - 1.16, 1.17.1, 1.17.6, 2, 3.1, 3.3, 3.4, 17, 18.1, 18.2, 18.3.1-18.3.6, 18.3.9, 19, 21.1, 21.2.1-21.2.3, 21.2.5-21.2.10, 21.2.11.1, 21.2.11.3, 21.2.11.4, 21.3.1, 21.3.2, 21.3.4, 21.3.5, 22, 24, 25.1, 25.7, 25.8, 25.10-25.13, 25.14.1.5, 25.14.1.9-25.14.1.13, 25.15, 26.1, 26.6, 27
Jonathan Cooper	P. Eng.	Senior Water Resources Engineer	Ausenco Sustainability Inc.	Yes	1.12.11, 1.17.8, 1.17.9, 16.3, 18.3.10, 25.14.1.7, 26.8, 26.9
Peter Mehrfert	P. Eng.	Principal Process Engineer	Ausenco Engineering Canada Inc.	Yes	1.8, 1.17.5, 13, 25.4, 25.14.1.4, 25.14.2.2, 26.5
Scott Elfen	P. E.	Global Lead Geotechnical Services	Ausenco Engineering Canada Inc.	Yes	1.12.10, 1.17.7, 1.17.10, 18.3.8, 21.2.12, 25.14.1.6, 26.7, 26.10
Scott Weston	P. Geo.	Vice President, Business Development	Ausenco Sustainability Inc.	Yes	1.13, 1.17.11, 3.2, 20, 25.9, 25.14.1.8, 26.11
Cale DuBois	P. Eng.	Principal Consultant	Mining Plus Canada Consulting Ltd.	Yes	1.17.3, 16.2, 25.6.1, 26.3
Jason Blais	P. Eng.	Manager, Plan Division	Mining Plus Canada Consulting Ltd.	Yes	1.10.1, 1.10.2, 12.1.1, 15, 16.4, 25.6.2, 25.6.3, 25.14.1.3.1, 25.14.1.3.2, 25.14.2.3.1, 25.14.2.3.2
John Caldbick	P. Eng.	Senior Principal Consultant	Mining Plus Canada Consulting Ltd.	Yes	1.10.3, 1.12.9, 1.17.4, 16.1, 16.5, 16.6, 18.3.7, 21.2.4, 21.2.11.2, 21.3.3, 25.6.4, 25.14.1.3.3, 25.14.2.3.3, 26.4
Brian S. Hartman	P. Geo.	Principal Consultant	Ridge Geosciences LLC.	Yes	1.2.1, 1.3.1, 1.4.1, 1.5.1, 1.6.1, 1.7.1, 1.9.1, 1.9.2, 1.17.2.1, 4.1, 5, 6.1, 7.1, 7.2, 8.1, 9.1, 10.1, 11.1, 12.1.2, 12.1.3, 14.1, 14.2, 23, 25.2, 25.3, 25.5.1, 25.14.1.1, 25.14.1.2, and 26.2.1 as they pertain to Kwanika Central and Kwanika South
Ronald G. Simpson	P. Geo.	Consultant	GeoSim Services Inc.	Yes	1.2.2, 1.3.2, 1.4.2, 1.5.2, 1.6.2, 1.7.2, 1.9.3, 1.17.2.2, 4.2, 6.2, 7.3, 8.2, 9.2, 10.2, 11.2, 12.2, 14.3, 21.2.11, 25.5.2, 25.14.2.1, and 26.2.2 as they pertain to Stardust.

2.4 Site Visits and Scope of Personal Inspection

A summary of the site visits completed by the QP's is presented in Table 2-2.

Table 2-2: Summary of QP's Site Visits

Qualified Person	Date of Site Visit	Days on Site
Brian Hartman	Has not visited site	-
Cale DuBois	Has not visited site	-
Jason Blais	September 20, 2022	1
John Caldbick	Has not visited site	-
Jonathan Cooper	Has not visited site	-
Kevin Murray	Has not visited site	-
Peter Mehrfert	Has not visited site	-
Ronald Simpson	June 14, 2010, October 19, 2017, and September 23, 2020	3
Scott Elfen	September 20 and 21, 2022	2
Scott Weston	Has not visited site	-

Jason Blais conducted a site visit on September 20, 2022 and was accompanied by Scott Elfen (Ausenco Engineering) and Kyle Dziama (NorthWest Copper). The site was accessed by vehicle using the available network of logging roads and drill site access roads. The following activities were performed during the site visit:

- Mine Site Access – Driving through the proposed site access
- Observing mine project infrastructure and laydown area
- Observing drillhole core geology and geotechnical logging in Kwanika Core Shack (QA/QC, core cutting, sampling, logging instrumentation)
- Observing representative 2021 drill program drill core and obtaining corresponding drillhole logging information.
- Talking with NorthWest Copper project geologists and project geotechnical engineers
- Observing drillhole core laydown areas and representative hole intervals
- Visiting drillhole collars for representative holes from Kwanika Central and Kwanika Block Cave and observing Kwanika Central open pit area.
- Visiting drillhole collars for representative holes from Kwanika South zone and observing Kwanika South proposed open pit area
- Visiting the proposed TMF Northern and South Dam Location
- Observing the proposed Stardust portal and exhaust raise collar
- Observing the proposed Kwanika Central Block Cave decline portal, conveyor portal, exhaust raise, and intake raise locations.

2.5 Effective Date

This technical report has the following significant dates:

- Mineral Resource Statements: January 4, 2023 (effective date)
- Financial analysis: January 4, 2023.

The effective date of this report is based on the date of the financial analysis, which is January 4, 2023.

2.6 Information Sources and References

2.6.1 General

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

2.6.2 Previous Technical Reports

- "Stardust Project Updated Mineral Resource Estimate NI 43-101 Technical Report." Report prepared by Ronlad G. Simpson of Geosim Services Inc. Effective Date: July 2, 2021.
- "NI 43-101 Technical Report for the Kwanika Project Resource Estimate Update 2019." Report prepared by Moose Mountain Technical Services. Effective Date: April 17, 2019.
- "Stardust Project NI 43-101 Technical Report." Report prepared by Ronald G. Simpson of Geosim Services Inc. Effective date: January 8, 2018.
- "NI 43-101 Technical Report for the Kwanika Project Preliminary Economic Assessment Update 2017." Report prepared by Moose Mountain Technical Services. Effective Date: April 19, 2017.
- "Independent Technical Report for the Kwanika Copper-Gold Project, Canada." Report prepared by SRK Consulting (Canada) Inc. Effective Date: December 1, 2016.
- "NI 43-101 Technical Report for The Kwanika Property Preliminary Economic Assessment 2013." Report prepared by Moose Mountain Technical Services. Effective Date: March 4, 2013.
- "NI 43-101 Technical Report on the Kwanika Project, Fort ST. James, British Columbia, Canada." Report prepared by Roscoe Postle Associates INC. Effective Date: March 3, 2011.
- "NI 43-101 Technical Report on the Kwanika Project, Fort ST. James, British Columbia, Canada." Report prepared by Scott Wilson Mining. Effective Date: March 4, 2010.
- "NI 43-101 Technical Report on the Kwanika Project, Fort ST. James, British Columbia, Canada." Report prepared by Scott Wilson Mining. Effective Date: April 8, 2009.

2.7 Abbreviations and Acronyms

Abbreviation	Description
AACE	Association for the Advancement of Cost Engineering International
AAS	Atomic Adsorption Spectroscopy
Ag	Silver
AMP	Adaptive Management Plan
As	Arsenic
ARD	Acid rock drainage
Au	Gold
BWi	Bond Work Index
Cd	Cadmium
CD	Clastic-dominated
CDE	Canadian Development Expense
CEE	Canadian Exploration Expenses
CG	Canadian Government
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CoV	Cut-off Value
CRD	carbonate replacement massive sulphides deposits
CRF	Cemented Rock Fill
CRM	Certified reference material
Cu	Copper
CV	Coefficient of Variation
DCF	Discounted cash flow
DDH	Diamond drillhole
DEM	Digital Elevation Model
DGPS	Differential global positioning system
E	east
EA	Environmental Assessment
ECCC	Environment and Climate Change Canada
EM	Electromagnetic
ENE	East northeast
FSR	Forest Service Road
FX	Foreign Exchange Market
G&A	General and Administrative
GDL	Global Discovery Labs
GPS	Global positioning system
Hg	Mercury
HG	High-grade
HV	High voltage
ICP	Inductively coupled plasma
IP	Induced polarization
IRR	Internal rate of return
LiDAR	laser imaging, detection and ranging
LOM	Life of mine

Abbreviation	Description
MA	Mechanical availability
MIBC	Methyl Isobutyl Carbinol
ML	Mineral Leases
Mo	Molybdenum
MRE	Mineral Resources Estimate
MTO	Material quantity take off
NAG	Non-acid generating
N	north
NN	Nearest Neighbour
NNE	North northeast
NNW	North-northwest
NPV	Net present value
NSP	Net smelter price
NSR	Net smelter return
OTCQX	Over-the-counter stock market
Pa	Pascal
PAG	Potential acid generating
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Prefeasibility study
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
ROM	Run-of-mine
RWi	Bond Rod Work Index
S	south
SAG	Semi-autogenous grinding
SG	Specific gravity
SGS	SGS Canada
SMC	SAG mill comminution
SSE	South-southeast
SW	Southwest
TC	Treatment costs
TSF	Tailings Storage Facility
UTM	Universal Transverse Mercator
W	west
WBS	Work breakdown structure
WNW	West-northwest
WRSF	Waste Rock Storage Facilities
WSA	Water Security Agency
WSW	West southwest
Zn	Zinc

Table 2-3: Units of Measure

Abbreviation	Description
%	Percent
°	degree
°C	Celsius
C$	Canadian dollar
cm	centimetre
cm^2	square centimetre
dmt	dry metric tonne
g/l	grams per litre
g/t	grams per tonne
h	hour
ha	hectare
kg	kilogram
km	kilometre
koz	thousand ounces
Kt	kilotonne
kW	kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per tonne
L	litre
lb	pound
L/s	Litres per second
M	million
m	metre
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
mbgs	Metres below ground surface
mm	millimetre
µm	micron
Mt	megatonne
Mt/a	Million tonnes per annum
MW	megawatt
oz	ounce
ppm	parts per million
ppb	parts per billion
S	second
t	tonne
t/d	metric tonnes per day
t/h	metric tonnes per hour
US$	United States dollar
wmt	wet metric tonne
w/w	percent solids

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, environmental, permitting, social licence, closure, taxation, and marketing for sections of this Report.

3.2 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information supplied by NorthWest Copper and experts retained by NorthWest Copper for information related to environmental (including tailings and water management) permitting, permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Archer CRM, 2008. Archaeological Overview Assessment – Kwanika Claim. Prepared for Serengeti Resources Inc.
- B.C. Conservation Data Centre. 2018. B.C. Species and Ecosystems Explorer. B.C. Ministry of Environment. Victoria, B.C. Available: http://a100.gov.bc.ca/pub/eswp/ Accessed February 2019.
- British Columbia Ministry of Environment (ENV). 2016. Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators. Version 2. June 2016.
- Falkirk Resource Consultants Ltd., 2018. Kwanika PFS Kwanika Copper-Gold Project - Initial Summary of Terrestrial Assessment. memorandum prepared for Serengeti Resources.
- Falkirk Resource Consultants, 2019a. Kwanika Project – Preliminary Socio-Economic, Cultural Baseline Studies and Community Engagement. Prepared for Serengeti Resources Inc.
- Klohn Crippen Berger Ltd., 2018. Kwanika Prefeasibility Study – Tailings and Waste Rock Alternatives Assessment. Letter report submitted to Merit Consultants International.
- Klohn Crippen Berger Ltd., 2019. Kwanika Prefeasibility Study – Waste Rock Geochemical Characterization – Preliminary Static Results DRAFT. Report submitted to Merit Consultants International.
- Klohn Crippen Berger Ltd., 2019a. Kwanika Prefeasibility Study – 2018 Hydrogeology Site Investigation. Report submitted to Merit Consultants International.
- Palmer Environmental Consulting Group Inc., 2019. Kwanika Copper Corporation, – 2018 Hydrology and Climate Technical Report. Report submitted to Falkirk Resource Consultants Ltd.
- Palmer Environmental Consulting Group Inc., 2019a. Kwanika Copper Corporation, 2018 Water Quality Technical Report. Report submitted to Falkirk Resource Consultants Ltd.
- Palmer Environmental Consulting Group Inc., 2019b. Kwanika Copper Corporation, 2018 Fish and Fish Habitat Technical Report. Report submitted to Falkirk Resource Consultants Ltd.

This information is used in Section 20 of the Report. The information is also used in support of the recommendations provided in Chapter 26 of the Report.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by experts retained by NorthWest Copper for information related to taxation as applied to the financial model, as received by email from Peter Lekich on January 4, 2023, titled, “Kwanika PEA-Press Release for Review.” NorthWest Copper retained Sadhra & Chow LLP, Chartered Professional Accountants, to provide taxation input. This information is used in the economic analysis in Section 22.

3.4 Markets

The QPs have fully relied upon, and disclaim responsibility for information derived from NorthWest Copper, and experts retained by NorthWest Copper for this information as received by email from Peter Bell on December 07, 2022, titled, “Kwanika PEA - Financial Model and Slide deck.” The email contained information related to commodity pricing, marketing terms, TC & RC, and concentrate transportation costs.

This information is used in Section 19 of this Report. The information is also used in support of Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

The Kwanika and Stardust Properties are discussed separately below.

4.1 Kwanika

The Kwanika property is located in North-Central British Columbia, in the Omineca Mining Division, around 140 km northwest (around 200 km by road) of Fort St. James. The project area is on NTS map sheets 93N06 and 93N11, at latitude 55.53º N and longitude 125.35º W.

4.1.1 Tenure History

Various claims in the area have been held by different operators throughout the years. Between 1965 and 1991, Hogan Mines, Canex Aerial Explorations, Great Plains Development Company of Canada Ltd., Bow River Resources, Pechiney Development Ltd., Placer Developments Ltd., Aume Resources Ltd., Daren Resources Ltd., and Eastfield Resources Ltd. held different tenures in the area and allowed them to lapse. In 1995 Discovery Consultants initially staked three, two-post claims east of Kwanika Creek covering the area of the historic South Zone resource and added a fourth claim in 2000. The amalgamated tenure lapsed on May 18,2002.

The entire Kwanika valley bottom was open ground when Myron Osatenko and David Moore of Serengeti Resources Inc (Serengeti) staked the ground in the late fall of 2004 and added significantly to the block on November 30th, 2004 when online staking became available.

In 2016 Serengeti formed a Joint Venture agreement with POSCO DAEWOO Corporation (POSCO) forming the private company Kwanika Copper Corporation (Kwanika Copper) which then owned the Kwanika tenure. POSCO had the option to earn in up to 35% of Kwanika Copper. In 2017, POSCO had met the requirements and had earned 35% of Kwanika Copper with Serengeti controlling the remaining 65%.

From 2019 onward Serengeti started to earn back shares of Kwanika Copper. On March 5th of 2021, Serengeti, and Sun Metals Corporation (Sun Metals) merged to form NorthWest Copper. This was followed by an announcement on December 29, 2021, of NorthWest Copper's intent to purchase the remaining 31% share of Kwanika Copper from POSCO. The deal consisted of three tranches of consideration shares issued to POSCO. The first and second tranche were completed on February 23 and April 25, 2022, the third and final tranche was closed on September 7, 2022. As part of the Tranche 1 closing, the shareholder joint venture agreement was terminated and any interest or rights of POSCO with respect to Kwanika project under the shareholder joint venture agreement, including offtake rights, were also terminated.

4.1.2 Mineral Tenure

NorthWest Copper owns a 100% interest in the Kwanika project, which is situated amongst a group of 59 unpatented mineral claims covering an area of 24,152.04 ha. The property is not subject to any royalties or other outstanding liabilities. There are no government minimum spend obligations to retain these claims. Table 4-1 lists the claims for the Kwanika project area and Figure 4-2 shows the claim map. The resource outlined in this report is contained within claims 501733, 514432, 514433, and 502953.

Figure 4-1: Kwanika Property Location Map

Source: NorthWest Copper, 2022.

Table 4-1: Mineral Tenure Information for the Kwanika Project

Title Number	Claim Name	Map Number	Issue Date	Good to Date	Status	Area (ha)
501129	GER	093N	2005/JAN/12	2033/JAN/26	GOOD	458.017
501134	Val 6	093N	2005/JAN/12	2033/JAN/26	GOOD	458.385
501190	GER1	093N	2005/JAN/12	2033/JAN/26	GOOD	457.821
501250	VAL7	093N	2005/JAN/12	2033/JAN/26	GOOD	458.378
501521	VAL8	093N	2005/JAN/12	2033/JAN/26	GOOD	440.233
501733	Kwanika 1	093N	2005/JAN/12	2032/DEC/04	GOOD	457.642
502038	VAL11	093N	2005/JAN/12	2033/JAN/26	GOOD	183.35
502129	Val 12	093N	2005/JAN/12	2033/JAN/26	GOOD	458.443
502168	VAL13	093N	2005/JAN/12	2033/JAN/26	GOOD	440.375
502196	Val 14	093N	2005/JAN/12	2033/JAN/26	GOOD	293.507
502227	Val 15	093N	2005/JAN/12	2033/JAN/26	GOOD	275.234
502953	Kwanika4	093N	2005/JAN/13	2032/DEC/04	GOOD	73.296
505271		093N	2005/JAN/31	2032/DEC/04	GOOD	458.168
505277	Kwanika5	093N	2005/JAN/31	2032/DEC/04	GOOD	458.45
506007	kwanika7	093N	2005/FEB/06	2032/DEC/04	GOOD	458.624
514432		093N	2005/JUN/13	2032/NOV/19	GOOD	439.522
514433		093N	2005/JUN/13	2032/NOV/19	GOOD	403.038
514447		093N	2005/JUN/13	2033/JAN/26	GOOD	458.39
514448		093N	2005/JUN/13	2033/JAN/26	GOOD	458.388
514449		093N	2005/JUN/13	2033/JAN/26	GOOD	274.935
514450		093N	2005/JUN/13	2033/JAN/26	GOOD	458.061
514451		093N	2005/JUN/13	2033/JAN/26	GOOD	513.05
514455	KWANIKA 8	093N	2005/JUN/13	2032/JUN/13	GOOD	18.316
546495	KWANIKA 9	093N	2006/DEC/04	2032/DEC/04	GOOD	458.7669
546496	KWANIKA 10	093N	2006/DEC/04	2032/DEC/04	GOOD	458.8842
546497	KWANIKA 11	093N	2006/DEC/04	2032/DEC/04	GOOD	458.9818
546498		093N	2006/DEC/04	2032/DEC/04	GOOD	459.0775
546500	KWANIKA 13	093N	2006/DEC/04	2032/DEC/04	GOOD	459.1835
546501	KWANIKA 14	093N	2006/DEC/04	2032/DEC/04	GOOD	459.2853
546502	KWANIKA 15	093N	2006/DEC/04	2032/DEC/04	GOOD	459.3943
546503	KWANIKA 16	093N	2006/DEC/04	2032/DEC/04	GOOD	459.5061
546507		093N	2006/DEC/04	2032/DEC/04	GOOD	459.65
546508	KWANIKA 18	093N	2006/DEC/04	2032/DEC/04	GOOD	459.8098
546509	KWANIKA 19	093N	2006/DEC/04	2032/DEC/04	GOOD	460.0162
546510	KWANIKA 20	093N	2006/DEC/04	2032/DEC/04	GOOD	460.2152
546511	KWANIKA 21	093N	2006/DEC/04	2032/DEC/04	GOOD	460.3846
546512	KWANIKA 22	093N	2006/DEC/04	2032/DEC/04	GOOD	18.4218
546553	KWANIKA 24	093N	2006/DEC/04	2032/DEC/04	GOOD	18.3287
546554	KWANIKA 25	093N	2006/DEC/04	2032/DEC/04	GOOD	36.6609
546555	KWANIKA 26	093N	2006/DEC/04	2032/DEC/04	GOOD	36.6704
546556	KWANIKA 27	093N	2006/DEC/04	2032/DEC/04	GOOD	55.0316
546557	KWANIKA 28	093N	2006/DEC/04	2032/DEC/04	GOOD	36.6974

Title Number	Claim Name	Map Number	Issue Date	Good to Date	Status	Area (ha)
546558	KWANIKA 29	093N	2006/DEC/04	2032/DEC/04	GOOD	18.3516
959209	VAL 16	093N	2012/MAR/12	2033/JAN/26	GOOD	385.2572
959229	VAL 17	093N	2012/MAR/12	2033/JAN/26	GOOD	330.3451
997183	KWANIKA EAST 1	093N	2012/JUN/14	2033/JAN/26	GOOD	457.0699
997222	KWANIKA EAST 2	093N	2012/JUN/14	2033/JAN/26	GOOD	438.8143
997242	KWANIKA EAST 3	093N	2012/JUN/14	2033/JAN/26	GOOD	457.315
997247	KWANIKA EAST 4	093N	2012/JUN/14	2033/JAN/26	GOOD	384.1428
997262	KWANIKA EAST 5	093N	2012/JUN/14	2033/JAN/26	GOOD	457.0691
997322	KWANIKA EAST 6	093N	2012/JUN/14	2033/JAN/26	GOOD	274.244
997342	KWANIKA EAST 7	093N	2012/JUN/14	2033/JAN/26	GOOD	365.8306
1012554	VAL 18	093N	2012/SEP/04	2033/JAN/26	GOOD	18.3413
1018213	SMOKE	093N	2013/APR/02	2033/JAN/26	GOOD	1810.804
1018214	ROTTACKER	093N	2013/APR/02	2033/JAN/26	GOOD	1784.5318
1018215	ROTTACKER	093N	2013/APR/02	2033/JAN/26	GOOD	294.1287
1018949	SMOKE 2	093N	2013/APR/29	2033/JAN/26	GOOD	658.2911
1031342	KGV	093N	2014/OCT/03	2033/JAN/26	GOOD	530.9147
1044440	KGV	093N	2016/MAY/30	2033/JAN/26	GOOD	457.9959
					Total:	**24152.0363**

4.1.3 Surface Rights

Surface rights over the Kwanika property are owned by the Crown and administered by the Government of B.C. and would be available for any eventual mining operation. The ownership of other rights (Aboriginal, placer, timber, water, grazing, trapping, outfitting, etc.) affecting the property were not investigated by the author.

4.1.4 Agreements

NorthWest Copper and its predecessor Serengeti have worked closely with the Takla on the Kwanika project. On September 14th, 2020, a new Exploration Agreement was announced between Serengeti (now Northwest Copper) and Takla. The new Exploration Agreement replaced an expired agreement and is valid through to September 14th, 2025. The agreement respects Aboriginal title, rights, and interests, and continues to recognize Takla's stewardship role in environmental and wildlife management and monitoring and traditional land use and knowledge.

4.1.5 Royalties

The property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

4.1.6 Permits and Authorizations

NorthWest Copper has an exploration permit issued by the B.C. Ministry of Energy and Mines and Low Carbon Innovation authorizing mineral exploration for the Kwanika project. The permit is good until August 19, 2027, with the option to renew at the discretion of the B.C. Ministry of Energy and Mines and Low Carbon Innovation.

4.1.7 Environmental Considerations

NorthWest Copper conducts routine baseline environmental monitoring through engagement and in collaboration with First Nations rights and titleholders. This includes measuring surface water flow, water quality, and recording wildlife sightings. Additionally, NorthWest Copper maintains a weather station on the adjacent Stardust property that is appropriate for collecting relevant climate data for this location. There are no known environmental liabilities on the property.

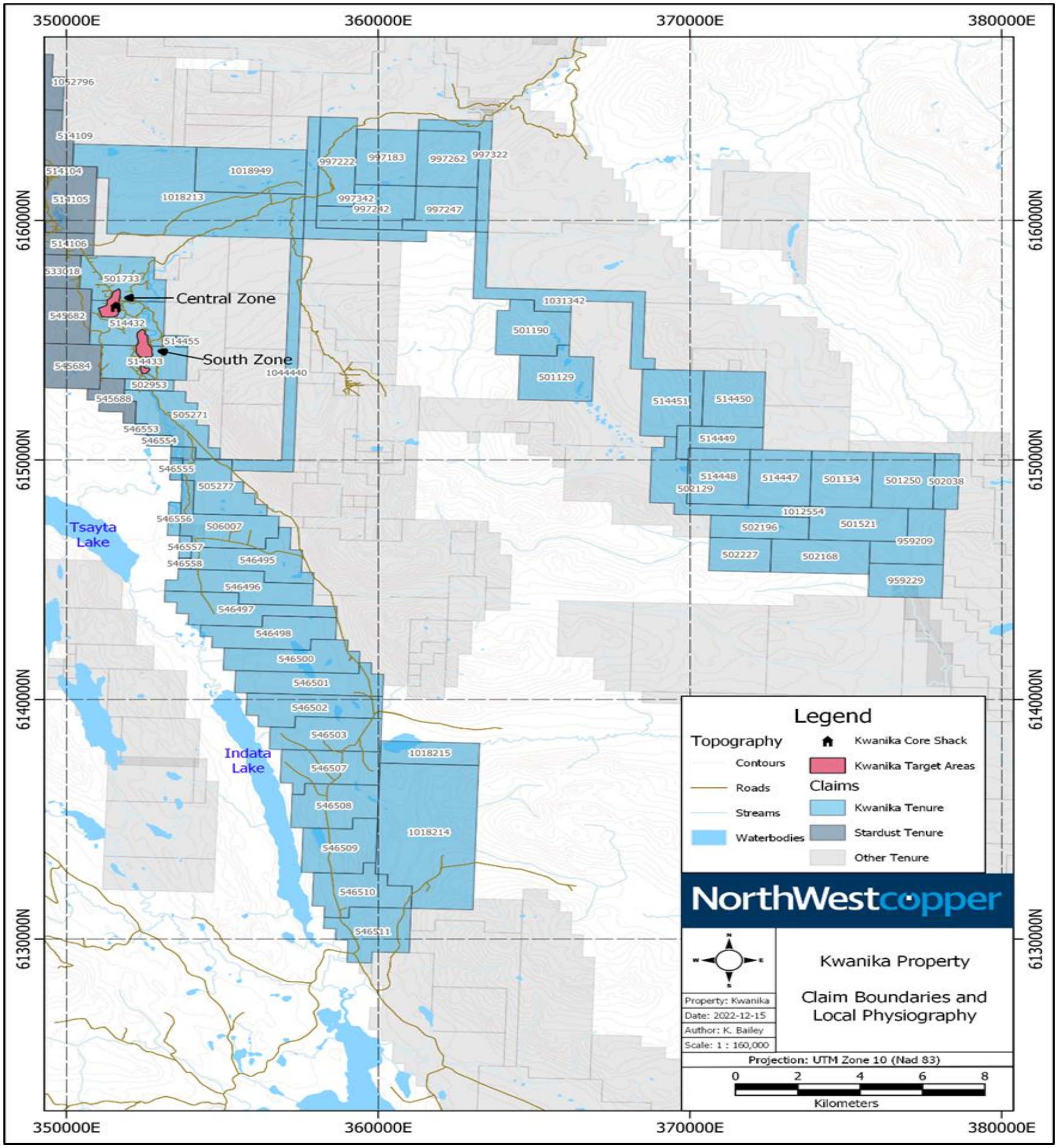
4.1.8 Comment

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

4.2 Stardust

The Stardust property is located around 150 km north of Fort St. James in the Omineca Mining Division of north-central British Columbia on NTS 93N/11W at latitude 55° 34' N (Northing 6160175) and 125° 25' W (Easting 347850), UTM Zone 10, NAD 83 (Figure 4-2).

Figure 4-2: Kwanika Claim Map



Source: NorthWest Copper, 2022

Figure 4-3: Stardust Property Location Map

Source: NorthWest Copper, 2022.

4.2.1 Tenure History

Pursuant to agreements dated July 15, 1989, and February 21, 1992, Alpha Gold acquired interest in 77 mineral claims known as the Lustdust property, Omineca Mining Division. In 2003, Alpha acquired the retained 5% net profits interest and the 2% NSR royalties. In 2003, net smelter returns were purchased for these claims. Also, during 2003, an additional 8 two-post claims overlying the historic Takla Bralorne Mercury Mine were acquired by purchase. In June 2005 all these claim holdings were converted to 11 “cell” claims.

In 2006, six additional “cell” claims were acquired bringing the total to 17 contiguous claims covering an area of 8,561 ha (Figure 4-4). In 2011 an additional 3 claims were acquired bringing the total area to 9,583 ha. “Cell” claims are geographic blocks with boundaries defined by a computer mapping system. No fractions or ownership disputes are possible with this type of claim.

In August 2013, Alpha Gold was renamed ALQ Gold Corp.

In June of 2016, Lorraine Copper acquired the property from Alpha Gold. The completion of the sale was announced in a news release dated September 26, 2016. It was stated that “Lorraine Copper purchased a 100% interest in the Lustdust property by (i) issuing ALQ 5.5 million LLC common shares and (ii) paying ALQ $50,000 in cash. After acquisition, Lorraine Copper decided to change the property name to ‘Stardust’.

In September 2017, 1124245 B.C. Ltd. (subsequently renamed “Sun Metals Corp.”) was granted an option to acquire a 100% interest in the property subject to certain royalties and terms. Sun Metals fulfilled the 2017 expenditure requirement by completing an exploration program by year end.

In April 2019, Sun Metals acquired all outstanding shares of Lorraine Copper and thereby achieved a 100% interest in the Stardust project.

In March 2021, Sun Metals and Serengeti announced the completion of a merger and a name change to NorthWest Copper Corp.

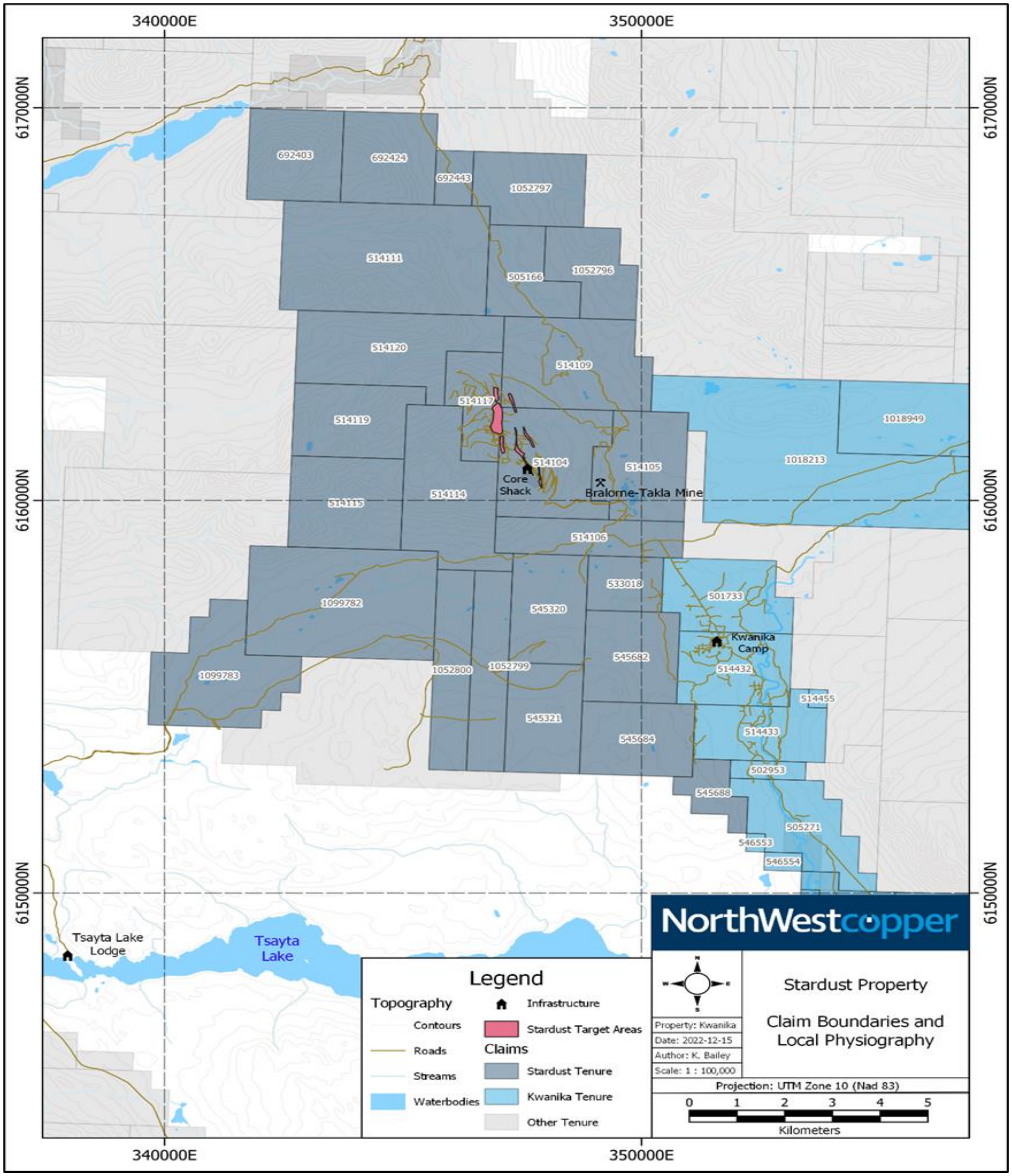
4.2.2 Mineral Tenure

NorthWest Copper owns a 100% interest in the Stardust project. The claims are registered to Tsayta, a wholly owned subsidiary of NorthWest Copper. The Stardust project encompasses 26 mineral claims covering 12,932.39 ha. Claim details are presented in Table 4-2 and Figure 4-4. A single small claim in the centre of the property covers the site of a historic mining drift into the Number 1 Vein Zone that is excluded from the project claims. Claim numbers 1099782 and 1099783 have a minimum annual spend required to retain the claims. These claims lie away from the deposit area.

Table 4-2: Mineral Tenure Information for the Stardust Project

Title Number	Claim Name	Map Number	Issue Date	Good to Date	Status	Area (ha)
505166	Alpha 1	093N	2005/JAN/29	2032/DEC/15	GOOD	347.159
514104		093N	2005/JUN/07	2032/DEC/15	GOOD	603.621
514105		093N	2005/JUN/07	2032/DEC/15	GOOD	493.88
514106		093N	2005/JUN/07	2032/DEC/15	GOOD	365.99
514109		093N	2005/JUN/07	2032/DEC/15	GOOD	694.665
514111		093N	2005/JUN/07	2032/DEC/15	GOOD	1205.807
514114		093N	2005/JUN/08	2032/DEC/15	GOOD	695.24
514115		093N	2005/JUN/08	2032/DEC/15	GOOD	548.9
514117		093N	2005/JUN/08	2032/DEC/15	GOOD	274.284
514119		093N	2005/JUN/08	2032/DEC/15	GOOD	457.193
514120		093N	2005/JUN/08	2032/DEC/15	GOOD	712.906
533018	ALPHA 2	093N	2006/APR/25	2032/DEC/15	GOOD	219.652
545320	LUSTDUST	093N	2006/NOV/13	2032/DEC/15	GOOD	439.3722
545321	LUSTDUST	093N	2006/NOV/13	2032/DEC/15	GOOD	439.653
545682	NAT 1	093N	2006/NOV/22	2032/DEC/15	GOOD	457.8047
545684	NAT 2	093N	2006/NOV/22	2032/DEC/15	GOOD	439.7042
545688	NAT 3	093N	2006/NOV/22	2032/DEC/15	GOOD	164.9228
692403	UTM2	093N	2010/JAN/01	2032/DEC/15	GOOD	456.4748
692424	UTM3	093N	2010/JAN/01	2032/DEC/15	GOOD	456.4704
692443	UTM4	093N	2010/JAN/01	2032/DEC/15	GOOD	109.5661
1052796	KW2	093N	2017/JUN/28	2032/DEC/15	GOOD	347.1348
1052797	KWN	093N	2017/JUN/28	2032/DEC/15	GOOD	420.0218
1052799	WESTSIDE 1	093N	2017/JUN/28	2032/DEC/15	GOOD	402.9199
1052800	WESTSIDE 2	093N	2017/JUN/28	2032/DEC/15	GOOD	402.9211
1099782	KWANIKA CREEK E	093N	2022/DEC/09	2023/DEC/09	GOOD	1098.4474
1099783	KWANIKA CREEK W	093N	2022/DEC/09	2023/DEC/09	GOOD	677.6792
					Total:	**12932.3894**

Figure 4-4: Stardust Claim Boundaries and Local Physiography



Source: NorthWest Copper, 2022.

4.2.3 Surface Rights

Surface rights over the Stardust property are owned by the Crown and administered by the Government of B.C. and would be available for any eventual mining operation. The ownership of other rights (Aboriginal, placer, timber, water, grazing, trapping, outfitting, etc.) affecting the property were not investigated by the author.

4.2.4 Agreements

NorthWest Copper and its predecessors Sun Metals have worked closely with Takla on the Stardust project. On August 19, 2020, a new Exploration Agreement was announced between Sun Metals (now NorthWest Copper) and Takla. The new Exploration Agreement replaced an expired agreement and was valid through to December 31, 2021. NorthWest Copper and Takla agreed to work using the terms of the previous agreement for the 2022 field season. NorthWest Copper is working with Takla and hopes to have a new exploration agreement in 2023 and the future. The previous agreement respects Aboriginal title, rights, and interests, and continues to recognize Takla's stewardship role in environmental and wildlife monitoring.

4.2.5 Royalties

The property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

4.2.6 Permitting Considerations

NorthWest Copper has an exploration permit issued by the B.C. Ministry of Energy and Mines and Low Carbon Innovation authorizing mineral exploration for the Stardust project. The permit is good until December 31, 2023, with the option to extend at the discretion of the B.C. Ministry of Energy and Mines and Low Carbon Innovation.

4.2.7 Environmental Considerations

The historical Bralorne Takla Mercury Mine is located within the property boundaries. This historical mine site is under the jurisdiction of the Crown Contaminated Sites Program.

The Crown Contaminated Sites Program (CCSP) in the Ministry of Forests, Lands, Natural Resource Operations and Rural Development manages contaminated sites on Crown land for which there is no existing responsible party. These are typically historical abandoned mine sites that make up a small fraction of the contaminated sites on Crown land. CCSP is not involved with contaminated sites on Crown land where there are specified parties responsible for the contamination.

Full remediation and cleanup programs were completed on this site through CCSP in 2018. At this point, only ongoing monitoring through CCSP and their contractors is required. NorthWest Copper is not involved with or responsible for any of the ongoing monitoring programs.

4.2.8 Comment

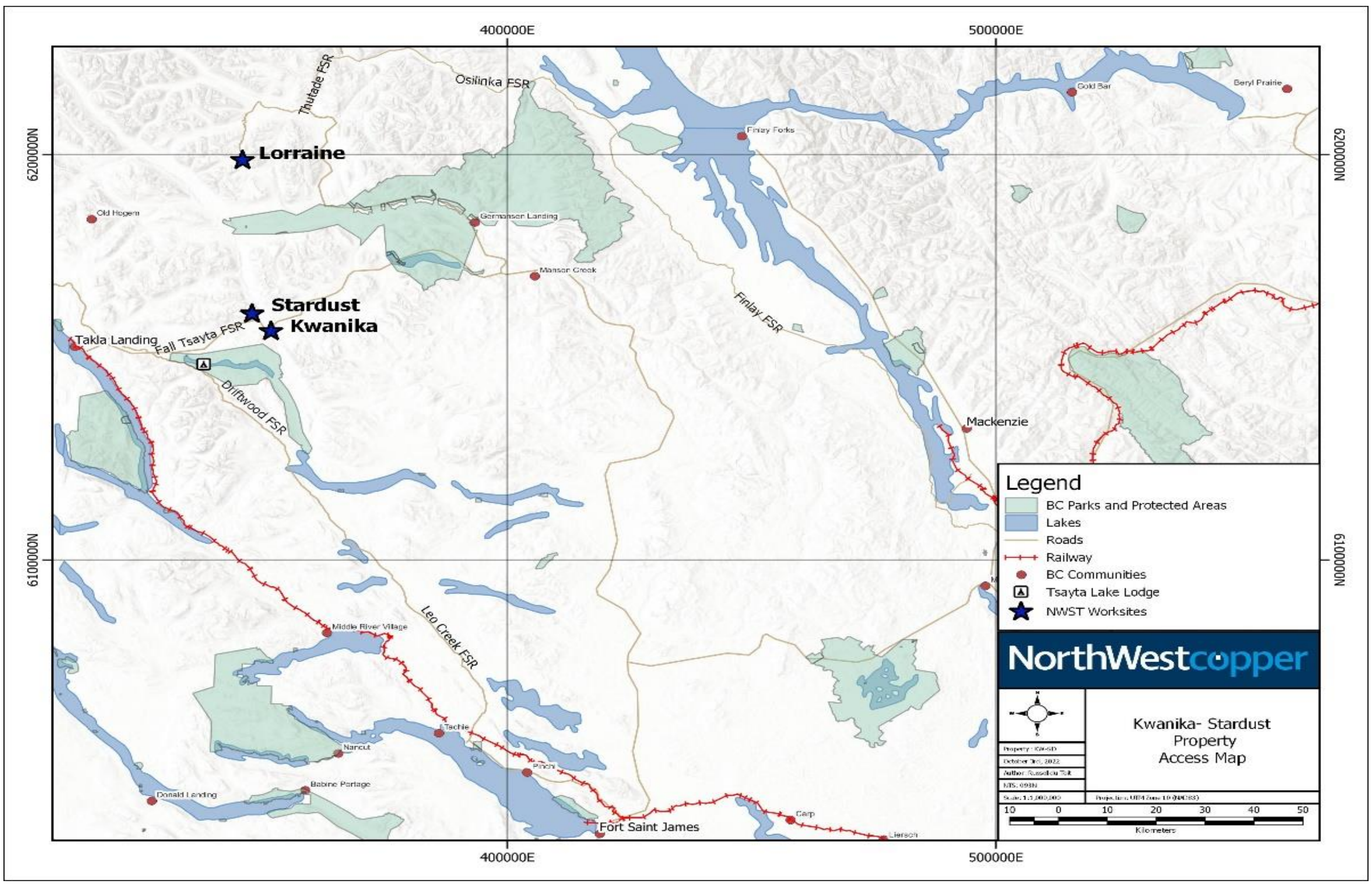
To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5 ACESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The project is located around 140 km northeast of Fort St. James in north-central British Columbia. The project is accessible by road from Highway 16 at Fort St. James by travelling 30 km along a paved road towards Tachie Lake, then north for 68 km along the all-weather Leo Creek FSR, 54 km along the Driftwood FSR, and 29 km along the Fall-Tsayta logging road (Figure 5-1). The Fall-Tsayta FSR is suitable for passage of four-wheel-drive vehicles in all seasons (pending snow removal) and has been maintained seasonally by NorthWest Copper and its predecessor companies since the fall of 2006. The road is generally snow-free from May to October. Driving time from Fort St. James is around three hours in good road conditions. The Properties are also accessible by float plane, about a one-hour trip to Tsayta Lake from either Prince George or Smithers, followed by a half-hour drive to the site.

Figure 5-1: Project Access Map



Source: NorthWest Copper, 2023

5.2 Climate

The climate is cool and moderate with warm, moist summers and cool winters. The average temperature for this area (based on data from Fort St. James) is 3.1°C, with a peak average monthly temperature of 21.9°C in July and an average monthly low of -15.8°C in January. Winter temperatures are commonly below freezing and can fall as low as -30°C for short periods of time. The region receives an average of 295 mm of rainfall and 192 cm of snowfall annually, with 138 days per year where precipitation exceeds 0.2 mm. The project is commonly snow-covered from late October to May. The exploration season is typically July through October, but it is possible to drill year-round.

5.3 Local Infrastructure and Resources

The project is in close proximity to the communities of Prince George (population 74,000), Burns Lake (population 1,780), Houston (population 2,990) and Smithers (population 5,350). These established centres have provided the necessary supplies and services to operate past exploration programs. Prince George has a mineral resource sector economic base and is a five-hour drive from the Properties.

Smaller population centres closer to the project are Takla Landing, which is the main community, and Fort St. James, which is the main community for the Nak'azdli Whut'en First Nation.

Canadian National (CN) Railway Company maintains an active rail line to Fort St. James (around 200 km via road) that could potentially be used for concentrate transport. The Kemess power line lies around 80 km from the project site.

A temporary core logging tent, core cutting shack, an outhouse constructed in 2021 and historical buildings, including 1 office and 4 storage buildings, are the only usable structures on the Kwanika property. Similarly, there is a temporary core logging building, core cutting shack, office tent and outhouse on the Stardust property that was constructed in 2018. There are also several fishing lodges and guiding camps within the area, including the Tsayta Lake Lodge, accessible by a 4 km road extending south from the 7.5 km point on the Fall-Tsayta FSR, which was the operations-base for the exploration programs carried out in 2021 and 2022.

5.4 Physiography

The Kwanika property occupies a broad, till-blanketed valley which ranges in elevation from 900 m to 1,200 m. The local topography is gently to moderately sloping, with sparse bedrock exposure. The only observable rock outcrops on the property are along the meandering Kwanika Creek, where fluvial processes have locally eroded the till blanket.

Kwanika Creek lies east of the Pacific divide, draining southward into the Nation Lakes chain, and eventually into the Arctic Ocean. The property is moderately forested with spruce and lodgepole pine, broadleaf deciduous trees, and shrubs, such as alder, birch and aspen, and underlying lichen and mosses.

The Stardust property terrain is moderate, ranging in elevation from 1,000-1,525 m with little outcrop exposure. Lower elevations are covered by widely spaced lodgepole pine. At elevations above 1,200 m, forest cover consists of over-mature spruce and balsam with an undergrowth of white rhododendron. Despite fairly moist summers, many drainages are seasonal in nature with progressively diminished flows during the late summer and fall.

5.5 Comment

The accessibility, climate, local resources, and physiography of the Kwanika-Stardust project site sufficiently well understood to allow for mineral exploration and mineral resource estimation.

6 HISTORY

6.1 Kwanika

The first exploration on the Kwanika property occurred in the 1930s and 1940s following the discovery of mercury at Pinchi Lake. Initial exploration concentrated on prospecting for mercury mineralization along the Pinchi fault and for placer gold in Kwanika Creek.

Copper mineralization was first recognized along Kwanika Creek by prospectors Almond and Thurber in 1964. A. Hodgson and G. Bleiler were first to stake the property for Hogan Mines Ltd. (Hogan) in 1965. During that year, Hogan conducted a small X-ray drilling program (27.4 m) as well as a trenching and geochemical program (Macdonald, 1965; Buskas, Garrett & Morton, 1989).

The property was subsequently optioned to Canex Aerial Exploration Ltd. (Canex) in 1966 (Pentland, 1966; Sawyer 1969). Canex's work included geological, geochemical (sediment and water, parameters not defined) and magnetic/IP surveys on a 67.6 km cut grid, as well as drilling 11 diamond drillholes (856 m). Geophysics identified an IP anomaly coincident with mineralized outcrops along Kwanika Creek. Drilling confirmed that this IP anomaly was caused by sulphide mineralization that comprised up to 5% of the rock mass. A second IP anomaly with a coincident 300 gamma magnetic response and a frequency effect of 3% was also identified to the west of Kwanika Creek. It remained untested as it was thought to be located in a sedimentary environment and within the Pinchi fault zone.

The Canex option was terminated, and the property was acquired by Great Plains Development Company of Canada (Great Plains) in 1969. Great Plains conducted a magnetic survey and drilled seven diamond drillholes (1,320 m) to test the previously identified IP and magnetic low anomalies (Sawyer, 1969; Buskas, Garrett & Morton, 1989). The drilling program outlined an area about 490 m by 300 m of copper mineralization grading around 0.20% Cu. No gold analysis was performed, and molybdenum was analyzed only in selected sections.

In 1972, Bow River Resources Ltd. (Bow River) mapped the property and drilled six percussion holes for a total of 549 m (Buskas, Garrett & Morton, 1989).

Pechiney Developments Ltd. (Pechiney) optioned the property in 1973 and conducted a 64.4 km grid IP and resistivity survey (Hallof & Goudie, 1973). When the results were interpreted with previous drillhole data, it was determined that the best copper grades corresponded to anomalies with frequency effects over 3% and resistivities over 100 ohm-m. In 1974, Pechiney conducted a 30-hole, 2,993 m percussion drilling program (Guelpa, 1974); however, assay results for this work are not available.

In 1981, Placer Developments Ltd. conducted a geochemical survey further south which consisted of 35 soil samples and 16 rock samples (Bulmer, 1981). Soil samples were collected from a grid with 100 m sampling interval and a line spacing of 200 m. Rock samples were collected from outcrops on the soil grid as well as along Kwanika Creek. The survey identified anomalous copper (up to 2,520 ppm), molybdenum (up to 730 ppm) and mercury (up to 90 ppb) within cataclastized granite along Kwanika Creek, near the Pinchi fault.

In 1983, Aume Resources Ltd. conducted a geochemical survey at the northern end of the Kwanika property to investigate the gold content of mercury mineralization associated with the Pinchi fault (Culbert, 1983). The survey consisted of 43 soil samples, 37 stream sediment samples and 12 rock samples, which were collected during line traverses and included samples collected outside the property boundaries. Assay results supported the high concentration of mercury associated with the Pinchi fault (up to 6,400 ppb), although Au and Ag values were not anomalous.

In 1986, Daren Resources Ltd. conducted a geochemical survey in the northwest corner of the Kwanika property, which included work on the northwestern and western periphery of the property (Christoffersen, 1986). The regional survey consisted of 96 soil samples, 14 silt samples, and 15 rock samples. The results obtained from this survey confirmed previously identified low order gold, silver, and arsenic anomalies, with the best sample grading 275 ppb Au, 58 ppm As, and 1.1 ppm Ag.

In 1989, W. Halleran staked the Swan property, located in the northern portion of the Kwanika claims at 55°30'N, 125°19'W (Carpenter, 1999), on ground previously abandoned by Bow River. Halleran was able to demonstrate the association of gold with the copper mineralization and subsequently optioned the property to Eastfield Resources Ltd. (Eastfield) (Buskas, Garrett & Morton, 1989). During 1989, Eastfield conducted an extensive exploration program which consisted of cutting 22.6 km of grid lines, a geochemical survey (55 soils at 50 m intervals, 143 stream sediments on Kwanika Creek tributaries and 162 rock samples), and a 23.3 km IP survey. Work conducted during this period also consisted of geological mapping, prospecting, and resampling historical core. Results from the geochemical survey indicated that the highest and most consistent copper-gold anomalies were restricted to the North copper zone (values up to 9,462 ppm Cu and up to 1,227 ppb Au). A comprehensive analysis of the geophysical chargeability results in conjunction with geochemical, drillhole and geological surveying data yielded six targets for future exploration which extended throughout the property. Furthermore, it was determined that the best copper mineralization was not always associated with the strongest sulphide mineralization, suggesting that significant copper mineralization may be associated with less intense IP anomalies.

Eastfield also carried out a small drilling program in 1991 consisting of 4 diamond drillholes totalling 549 m (Morton, 1991). The program intended to test geophysical targets to the north and west of the Pechiney 1974 percussion holes. The drilling program failed to identify new zones of significant mineralization.

Discovery Consultants (Discovery) re-staked the Swan property and continued exploration in 1995 with a limited heavy mineral stream sediment (two samples) and rock (15 samples) geochemical program (Carpenter, 1996). The heavy mineral stream sediment samples from the west edge of the property yielded anomalous gold values of 3,180 ppb and 4,580 ppb, whereas the rock samples had values up to 73 ppb Au and 2,607 ppm Cu. In 1999, Discovery obtained an additional three heavy mineral stream sediment samples from the east side of the property which yielded anomalous gold values of 7,450 ppb and 1,730 ppb (Carpenter, 1999).

A historical Mineral Resource estimate for what is currently referred to as the South Zone deposit was produced in 1976. The estimate stated a Mineral Resource of 36 Mt grading 0.20% Cu (Pilcher and McDougall, 1976). No mention was made of the source of this estimate or how the estimate was completed. Serengeti was able to obtain a similar result using the same dataset and a polygonal method. The estimate is only referenced herein for historical completeness, and it should not be relied upon.

No further work was performed on the property until Serengeti acquired it in 2004. Subsequent work carried out is described in Section 9, Exploration.

6.1.1 Production

There has been no production from the Kwanika property.

6.2 Stardust

The Stardust area was first staked in 1944 when the No. 1 Zone (Takla Silver Veins) was discovered near the southern end of the property. Since that time numerous operators have investigated the property and immediately surrounding area and a number of mineralized zones have been identified.

The Bralorne Takla Mercury mine was in operation from November 1943 to September 1944 when mining ceased. During nine months of operation, 59,914 kg of mercury were recovered from 10,206 t of milled Mineralized material from the two largest orebodies (Geological Survey of Canada Memoir 252, page 157).

Bralorne Mines Ltd. explored the property from 1952 to 1954. In 1960 Bralorne again acquired the property and from 1960 to 1962 carried out further work (drilling and trenching) in a joint venture with Noranda Exploration Company, Ltd., and Canex Aerial Exploration Ltd. A limited sampling program was also carried out by Bralorne alone in 1963.

The option held by Bralorne was transferred to Takla Silver Mines Ltd. which was organized in September 1964 to explore and develop the property. A new adit, bypassing the old one, was begun in 1964 and advanced to a total length of 229 m in 1965. Diamond drilling during 1965-1966 totalled 259 m underground and more than 762 m on surface. In July 1968, an agreement was reached with Anchor Mines Ltd. by which a new company, Anchor-Takla Mines Ltd., was incorporated for the purpose of performing joint venture work on the property. Additional ground was acquired in the A.G. 1-6, Ag 1-4, and Keno 1-8 claims. Diamond drilling during the fall of 1968 totalled 573 m in 17 holes underground, and 1,337 m in 13 holes on surface. The underground work was confined to the No. 1 zone. The company (Anchor-Takla) was dissolved in 1977.

In 1977, Granby located the K, L and M claims comprising 38 units to cover a large area with apparent mineral potential. The M claims adjoined Crown Granted Mineral Claims L.6181, 6184, 6186 and 6188 which formed part of the former Bralorne Takla Mercury Mine property. Pioneer Metals Corporation acquired 100% interest in the property early in 1985 and followed with some geological work in 1986.

The Air claim was added to the property in late 1978, and in 1979 three fractions and 52 metric claim units were located.

In 1978 Granby cut 67 km of grid line, carried out a soil geochemical survey and mapped the property at a scale of 1:5,000. In 1979 a Pulse EM survey was conducted by Glen White Geophysics Ltd., followed by a diamond drill program later in the year.

In 1989 Alpha acquired the property and in 1991 completed 988 m of drilling in 11 holes on Zone 3. They followed in 1992 with 30 diamond drillholes totalling 1,520 m on Zone 4B. In 1993, Alpha Gold completed a further 24 diamond drillholes on Zone 4B and purchased 8 two-post claims which overlie the historical Bralorne Takla mine. A total of four drillholes were collared in the mine area but only three were successfully completed. An extensive soil geochemical survey was also conducted in the mine area.

Teck Exploration Ltd., under option from Alpha, drilled 16 holes totalling 3,063 m in 1997. Drilling targeted the manto and skarn styles of mineralization that were traced by trenching in 1996. Alpha completed 1,103 m in a 14-hole diamond drilling program in 1998 that targeted Zones 1, 2 and 3. In 1999, Alpha completed an 18-hole, 3,045 m drilling program that accomplished two objectives. It extended the strike length of the skarn zone 1,000 m further to the north (hole LD99-06 intersected 5.2 m grading 8.3% copper) and provided very encouraging information on a previously untested 400 m gap between the most southerly skarn holes and most northerly exposures of manto mineralization. In 2000, Alpha drilled 4,680 m of diamond drilling in 29 holes. Most of the drillholes targeted prospective skarn zones, although the company did test areas further west for potential porphyry mineralization. In 2001, Alpha drilled 5,610 m in 18 holes on the Canyon Creek Skarn (CCS) Zone and peripheral targets.

Alpha drilled 19 NQ bore holes totalling 7,790 m between July 8 and September 6, 2002, on the CCS deposit. An additional 42 NQ holes totalling 7,908 m, were completed in 2003 and 32 holes totalling 6,010 m in 2004. Most of the drilling was on the CCS deposit.

In 2005, Alpha Gold drilled 5,153 m in 16 diamond drillholes. Drilling a coincident gold-arsenic soil geochemistry anomaly 300 m east of the CCS deposit resulted in the discovery of the East zone. In 2005 Alpha also conducted a broad, grid-based soil sampling and bedrock mapping program that covered not only the Dream Creek area north of the Canyon skarn zone but also part of the Pinchi fault system at the former Bralorne Takla mercury mine.

In 2005, a mineral resource estimate was prepared by Snowden reportedly in conformance with the requirements set out in the standards defined by NI 43-101 (Palmer & Hanson, 2005). However, this report was never filed publicly on SEDAR.

In 2006, diamond drilling extended the sinuous geometry of the Canyon Creek copper skarn system both downdip and to the south. Alpha drilled 6,855 m in 31 NQ diamond drillholes and 3,054 m in 24 rotary holes. Trenching of a gold soil anomaly southeast of the Canyon Creek zone discovered the GD zone. The company completed a reverse circulation drilling program in an area surrounding the historic Bralorne-Takla mercury mine to evaluate gold soil anomalies outlined in 2005.

In 2007, Alpha Gold completed 50 line-km of soil geochemistry and IP, mapping, and 11 boreholes totalling about 2,757 m. In 2008, Alpha completed about 2,400 m of drilling on untested targets on the southern portion of the property.

In 2009, Alpha completed 6,367 m of core drilling in 17 holes, mainly targeting the CCS zone. In 2010, Alpha drilled 14 holes (3,987 m) in the Canyon Creek and Canyon Creek Extension zones.

In 2012, Aurora Geoscience was engaged by Alpha Gold to carry out a data evaluation and report on project potential.

No work was carried out between 2012 and the time the Stardust project was acquired by Lorraine Copper.

The 2017 exploration project carried out by Lorraine Copper, included a geochemical survey, IP, and magnetometer surveys and a 3-hole diamond drill program totalling 344 m.

Work by Sun Metals between 2018 and 2020 is described in Sections 9 and 10.

A summary of work performed by the various parties is shown in Table 6-1 the following page. Note that what is listed in the table is not necessarily a complete compilation of exploration work on the property, as some original reports on exploration activities could not be located.

6.2.1 Mineral Resource Estimate

Two previous NI 43-101 compliant Mineral Resource Estimates were carried out on the project in 2010 and 2018 (Simpson, 2010 and Simpson, 2018). These estimates are no longer considered current due to the additional exploration work carried out on the project since 2017.

6.2.2 Production

There has been no production from the Stardust property.

Table 6-1: Stardust Exploration History

Year	Company	Work	Drillholes	Drilling (m)	Mag (km)	VLF EM (km)	IP (km)	Soil Samp.	Rock Samp.
1944		zone 1 discovery; claim staking							
1945	McKee Gp/Leta	trenching; drilling		0					
1952	Bralorne Mines	trenching; drilling							
1954	Bralorne Mines	drilling		0					
1958	Totem Minerals	mag, geochem.							
1960	Noranda Canex	rock cuts; trenching; test pits							
1963	Bralorne Mines	sampling							
1964	Takla Silver Mines	drifting							
1966	Takla Silver Mines	underground drilling	5	500					
1968	Takla Silver Mines	surf/underg drilling; bulk sample							
1968	Rip Van Mining	High-grade (HG) soil geochem; trenching							
1978	Granby Mining	geol; geochem; pulse EM						910	
1979	Zapata Granby	EM							
1979	Zapata Granby	drilling	3	615					
1980	Noranda (Zapata)	drilling	2	299					
1981	Noranda (Zapata)	geochem; drilling; EM; geol.	6	854		26.15		722	
1983	Golden Porphyrite	geol.; geochem.						521	56
1984	Golden Porphyrite	geochem.						66	3
1984	Equinox Res.	geochem.						62	14
1984	Golden Porphyrite	geochem.							9
1986	Welcome North	sampling							
1986	Pioneer Metals	geol.							
1986	Equinox Res.	geochem.						96	15
1989	Eastfield Res.	geochem; mag; vlfem; geol.			21	21		570	
1989	Eastfield Res.	geochem.; geol.					0.45	29	25
1991	Alpha Gold	drilling	11	988					
1991	Alpha Gold	resubmission of above AR?							
1992	Alpha Gold	drilling; trenching; geophys.	30	1,520		12.5			23
1993	Alpha Gold	summary report	24	2,041					
1996	Teck/Alpha	geochem; geol.; trenching						513	259
1997	Teck/Alpha	geochem; drilling	16	3,063					
1998	Teck/Alpha	drilling	14	1,105					
1999	Alpha Gold	drilling; geol.	18	3,050					
2000	Alpha Gold	drilling; geol., mag	29	4,680					
2001	Alpha Gold	drilling; geol.	18	5,609					
2002	Alpha Gold	drilling	19	7,790					
2003	Alpha Gold	drilling	42	7,908				695	

Year	Company	Work	Drillholes	Drilling (m)	Mag (km)	VLF EM (km)	IP (km)	Soil Samp.	Rock Samp.
2004	Alpha Gold	drilling; geochem.	21	6,010				724	
2005	Alpha Gold	drilling; geochem, geol.	17	5,153				587	
2005	Alpha Gold	resource comp. CCSZ							
2006	Alpha Gold	drilling; geochem, trenching	56	9,909					7
2007	Alpha Gold	airmag/em; drilling	34	8,898					
2008	Amark	airmag			74				
2008	Alpha Gold	drilling	5	2,140					
2009	Alpha Gold	drilling; trenching	17	6,367					
2009	Alpha Gold	resource estimate							
2010	Alpha Gold	drilling; geol.	14	3,987				12	28
2010	Alpha Gold	resource comp. CCSZ							
2011	Alpha Gold	geol; geochem						285	
2011	Alpha Gold	airmag/ZTEM			330.6	330.6			
2012	Alpha Gold	Evaluation							
2017	Lorraine Copper	drilling; geochem; IP/Mag	3	344	28.1		28.1	744	45
2018	Sun Metals	drilling; geochem; VTEM	23	6,877		1128		2804	73
2019	Sun Metals	drilling; geophysics	28	14,024					
2020	Sun Metals	drilling; geophysics	16	11,975					
		Totals	**471**	**115,708**	**453.7**	**1518.25**		**9340**	**557**

7 GEOLOGICAL SETTING AND MINERALIZATION

The following description of the regional tectonic and structural setting of Kwanika is largely taken from Osatenko *et al.*, 2020.

7.1 Regional Geology

The Kwanika porphyry deposits are located at the western margin of the Quesnel terrane (Quesnellia) as shown in Figure 7-1. Quesnellia is a Late Paleozoic to Early Jurassic Island arc that hosts numerous alkalic and calc-alkalic porphyry Cu ± Au ± Mo ± Ag deposits, and which extends north from the British Columbia-Washington State border for more than 1,000 km (Logan and Mihalynuk, 2014). This terrane formed adjacent to ancestral North America in response to eastward-dipping subduction of the Tethyan oceanic Cache Creek terrane (Mortimer, 1987).

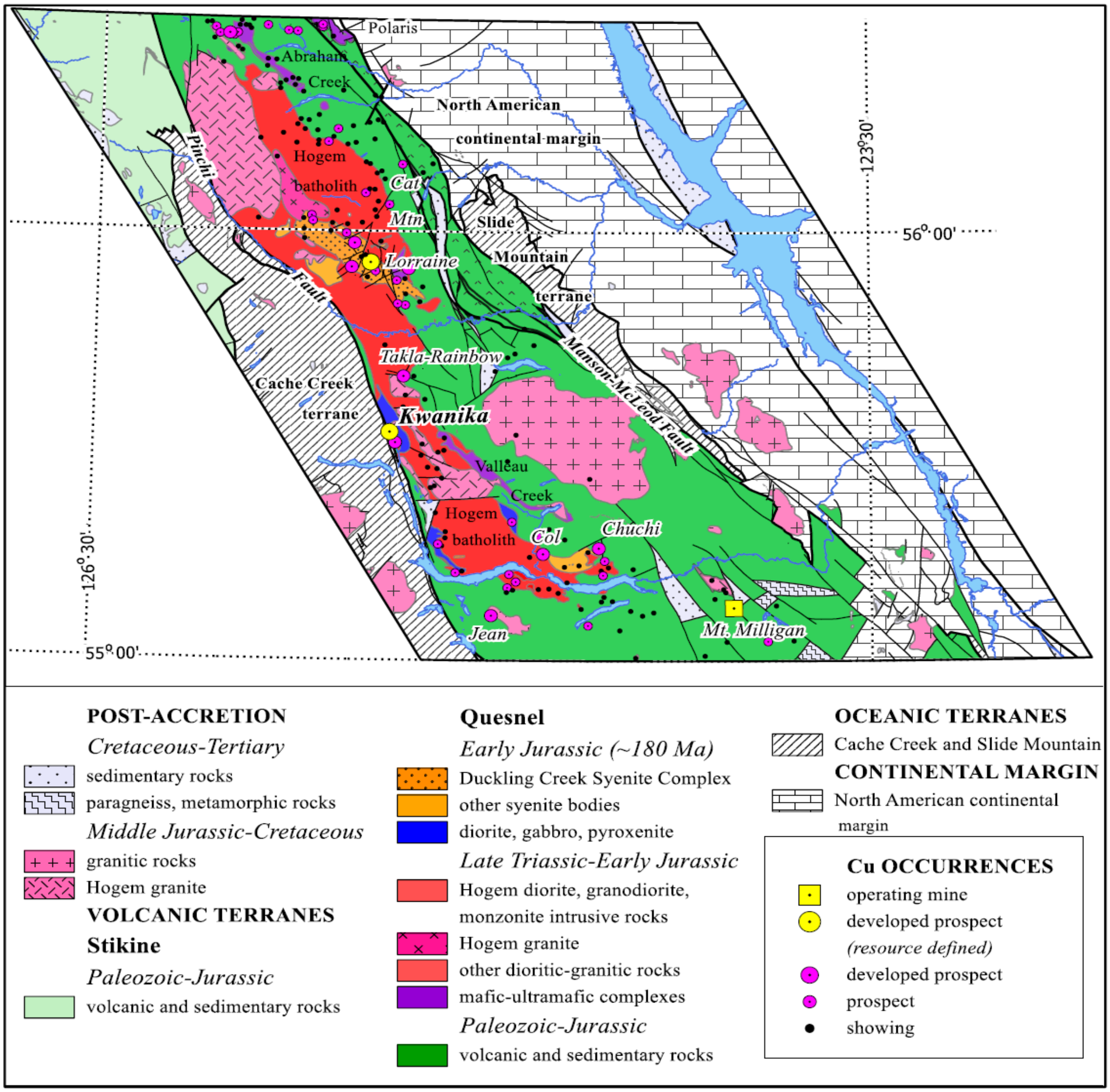
The Quesnel terrane is mainly composed of Late Triassic to Early Jurassic island arc-derived volcanic, sedimentary, and plutonic rocks of the Nicola (southern British Columbia) and Takla (northern British Columbia) Groups that developed above an eastward-dipping subduction zone (Mortimer, 1987; Monger and Price, 2000). In southern British Columbia, eastward migration of Mesozoic arc magmatism led to the growth of three temporally distinct, north-trending plutonic belts characterized by rocks of Late Triassic age in the west, through Late Triassic to Early Jurassic, and finally to Early Jurassic in the east. Associated with these plutonic belts are distinctive episodes of calc-alkalic Cu-Mo, alkalic Cu-Au, and calc-alkalic Cu-Mo porphyry metallogenic events responsible for the formation of the Highland Valley, New Afton/Ajax and Brenda deposits respectively (Logan and Mihalynuk, 2014). This trend continues to the north with the calc-alkalic Gibraltar deposit on the west, the alkalic Mount Polley deposit in the centre and the calc-alkalic Woodjam Southeast deposit on the east.

At the present latitude of Kwanika, the Quesnel terrane is separated from the Proterozoic and Paleozoic carbonate and siliciclastic rocks of the Cassiar terrane, part of the ancestral North American continental margin to the east, by Late Paleozoic chert, argillite and basalt of the Slide Mountain terrane which represents remnants of a Late Paleozoic marginal basin (Ferri, 1997). To the west, the Quesnel terrane is faulted against Paleozoic to Mesozoic chert, argillite, limestone, and basalt of the Cache Creek terrane. The Manson-McLeod fault system separates the Quesnel terrane from the Slide Mountain terrane to the east, and the Pinchi fault separates the Quesnel terrane from Cache Creek terrane to the west. These terrane bounding structures record protracted and complex displacement histories culminating in prominent dextral strike-slip motion during the Cretaceous to early Tertiary (Gabrielse, 1985).

In the Kwanika area the Quesnel terrane consists of Late Paleozoic island arc volcanic and sedimentary rocks of the Lay Range assemblage (Ferri, 1997), Late Triassic volcanic and sedimentary rocks of the Takla Group (Monger, 1977) and Early Jurassic volcanic and sedimentary rocks of the Chuchi Lake and Twin Creek successions (Nelson and Bellefontaine, 1996). These rocks are cut by several suites of Late Triassic, Early Jurassic, and Middle Jurassic plutons of the Hogem Suite (Garnett; 1978; Woodsworth et al., 1991). Unlike the discrete plutonic belts in southern British Columbia, these magmatic episodes are spatially transposed onto one another resulting in a 200 km by 25 km north-northwest-trending composite plutonic body called the Hogem batholith (Logan et al., 2010). Most phases of the Hogem batholith contain Cu-Au mineralization. However, significant mineralization is related to small, satellite intrusions (for example at Cat Mountain, a Late Triassic monzonite; at Lorraine, the Early Jurassic Duckling Creek Syenite Complex; at Mt. Milligan, the Early Jurassic MBX and Southern Star stocks; at Col; at Chuchi Lake syenite body; and at Kwanika in an Early Jurassic quartz monzonite). The Hogem batholith includes both calc-alkalic and alkalic suites as well as Alaskan-type ultramafic-mafic intrusions (Garnett, 1978; Mortensen et al., 1995; Nixon et al., 1997; Nixon and Peatfield, 2003; Jago et al., 2014).

Progressive subduction of Cache Creek led to amalgamation of the Stikine and Quesnel terranes, separated by relics of Cache Creek oceanic basin and formation of the Intermontane arc complex (Mihalynuk et al., 1999). Final terrane accretion to the North American margin occurred by the mid-Jurassic (Nixon et al., 1997; Nelson et al., 2013). Post-accretion Cretaceous granites host local uneconomic occurrences of Cu and Mo (Garnett, 1978). However, these intrusions were generated and emplaced well after east-dipping subduction beneath the Quesnel terrane had ceased.

Figure 7-1: Regional Geology

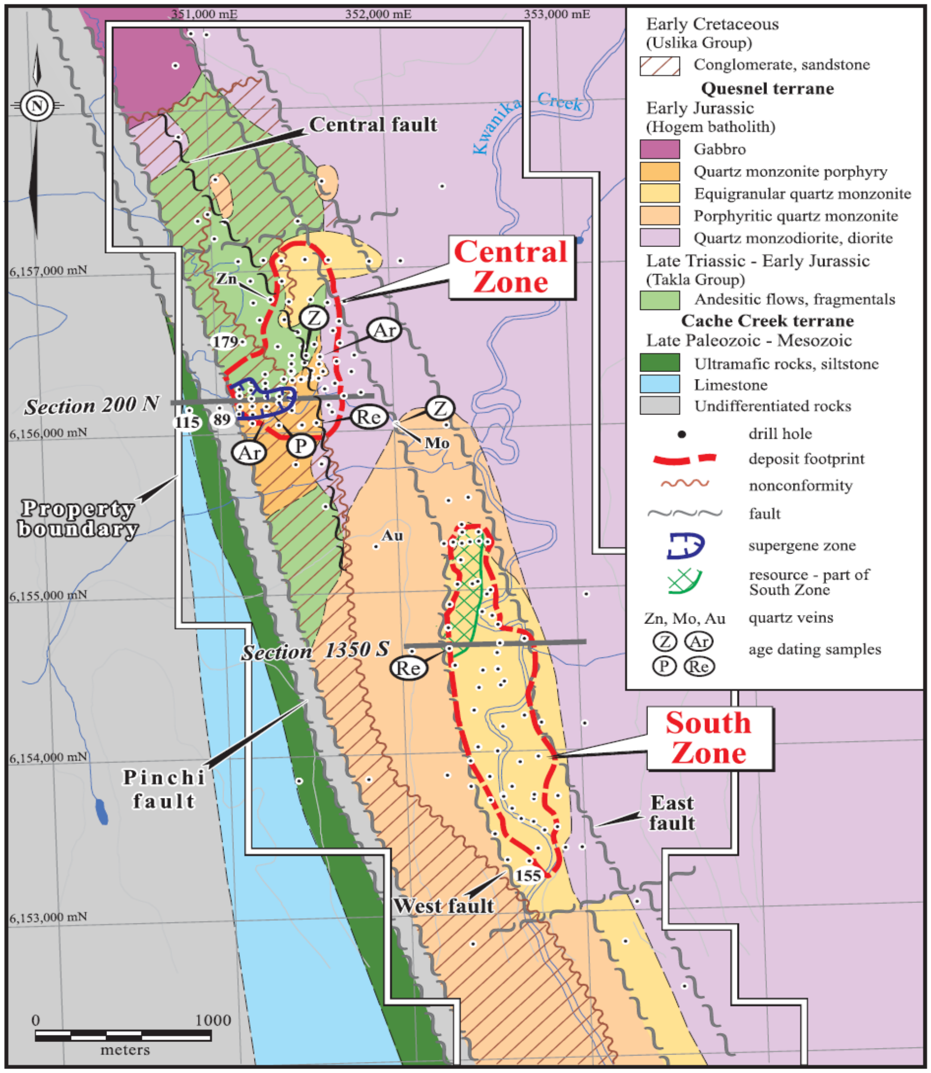


Source: Osatenko et al., 2020

7.2 Kwanika Property Geology

The Kwanika project consists of two mineralized areas: the Central Zone and the South Zone. The geology and alteration in each zone are described separately. Figure 7-2 shows the interpreted geology around the Central and South Zones.

Figure 7-2: Kwanika Local Property Geology



Source: Osatenko et al., 2020

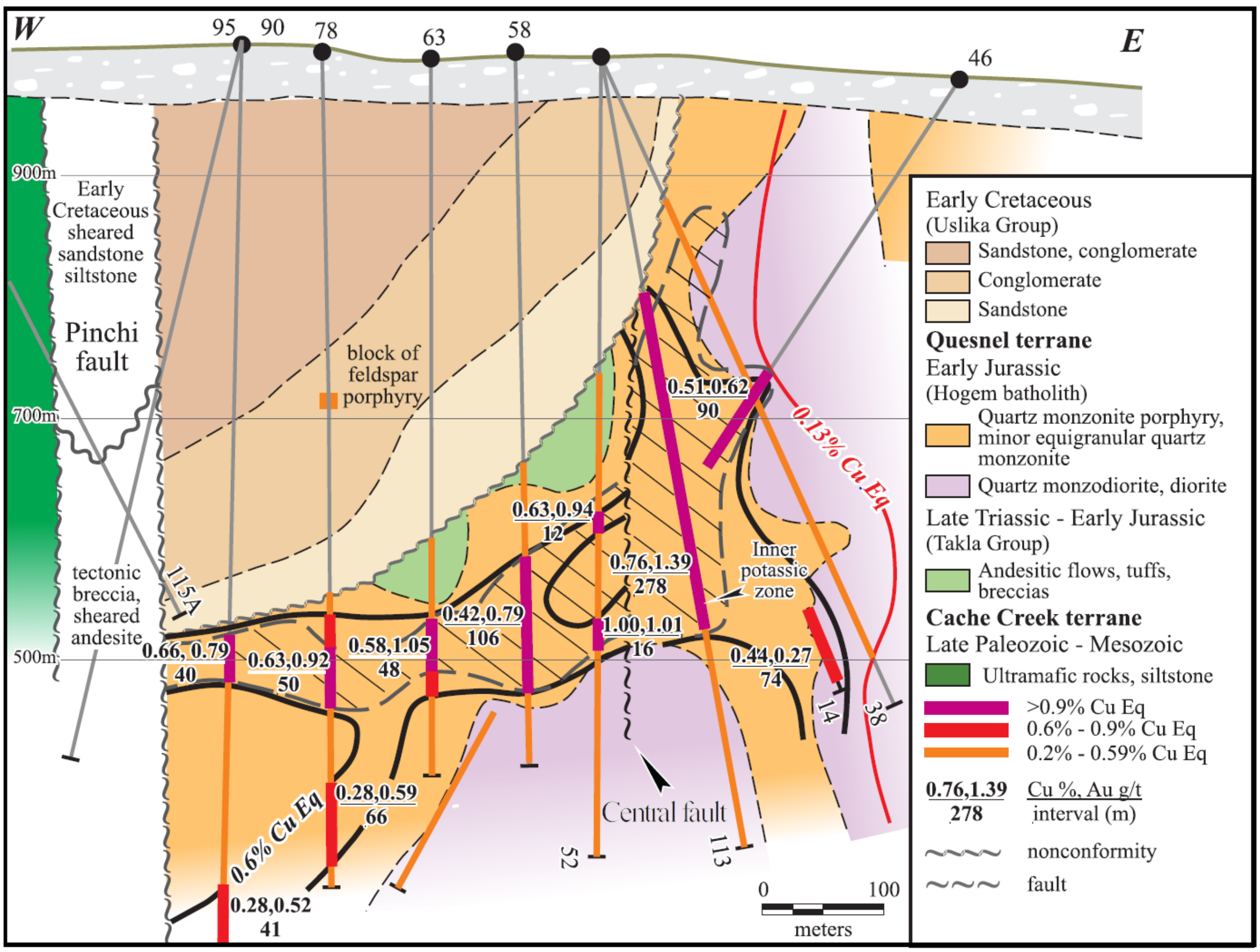
Mineralization in the Central and South Zones at Kwanika occurs in the Quesnel Terrane, immediately east of the Pinchi fault which places it against the Cache Creek Terrane and is associated with intrusive phases of the Hogem batholith. The mineralization is mostly covered by glacial sediments that average 25 m to 35 m in thickness and, thus, bedrock geology is interpreted from drill core and the few outcrops along Kwanika Creek in the South Zone.

7.2.1 Central Zone

The Central Zone is 1,400 m long by 400 m wide and extends more than 700 m below surface where it is open to depth on many drill sections. The western part of the zone is downfaulted by the Central fault and then cut off by the Pinchi fault further west. Mineralization is mainly hosted by a shallow to steeply-dipping plug and dyke complex of quartz monzonite porphyry. The quartz monzonite porphyry intruded Takla Group andesitic rocks in the west and pre-mineral quartz monzodiorite-diorite intrusions in the east. These rocks are, in part, non-conformably overlain in the west by Early Cretaceous sedimentary rocks preserved within a west-dipping half-graben.

East of the Central Fault, the Central Zone comprises a vertical to steeply-dipping quartz monzonite porphyry with similarly steeply-dipping grade contours and alteration shells. West of the Central Fault, the quartz monzonite porphyry, grade contours and alteration shell contacts are shallowly to moderately dipping to the west (Figure 7-3).

Figure 7-3: Geology of the Central Zone Shown on East-West Drill Section 200N Looking North



Source: Osatenko et al., 2020

The Cache Creek Terrane rocks are the oldest rocks near the Central Zone and occur west of the Pinchi fault. This area is covered but 3 km to the south along the projection of the Pinchi fault Garnett (1978) mapped limestone and gabbro/serpentinite. Inclined drilling west of the Central Zone encountered the vertically dipping Pinchi fault zone in two drillholes. It is about 80 m wide and contains strongly sheared sandstone and siltstone, clay-altered hematitic tectonic breccias and sheared andesite believed to be Takla Group. An ultramafic rock, intersected in the upper part of drillhole K-115, is part of the Cache Creek Terrane and marks the western boundary of the Pinchi fault. It is sheared and fine to medium-grained with 50% olivine, 25% feldspar and 25% pyroxene.

The oldest rocks east of the Pinchi fault consist of dark green andesite of the Takla Group that are mostly fine-grained flows and tuffs with local flow breccias. Andesites host mineralization only adjacent to contacts with quartz monzonite porphyry, and typically have lower Cu and Au grades than mineralization within the porphyry. The quartz monzodiorite-diorite body is the oldest and largest intrusive phase in the Central Zone area. It lies to the east and below the quartz monzonite porphyry and is intruded by quartz monzonite dykes. Quartz monzodiorites are pale gray to greenish grey, medium-grained and equigranular, whereas diorites are black and range from microcrystalline to medium-grained. Both are composed primarily of plagioclase and hornblende with local coarse aggregates of magnetite and lesser amounts of biotite, K-feldspar and quartz. This unit is an important host to mineralization.

Andesitic volcanic and quartz monzodiorite-diorite rocks are intruded by two phases of quartz monzonite, a porphyritic variety in the Central Zone and an equigranular variety in the South Zone. The quartz monzonite porphyry hosts the highest-grade mineralization, and its porphyritic texture is best recognized on the less altered eastern edge of the deposit. In areas where the potassic alteration is strongest, plagioclase phenocrysts have a corroded appearance and look like grains of rice. The porphyries are typically cream to pale orange and contain 40% to 50% plagioclase phenocrysts (<1 mm to 2 mm long) in a fine-grained matrix of K-feldspar, quartz, biotite and hornblende with accessory magnetite, rutile, zircon, and apatite.

The equigranular quartz monzonite is a medium-grained, commonly pinkish rock composed of plagioclase, K-feldspar, quartz, biotite and hornblende with accessory magnetite, rutile, titanite and apatite.

Late-mineral to post-mineral dykes are the youngest intrusive rocks and include feldspar porphyry, aphanitic dacite, biotite-hornblende diorite, biotite-pyroxenite and Tertiary andesite. These dykes are in sharp to locally faulted contacts with most units and are most common in the Central Zone. They are interpreted to be sub-vertical to steeply west-dipping to the east of the Central fault but to the west they dip more shallowly. Most of these dykes have a true thickness of less than 2 m.

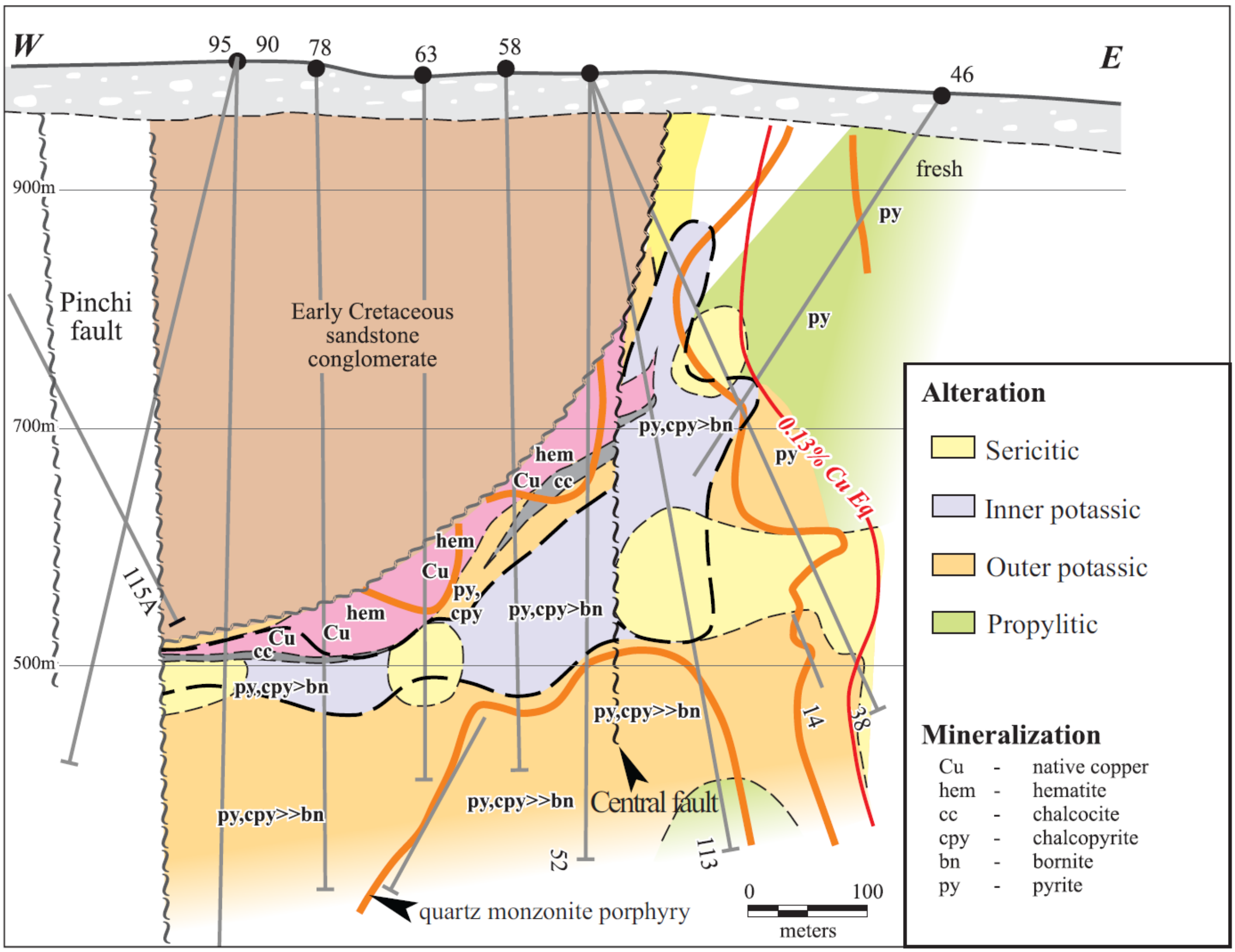
Quesnel Terrane intrusive and volcanic rocks east of the Pinchi fault have been eroded, down-dropped and overlain by Early Cretaceous clastic sedimentary rocks preserved in a half-graben that covers the western part of the Central Zone and preserves an Early Cretaceous or older supergene blanket. The sedimentary basin extends about 1.5 km north of the Central Zone, and more than 4 km to the south, where it tapers to a thickness of 20 m. These rocks dip moderately to the west and attain a maximum thickness of 435 m adjacent to the Pinchi fault. The sedimentary rocks are thickly bedded and consist of a thin hematitic basal breccia, a polymictic conglomerate with clasts of sandstone, siltstone, and unidentified volcanic and intrusive rocks, and an upper unit of mixed sandstone and conglomerate.

Palynological analysis of pyritic siltstone from drillhole K-55 contained spores, gymnosperm pollen (no angiosperm pollen) and terrestrial plant material (Sweet, 2009) indicative of deposition in a non-marine environment during the Valanginian to Early Albian of the Early Cretaceous. Similarly aged, fault-bounded sedimentary rocks in the region are correlated with the Uslika Group (Ferri et al., 2001).

7.2.1.1 Central Zone Alteration and Mineralization

Hydrothermal alteration in the Central Zone comprises an inner potassic core surrounded by an outer potassic shell that yields to a peripheral propylitic zone, all of which are variably overprinted by patchy sericite alteration (Figure 7-4). The inner potassic zone consists of creamy to pale pink secondary K-feldspar with minor albite, whereas the outer potassic zone comprises pink to red secondary K-feldspar and lesser biotite, cut by minor biotite, tourmaline, gypsum/anhydrite and magnetite veinlets. Both potassic zones contain small patches of less altered rock characterized by sericite-altered plagioclase and chlorite-altered biotite and hornblende. Rare, narrow, dyke-like bodies of hydrothermal breccia occur within 150 m of the bedrock surface in the central part of the zone and contain rotated and rounded fragments of highly silicified quartz monzonite (<1 cm to >6 cm long) in a matrix of quartz, pyrite, chalcopyrite, and tourmaline.

Figure 7-4: Distribution of Alteration and Mineralization Types in the Central Zone Shown on East-West Section 200N Looking North



Source: Osatenko et al., 2020

The inner potassic core, which is closely associated with the quartz monzonite porphyry, is texturally destructive and grades over short distances to the outer potassic shell. Veins and fracture fillings of creamy to pale pink secondary K-feldspar cut the outer potassic zone. The inner potassic zone hosts the highest Cu and Au grades (0.86% Cu equivalent), and also has the highest Au:(Au+Cu) ratio (0.60, defined as Au(g/t):(Au(g/t)+Cu (%)) in comparison to the 0.46 average ratio for the entire Central Zone). Mineralization consists of disseminated pyrite (1% to 2%) and chalcopyrite > bornite hosted by a stockwork of 5% to 15% (locally to >50%) quartz veinlets and, to a lesser extent, disseminated in altered wall rocks. Mineralized veinlets are typically 0.5 cm to 1 m in width and occur in at least two generations. Bornite typically replaces chalcopyrite.

The outer potassic shell is considerably more extensive than the inner potassic zone. The inner part is intense, texturally destructive secondary K-feldspar alteration that grades outward to a zone of quartz veinlets with secondary K-feldspar envelopes. Hydrothermal biotite is more common below the 450 m reduced level (RL) and in the peripheral part of this zone. This alteration is developed in quartz monzonite porphyry, quartz monzodiorite-diorite, and andesite. As in the inner potassic zone, pyrite (1% to 4%) and chalcopyrite >> bornite occurs in a stockwork of 1% to 5% quartz veinlets as well as disseminated through the altered wall rocks. Very rare molybdenite is present in quartz veinlets with pyrite, but it was not observed cutting the Au-Cu mineralization. Native Au was not observed in drill core but was recognized in a polished section where it occurs as inclusions in bornite.

Propylitic alteration occurs in a 100 m to 300 m wide zone around the potassic zones. It is also present deep beneath the quartz monzonite porphyry body in the southern part of the Central Zone. This alteration comprises chlorite and epidote, with minor sericite and calcite, 1% to 2% pyrite and trace chalcopyrite. Propylitic alteration affects all rock types in the Central Zone but is most strongly developed in andesite and diorite.

Mineralization in the Central Zone is overprinted by pale green sericite alteration that is most strongly developed in irregular-shaped zones. The sericite alteration did not introduce any copper. Post-mineral iron and magnesium carbonate veinlets (siderite, magnesite and/or ankerite) and calcite veinlets cut the mineralized quartz veinlets.

7.2.1.2 Supergene Mineralization

A paleo-supergene zone on the western side of the Central Zone is preserved immediately beneath the Early Cretaceous sedimentary rocks. It varies in thickness from 3 m to 70 m, due in part to the influence of local structures, and extends 200 m north-south and for up to 400 m east-west. The supergene profile consists of a typically narrow upper oxide zone of strongly hematitic rocks with disseminations and thin veinlets of native copper that transitions to an underlying sulphide zone of chalcocite and minor covellite. The chalcocite is commonly associated with fine-grained, creamy illite or sericite, and the secondary sulphides occur in fractures, in the matrix of quartz breccias, and replacing hypogene pyrite, chalcopyrite and bornite.

7.2.2 South Zone

The South Zone is 2,200 m long by about 330 m wide, and locally extends more than 600 m below the surface. The highest copper grades occur in a steeply-dipping, 800 m long tabular body in the northwest part of the Zone, with an upper part extending to the east. The South Zone is ovoid in plan and is confined to a northerly trending corridor bounded by the West and East faults. The West fault zone widens from 3 m to 5 m near the surface to a 75 m wide crushed zone at depth, and dips steeply to the west. The east side of the South Zone is not well-delineated, due to limited drilling, but the steeply-dipping East fault can be recognized by resistivity and chargeability highs and was confirmed in hole K-10-155 by a 2 m intersection within a broad zone of sericite alteration.

The pre-mineral quartz monzodiorite-diorite intrusions that occupy the eastern portion of the Central Zone also occur immediately east of the East fault in the South Zone. This intrusion is cut by porphyritic quartz monzonite and equigranular quartz monzonite. The porphyritic quartz monzonite is composed of plagioclase, K-feldspar, quartz phenocrysts, hornblende and biotite with accessory magnetite, apatite, titanite and rutile.

Mineralization in the South Zone is mostly hosted by an equigranular quartz monzonite intrusion. Re-Os isotopic dates on molybdenite indicate that mineralization in the South Zone may be up to three million years younger than mineralization in the Central Zone, although the intrusions in the two zones return similar U-Pb ages on zircon. The equigranular quartz monzonite is medium-grained, and contains plagioclase, K-feldspar, quartz, hornblende and biotite with accessory magnetite, apatite, titanite and zircon. Only minor quartz monzonite porphyry and quartz monzodiorite are present in the South Zone; these porphyries are similar to those in the Central Zone and are probably narrow dykes. Most petrographic samples indicate an episode of strong brittle deformation manifested by crackle breccias (fragmentation but no rotation).

The South Zone intrusive rocks are cut by sericite-altered and K-feldspar-altered quartz monzonite dykes thought to be late-mineral in age and post-mineral dykes of primarily andesite. These dykes display sharp to locally faulted contacts with the altered quartz monzonite and typically have a true thickness of less than 2 m.

7.2.2.1 South Zone Alteration and Mineralization

Potassic alteration, mainly in the form of red to orange secondary K-feldspar, is widespread throughout the South Zone. It occurs commonly as pervasive, often texturally destructive, secondary K-feldspar flooding. Secondary K-feldspar also occurs in envelopes around rare quartz veinlets and fractures. The potassic zone has been brecciated and replaced by zones of fine-grained quartz. Sericite alteration occurs as fine to coarse patches replacing feldspars. Overprinting the potassic and quartz alteration zones are irregularly shaped zones of an iron-rich assemblage of chlorite, quartz, and pyrite with minor hydrothermal biotite. This alteration is typically texturally destructive. A minor, poorly defined zone of propylitic alteration surrounds the potassic zone and is composed of epidote and hematite occurring as fracture infills, chlorite replacement of hornblende and biotite, carbonate veining and late-stage quartz veinlets.

Fine to medium-sized grains of pyrite, chalcopyrite, and minor molybdenite occur along micro-fractures and as disseminations within the zones of fine-grained quartz that replace potassically-altered quartz monzonite. These sulphides also occur as disseminations in the iron-rich alteration assemblage. Disseminated mineralization is cut by rare quartz veinlets that contain pyrite and chalcopyrite with selvages of fine-grained molybdenite. Molybdenite also occurs in fractures. Pyrite, typically 2.5% to 3.5%, is ubiquitous in the mineralized zone and occurs as fine to coarse grained, anhedral to euhedral crystals. Very minor sphalerite, galena, hypogene chalcocite, tetrahedrite, bornite and enargite occur mainly in the northern half of the South Zone. Petrography indicates that chalcopyrite, sphalerite, hypogene chalcocite and enargite are contemporaneous.

There is no significant supergene mineralization in the South Zone. Mineralization was high-standing during the Early Cretaceous and eroded into the sedimentary basin to the west, as demonstrated by large blocks of altered/mineralized rocks within the Early Cretaceous sedimentary rocks.

7.2.3 Kwanika Property Structural Geology

The Central, West, East and Pinchi faults (Figure 7-2) are the four major NNW-oriented faults on the property and have been identified from drillhole and geophysical data. These faults are interpreted to have a variable amount of dip slip (generally west side down) and strike-slip movement.

The most persistent structural feature within the Central Zone is the steeply west-dipping, NNW-oriented Central Fault. It separates an eastern domain characterized by steep lithological contacts and grade distribution from a western domain characterized by sub-horizontal orientation of rock types and grade boundaries. This same fault locally over-steepens the unconformity at the eastern limit of the Cretaceous basin. The sub-vertical Pinchi fault truncates the Central Zone between 500 and 750 metres below surface, and rocks of the Cache Creek Terrane occur west of the fault.

In the South Zone, the West fault is the most persistent feature and extends to, and an unknown distance beyond, the north property limit. The East fault has also been traced to the north, where it may form the eastern limit of a sub-basin of Cretaceous sedimentary rocks, located near the north end of the property.

7.3 Stardust Property Geology

Very strongly deformed Pennsylvanian to Permian Cache Creek units underlie about 80% of the Stardust property. These units form upright to overturned asymmetrical west-dipping folds that plunge north at shallow angles. These folds are subparallel to the north-northwest trending Pinchi Fault that lies along the eastern property boundary. Stratigraphy most commonly strikes at 320 to 330° and only 10% of strikes do not fall within a west-northwest to north-northeast range. Strike commonly varies over tens of metres, giving bedding a sinuous, rather than linear, appearance. Dips are generally vertical to moderate westerly but do exhibit large variance due to the intense deformation in the area. Other workers (Ash and MacDonald, 1993) have suggested that the intense deformation of the Cache Creek Group adjacent to the Pinchi Fault in the Stuart Lake area 140 km to the south-southeast makes normal stratigraphic interpretation nearly impossible. Likewise, since units on the Stardust property have been thickened, thinned, pinched, faulted off, and/or juxtaposed by the intense deformation that they have undergone during continental accretion and the ensuing intrusive phases, interpreting stratigraphy can be a difficult task. That said, previous reports (Ledwon and Beck, 2009 and 2010) indicate conformability to the stratigraphic column at the local scale, making stratigraphic interpretation feasible. There are slivers and lenses of units throughout the Stardust property when outcrop is present. Sedimentary slump structures have been observed at Stardust, but limited outcrop makes finding them difficult (Ledwon, 2011).

Much of the mapped regions of the property contains an assortment of intrusions that cut carbonate rocks interbedded with graphitic, siliceous, and calcareous phyllites, cherts, cherty argillites, and mafic flows. Intrusions are found throughout the property, except in the far north of the claims, where they may be concealed by overburden (Ledwon, 2011). Though most commonly dioritic to monzonitic, intrusions also range from felsic to tonalitic. A composite intrusive centre and linear dyke array known as the “Glover Porphyry” occurs in the central and north-central portions of the property. Though elsewhere dykes appear to be subparallel to stratigraphy rather than crosscutting it, here, intrusive body orientation is more northerly. Pervasiveness of biotite hornfels and skarn increases towards the stock (Evans, 1998) within the CCS Zone. Some of the intrusive phases contain significant amounts of magnetite and appear to be responsible for the large magnetic anomaly shown on published regional maps and in Alpha's 2000 ground-magnetics survey (Butler and Jarvis, 2000), Alpha’s 2008 airborne aeromagnetic survey (St-Hilaire, 2008), and Alpha’s 2011 Airborne AFMAG ZTEM geophysical survey (Legault et al., 2011), in the 2017 ground-magnetic survey (Scott, 2017) and in the 2018 airborne electromagnetic survey (Prikhodko et al., 2018).

Many of the unmineralized veins found on the property were seen to emanate from dykes and crosscut all other stratigraphy suggesting that non-Glover dykes may be the youngest rocks on the claims.

The majority of the mafic and andesitic volcanic rocks have been found in the north and western reaches of the claims. Moving eastward, a non-calcareous, often gradational package of argillite-phyllite-siliceous phyllite-chert dominates the centre of the property. This is followed by swaths of limestone closer to and to the west of the Pinchi Fault, and finally the Hogem Batholith to the east of the Pinchi Fault on the eastern edge of the claims. Linear dykes oriented subparallel to foliation can be found throughout the claims. The Glover Stock occurs in the central to north-central part of the property. East-west running creeks (Dream Creek, Canyon Creek) likely trace faults, and appear to have offset stratigraphy.

Newly acquired (in 2017) claims to the northeast of the property (1052797 and 1052796) are almost entirely to the east of the Pinchi Fault and, as indicated by 2005 BCGS regional mapping, are underlain by rocks of the Hogem Plutonic Suite. These claims, plus slivers of other claims to the east of the Pinchi, make up about 23 km^2 or roughly 20% of the property. Work previously completed by Serengeti at their Kwanika Creek property to the immediate east of these claims describes multi-phase monzonitic to dioritic Hogem Batholith intrusions within successions of andesitic Takla Volcanic Group rocks. Though geology at the Stardust property is likely similar, these claims were not mapped during the 2017 field season. Property geology is shown in Figure 7-5.

7.3.1 Supracrustal Rocks

Interpretations of primary stratigraphy are challenged by the strong regional deformation. In the area of extensive drilling of the 4b and CCS zones (Figure 7-5), however, several coherent rock panels may be described as follows:

Hanging wall assemblages to the Canyon Creek Skarn are dominated by a sequence of compositionally laminated, siliceous and/or argillaceous phyllites often with strong biotite compositional layers. These rocks are interpreted as ribbon cherts by British Columbia Geological Survey geologists. The argillaceous, clastic component, of these rocks may increase towards the skarn – calc silicate horizon, particularly to the south towards the 4b zone.

Skarn assemblages are developed in weakly compositionally layered limestones, in calcareous mafic tuffs, and rarely in siliceous phyllites.

Footwall assemblages to the Canyon Creek Skarn are dominated by rocks which are typically described as cherty argillites and/or cherts. Rocks in the footwall are similar to hangingwall rocks but qualitatively appear to have a higher proportion of quartz compositional layers and decreased biotite lamellae.

Stratigraphic units are more fully described below.

7.3.1.1 Limestone (LS)

Light to medium grey, sucrosic, recrystallized limestone, locally with weak stylolitic cleavages. These rocks bleach to off-white adjacent to skarn fronts. They may contain numerous internal horizons of both dark grey clastic beds and mafic tuffaceous horizons.

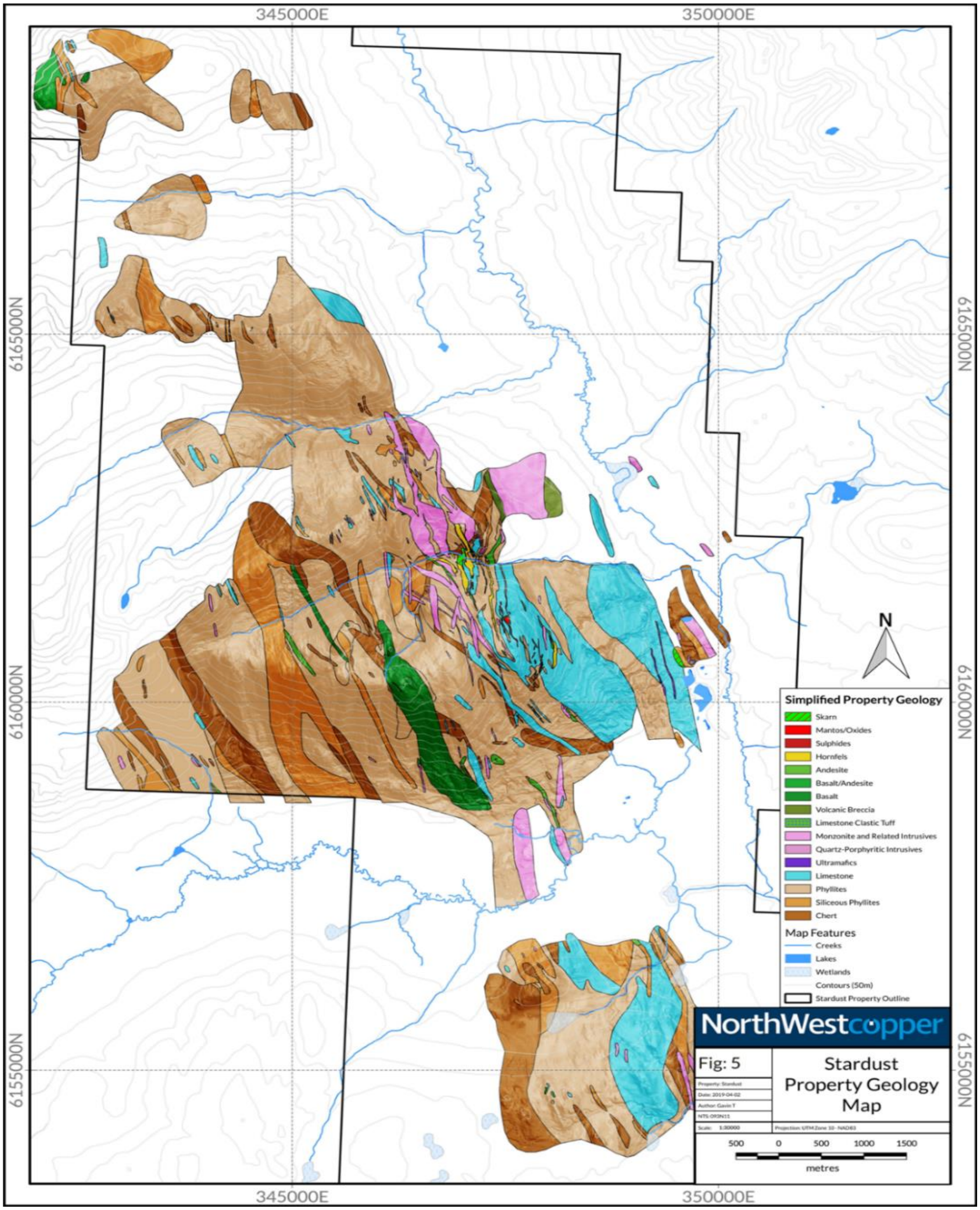
7.3.1.2 Calcareous Phyllite (CP)

Dark grey-brown, argillaceous interbeds are intercalated with thin, centimetre scale, calcareous lamellae.

7.3.1.3 Calcite Knot Limestone (Lcs)

Calcite knot limestones may contain either white cm scale calcite aggregates within a darker grey matrix, or they may be a gradational unit to mafic tuffs where 10-30% oval to cuspate calcite clasts are supported by a strongly calcareous, light to medium green matrix.

Figure 7-5: Stardust Property Geology



Source: NorthWest Copper, 2022

7.3.1.4 Siliceous Phyllite (SP)

These rocks are defined by compositional layers formed by alternating foliation parallel biotite ± lesser white micas, with quartz compositional layers. The protoliths of these rocks is interpreted, by many workers, as ribbon cherts.

7.3.1.5 Chert (C)

With an increase in quartz content, to greater than 75% rock volume, the rocks are logged as cherts. Minor increases in biotite compositional layers may shift these rocks into a phyllitic chert (PC) field.

7.3.1.6 Argillite (A)

Argillite is a composite unit that includes a wide range of fine-grained, essentially non-calcareous, carbonaceous, thinly bedded sedimentary rocks. It includes argillites (A), cherty argillites (CA), thinly bedded cherts, carbonaceous argillites (CA). Graphitic layers are common throughout. Locally, the thinly bedded units contain fine-grained, continuous pyrite or pyrrhotite layers that appear to be part of the original sediments. As with all supracrustal rocks, these units are strongly deformed.

7.3.1.7 Mafic Tuffs (MT)

Mafic tuffs are well-foliated and often well compositionally layered dark green, to green and white mottled rocks with highly chloritic and locally calcitic matrices. The chlorite is interpreted to result from alteration of mafic-intermediate tuffaceous materials. 1-30 cm limestone fragments are the dominant clasts, but fragments of intermediate and mafic volcanic rocks are also present. These rocks contain up to 2% finely disseminated pyrite and/or pyrrhotite and are geochemically anomalous for Pb, Zn, and Cu. Grading in limestone fragment size is common. Evans (1997, 1998) believed that there was only one mafic tuff unit and that it was a good marker bed. Previous fieldwork and core logging show that there are multiple mafic tuff units in the section, and they show enough lateral variation that their utility as marker beds may be limited.

7.3.2 Intrusive Rocks

Mineralization throughout the Stardust property shows a close association with the Glover Stock, a complex of porphyritic stocks and dykes ranging from diorite to monzonite to rhyodacite. Cu-Au skarn forms abundantly along stock and dyke contacts (and replaces these rocks) and Zn-Au-Pb-Ag-Cu replacement mineralization is locally well-developed along dyke margins at more distal locales. Overall, mineralization shows zonation relative to the inferred centre of the Glover Porphyry complex. Some compositional variations may be a consequence of alteration, but most differences reflect primary igneous variation. Intrusive rock units include:

7.3.2.1 Monzonite (M)

A medium-grained equigranular to weakly porphyritic rock composed of plagioclase>K-feldspar, abundant elongate hornblende, and euhedral biotite. Quartz is present, but in minor amounts. This unit crops out extensively as dikes throughout the southern and southwestern area, and the dikes seem to widen towards the 4b Zone. These dykes locally host replacement mineralization along their flanks.

7.3.2.2 Megacrystic Monzonite (Mp)

This intrusive phase is defined by the presence of very strongly plagioclase +/- quartz porphyritic monzonites. Contacts of these rocks with finer-grained phases may be gradational.

7.3.2.3 Quartz Monzonite (QM)

These rocks contain 10-15% free quartz as discrete, millimetre scale phenocrysts. The rock is also hornblende and biotite porphyritic and may be beginning to shift into a granodiorite field.

7.3.2.4 Diorite (D)

Diorites are fine to medium-grained, medium to dark gray-green and composed of plagioclase, biotite, and hornblende phenocrysts. Accessory magnetite is locally abundant. The phases are distinguished largely on the presence and the abundance of biotite and hornblende. This distinction can be difficult to make in the finer-grained units where potassic alteration has replaced the hornblendes with secondary biotite. Colour is determined by mafic phenocryst content and the degree of chloritic alteration.

7.3.2.5 Monzodiorite (MD)

A shift to increased percentages of fine-grained matrix plagioclase and a decrease in mafic phases, hornblende and biotite are the characteristics of this unit. Free quartz is not identified.

7.3.2.6 Felsic Dykes (Fd)

Felsic dykes occur across the property. These are weakly porphyritic felsic rocks with sparse to prominent 1-3 mm quartz and feldspar phenocrysts set in a sugary fine-grained matrix of quartz and feldspar. They are locally well flow-banded with banding generally parallel to their overall orientation. Felsic dykes are often pervasively argillic altered or silicified making them difficult to distinguish from altered fine-grained monzonite. Felsic dykes in the Number 1 Zone commonly have vein mineralization along one or both contacts.

7.3.2.7 Felsic Dykes (Fpd) Plagioclase Porphyritic

Distinctive elongate, sericitized feldspar phenocrysts are abundant within this rock matrix and may exceed 35% rock volume. The rock also contains 5-8% coarse quartz phenocrysts.

7.3.2.8 Mafic Dykes (Bd)

Medium to fine-grained, undifferentiated mafic dykes.

7.3.2.9 Ultramafics (UM)

Green to black, uralitically altered, ultramafic intrusions. In their unaltered state, the intrusions are likely pyroxenites. Elevated interstitial magnetite is common. Pyrrhotite is locally noted. The intrusions likely trace major strands of the Pinchi Fault. True brittle-ductile fabrics are common within these intrusions.

7.3.3 Stardust Structural Geology

Rocks underlying the Stardust property have experienced multiple deformational events. In the absence of geochronological data, definitive age relations between these events are difficult to establish. However, overall map patterns, rock fabrics and discordant rock fabrics in drill core suggest that at least two penetrative deformational processes, D1 and D2, have influenced the current map pattern.

The development of a pronounced planar S1 fabric, often coplanar to bedding and primary compositional layers, defines an early D1, deformational process. These fabrics are most likely axial planar to the tight to isoclinal, upright to west overturned, east-verging folds. The data of Ray et al. (2002) suggest these folds plunge around 40-50° to the north-northwest. The distribution of bedrock lithology has been profoundly influenced by this event.

The rotation of S1 fabrics is evidence for post D1 processes. Although S1 fabrics are clearly rotated, S2 penetrative foliations are weakly developed and may be measured in only very selective core and rock samples. Ray et al. (2002) suggest that D2 folds have similar orientations to D1 folds, but tend to be slightly more open, and have shallower 20° northwest plunges.

Regionally, folds in the Cache Creek assemblage are typically open (Schiarizza and McIntyre, 1999), but on the Stardust property folds are generally asymmetrical and overturned with short, shallow, west-dipping western limbs and long, steep, west-dipping eastern limbs. Locally they are isoclinal. Tight folding is likely due to buttressing against the Pinchi Fault, which is believed to have originally been a major thrust fault (Paterson, 1977). Where observed, these folds have a 10-60 degree N-NW plunge and minor axial plane shears are common. The noses of antiforms are structurally thickened and fractured zones that were favourable for manto mineralization (Evans, 1998; Megaw, 1999).

The entire property has a strong NW-trending grain reflecting bedding, tight asymmetric folding, and bedding plane faults. This structural fabric closely controls intrusive emplacement and most of the dykes of the Glover stock are strongly elongated along this N-NW structural grain. The most important, and consistent, fault structures demonstrated in drill core are roughly coplanar to bedding. Some of these faults have the appearance of early east-verging reverse faults, which are largely lithologically controlled and mostly identified in the immediate hangingwall to the Canyon Creek Skarn. These faults may be rotated into slightly steeper positions by latter extension faults.

The strongest and most strike discordant structural zone on the property is the structural zone and dyke system which hosts the Number 1 veins. This mineralized fault structure has a nearly north-south strike and moderate to steep west dip. In marked contrast, all structures, including lithology and major skarn bodies on the Stardust property have strike relationships which average 150° to 160° and steep westerly dips.

Compilation of the subsurface data with the surface geological plans suggests that right-stepping lithologic offsets, which occur both to the north and south of Canyon Creek, are related to fold vergence effects - an east-verging, right-stepping antiform - rather than a fault related offset.

Mapping of carbonates at property scale (Evans 1997; 1998) shows a wide outcrop band in the southern portion of the property that appears to decrease in width to the north, largely disappearing at Canyon Creek. This may be an artifact of limited outcrop exposures as integration of the subsurface information from drilling suggests the northern continuity of the most easterly limestone package may be significantly better than initially interpreted (Figure 7-3). The limestone is asymmetrically folded and plunges north at 15-20°.

7.3.4 Stardust Mineralization

Several styles of mineralization that are zonally related to each other are present on the property. From most proximal to most distal from the Glover Stock, they are:

- Molybdenum-Copper-Gold Porphyry consisting of quartz-K-spar, pyrite, molybdenite and/or chalcopyrite veinlets associated with potassic, sericitic, and propylitic alteration in intrusive rocks (Glover Stock).
- Multi-stage Garnet-Diopside skarn cut by Cu-Au-Ag-Zn bearing structures with surrounding dispersed Cu-Au mineralization (Canyon Creek Skarn).
- Structurally and stratigraphically controlled massive sulphide Zn, Au, Pb,
- Ag, Cu replacement bodies [CRD] (4b, 3, and 2 Zones) and their oxidized equivalents.
- Sulphosalt-rich veins (Zone 1) which follow faults and are strongly associated with fine-grained, linear, felsic dykes containing high values of Au, Ag, Pb, Zn, Sb and Mn.
- Mercury mineralization in limestone proximal to the Pinchi Fault.
- Sediment-hosted gold mineralization in limestone.

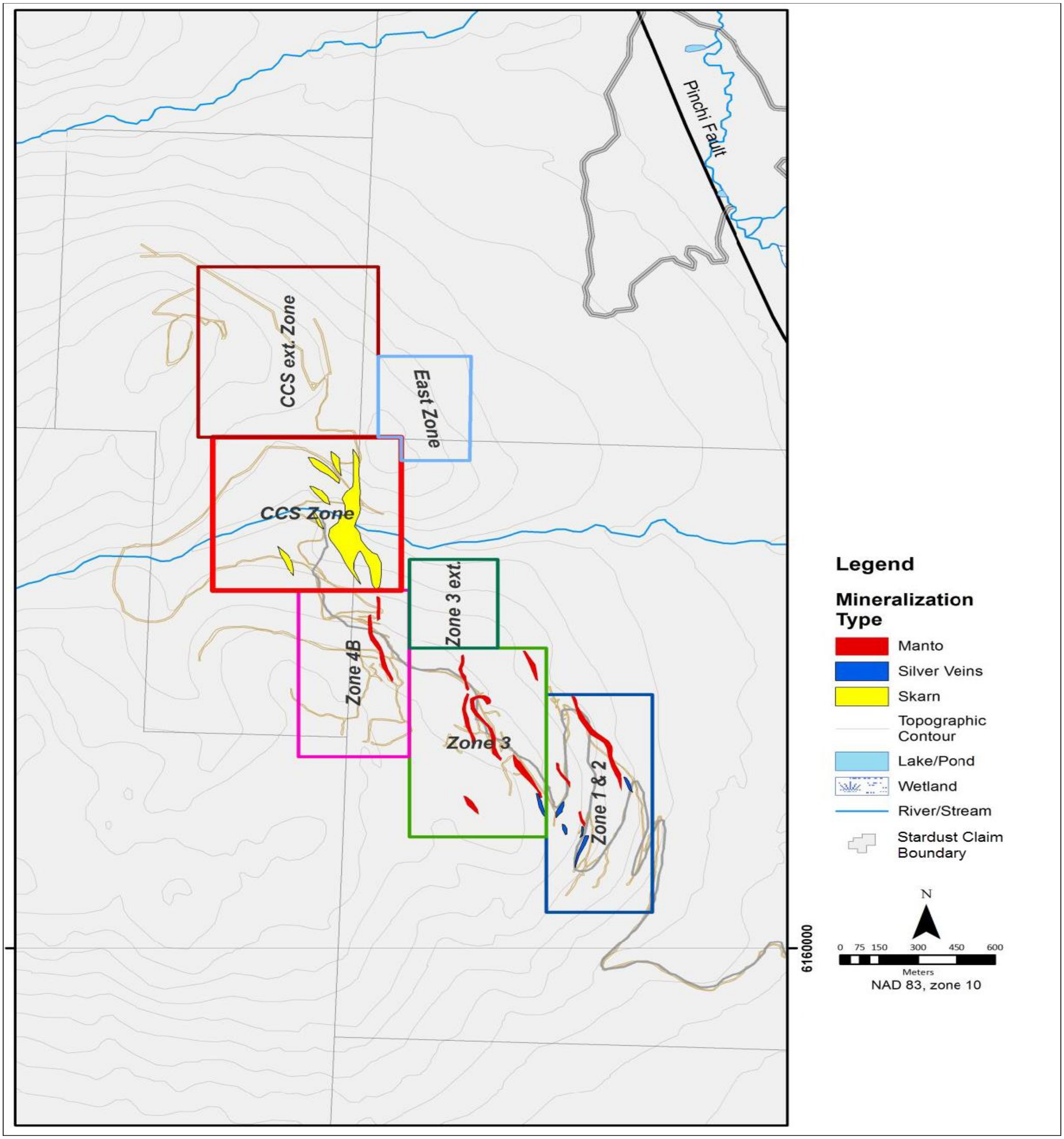
The location of the historic and current mineralized zones at Stardust are presented in Figure 7-6. Principle characteristics of the main mineralized zones are summarized below.

7.3.4.1 Zn-Pb-As-Sb Vein Zone: Number 1 Zone

The Number 1 Zone, located at the southern end of the property, was the site of the 1944 discovery of mineralization on the property. Here, the limestone and graphitic phyllites are cut by numerous monzonite and felsic dikes. Sulphosalt veins composed of nearly massive pyrite, sphalerite, galena, jamesonite, stibnite, arsenopyrite and freibergite with lesser open-space filling quartz and calcite occur both within the sedimentary rocks and along dike contacts. Dunne & Ray (2002) also report traces of very fine-grained calc-silicates in these bodies. Three separate veins have been recognized, all of which appear to dip steeply west. Felsic dikes are closely related to all three veins, but the veins do extend beyond the dikes in many places. The Number 1 Zone has the strongest structural control of any occurrence on the property. The presence of a regional antiformal crest is likely to be important to the development of significant mineralized zones as is the main fault structure. Argentiferous Manganese Oxide Mineralization (AMOM) occurs throughout the Number 1-Zone. AMOM is a typical distal alteration product in certain major CRD systems (Megaw, 1998) and the Number 1 Zone is strongly anomalous in Mn (Evans, 1997). Based on inclusion chemistry and mineralogic relationships, Dunne & Ray (2002) suggested that the mineralization in this zone might be related to high sulphidation-type veins. However, the alteration mineralogy and textures of quartz and other gangue minerals do not support the high sulphidation model for these veins.

The principal vein was explored by underground drifting and drilling in the 1945 and 1964-65 seasons. The three shoots (minimum 2 m true widths) above the adit level were reported to grade 3.6 g/t Au, 780 g/t Ag, and 5% combined Pb and Zn with 5% Sb. Historic drilling had notoriously bad recovery problems, so in many cases grade was not reported for potentially significant intersections. Compilation of all available data during the 2003 exploration season clearly indicated that the currently known strike length of the Number 1 Fault exceeds 750 m with a significant mineralized zone developed over around 450 m.

Figure 7-6: Stardust Mineralized Zones



Source: NorthWest Copper, 2022

7.3.4.2 Zn-Au-Ag-Pb CRD Mineralization: Number 2, 3, 3 Extension, 4b, and East Zones

Mineralization in these zones consists of roughly stratigraphically concordant massive sulphide bodies (mantos) and their oxidized equivalents. The mantos are best developed along permeable and karsted (?) carbonate beds in close proximity to chlorite-altered mafic tuff beds. The mantos occur through the Number 2 to Number 4b Zones and appear to merge into the CCS Zone. Drilling results have failed to find substantial discordant chimney feeders to these mantos, although narrow feeders may have been hit locally (Megaw, 1999). The mantos occur dominantly in structurally thickened and deformed zones along the crests of antiforms. There is some evidence for nesting, or repetition, of mantos in successive limestone beds, giving an overall morphology reminiscent of the stacked "saddle-reef" mantos.

7.3.4.2.1 Number 2 Zone

The Number 2 Zone is a minor oxidized replacement zone similar to the Number 3 Zone. The Number 2 Zone is located very close to the crest of a regional antiform which lies just north of the Number 2 Zone trenches. Surface sampling indicates an average of 2.3 g/t Au, 109 g/t Ag, 2.16 % Zn and 2.09 % Pb across an average of 5.3 m true width. This zone has a strike length, based on surface oxidation, of around 200 m. Its continuity at depth is much more problematic as significant intersections have not been obtained from drillholes to date.

7.3.4.2.2 Number 3 Zone

The Number 3 Zone contains the largest identified CRD resource identified to date at Stardust. It is thoroughly oxidized to depths of greater than 100 m from the surface. The style of mineralization may be highly amenable to low-cost heap-leach extraction processes.

The thickest portions of this manto zone occur in carbonates surrounding a mafic tuff bed along the crest of a regional scale antiform. The manto may have the form of an oxidized saddle-reef replacement body. Drilling has failed to find a feeder vertically beneath it, suggesting that it was probably fed from one end with fluid migration concentrated along the non-reactive tuff bed. Evans (1997) felt that the conduit for this system was downdip along the west limb of the antiform (possibly with a NW rake). This zone, based on the trace of oxidation exposed in surface trenches, has a strike length exceeding 600 metres. The Number 3 zone appears to weaken to the south, south of the Number 2 Zone trenches. The northern extension of the Number 3 Zone has received very limited exploration, as has the downdip extensions to this mineralization.

7.3.4.2.3 Number 4b Zone

The Number 4b Zone CRD manto is developed along the 4b antiform, a tight fold, with 60-degree west dips and a 10-15° plunge to the NW. The trace of this fold lies some 300 m to the west of the Number 3 Zone antiform. The two zones are linked by a north-northwest plunging synform. Mineralization occurs as a series of aligned, discontinuous (?) massive sulphide pods (with sparse calc-silicate minerals) following the crest of the fold and also along the contact between limestone on the east and hornfelsed graphitic phyllites to the west. A mafic tuff horizon within the limestone appears to be a major conduit for fluid movement, as is seen in the Number 3 Zone. The 4b Zone is, however, essentially unoxidized: sphalerite, arsenopyrite, coarse-grained well-zoned pyrrhotite, and pyrite are prominently displayed in surface trenches along the zone.

7.3.4.2.4 East Zone

The East Zone was discovered in 2005 by drilling a coincident gold-arsenic soil geochemistry anomaly around 300 m east of the Canyon Creek Skarn. This gold-silver-copper-zinc massive sulphide zone is completely “blind” and has been intersected by five drillholes over a strike length of 150 m. It is open along strike to the north and in both dip directions. The massive sulphide mineralization consists of pyrite, sphalerite, arsenopyrite, and chalcopyrite. The preliminary interpretation is that the zone is a carbonate replacement similar to the Number 3 and Number 4B zones.

7.3.4.3 Canyon Creek Skarn (Number 4 Zone)

The Canyon Creek Skarn (CCS), is the skarn replacement zone lying north of the 4b Zone. The discovery of this skarn is recent enough that it was not included in Ray and Dawson's (1998) compilation on B.C. skarns. Prior to the 2001 season, this zone had been cut by 41 drillholes (97-9, 10, and 11; LD99-03 through 12; and LD00-02 through 29) and a few trenches (Evans, 1997, 1998; Megaw 1999, 2000). A high percentage of the pre-2001 holes in skarn intercept high-grade Cu-Au mineralization along structures cutting garnet-pyroxene skarn. Some of these mineralized structures were surrounded by zones of dispersed mineralization a few metres wide (Megaw, 1999; 2000).

At shallow levels, the skarn is composed of early coarse-grained green tan, grossular-andradite garnet with minor fine-grained greenish-yellow diopside and rare vesuvianite or pyroxene (Ray et al., 2002). Specularite is locally very common as euhedral plates. At depth, a brown garnet stage crosscuts and overprints the green stage, and at even greater depths, a red-brown garnet stage appears (Megaw, 1999). These minerals replace massive limestone and locally replace intrusions (endoskarn). Drilling in 2001 showed that endoskarn increases with depth (cf. LD01-44, 45). Biotite hornfelsed siliceous phyllite is also overprinted by skarn, especially on the north side of Canyon Creek. Mafic tuff units are altered to distinctive green, banded chlorite-garnet units with 5-15% disseminated pyrite and trace chalcopyrite and sphalerite.

Retrograde hydration of the garnet-diopside skarn also increases with depth. In the retrograde zones, the brown-red, brown, and green garnet stages are hydrated to a cream-coloured mass of very fine-grained amphibole, chlorite, quartz, and clays or dark grayish-green masses of felted chlorite, locally preserving the shapes of dodecahedral garnet crystals. Retrograde alteration is often accompanied by a dramatic increase in magnetite, both as fine-grained masses and as pseudomorphs after bladed specularite, and increased amounts of chalcopyrite (Megaw, 2000, Ray et al., 2002)

Mineralization in the skarn occurs as Ag and Au-bearing chalcopyrite and bornite with abundant pyrite, variable sphalerite, and rare arsenopyrite and stibnite emplaced along and surrounding structures that cut the skarn (Megaw, 1999). Much of the sulphide replaces skarn silicates. Numerous stages of sulphide mineralization are identified as:

1. Chalcopyrite deposited in interstices and along garnet grain boundaries.
2. Early pyrrhotite (often later pseudomorphed to pyrite) with minor chalcopyrite and locally intergrown with sphalerite.
3. Pyrite or pyrrhotite (pseudomorphed to pyrite) that is brecciated and healed with later sphalerite or replaced by chalcopyrite.
4. Massive to dispersed, banded and chaotic chalcopyrite along structures and replacing adjoining skarn.
5. Magnetite with interstitial chalcopyrite and/or sphalerite, pyrite or pyrrhotite.
6. Sphalerite with chalcopyrite cut by later pyrite veinlets.
7. Massive sphalerite, brecciated and healed by chalcopyrite and sphalerite.
8. Mineralized skarn, brecciated and healed with epithermal style chalcedonic quartz.

9. Calcite veins filled with Au sulphides/sulphosalts cutting skarn.

The skarn silicates tend to end abruptly and massive sphalerite-chalcopyrite-pyrite-pyrrhotite mineralization is locally well-developed along the contact of skarn with recrystallized limestone (marble front). It is near this front that the very high-grade gold grades associated with the 2002 drilling have been recognized (Oliver, 2002). More recent drilling by Sun Metals in 2018 resulted in the discovery of the 421 zone, a deeper and wider extension of the previously explored zones. High-grade gold and sulphide-rich replacement bodies may be considered transitional mineralization between the skarn and 4b style of replacement mineralization.

The mineral zones at Canyon Creek collectively extend around 1,200 m along strike and have been intersected from surface down to 900 m in depth.

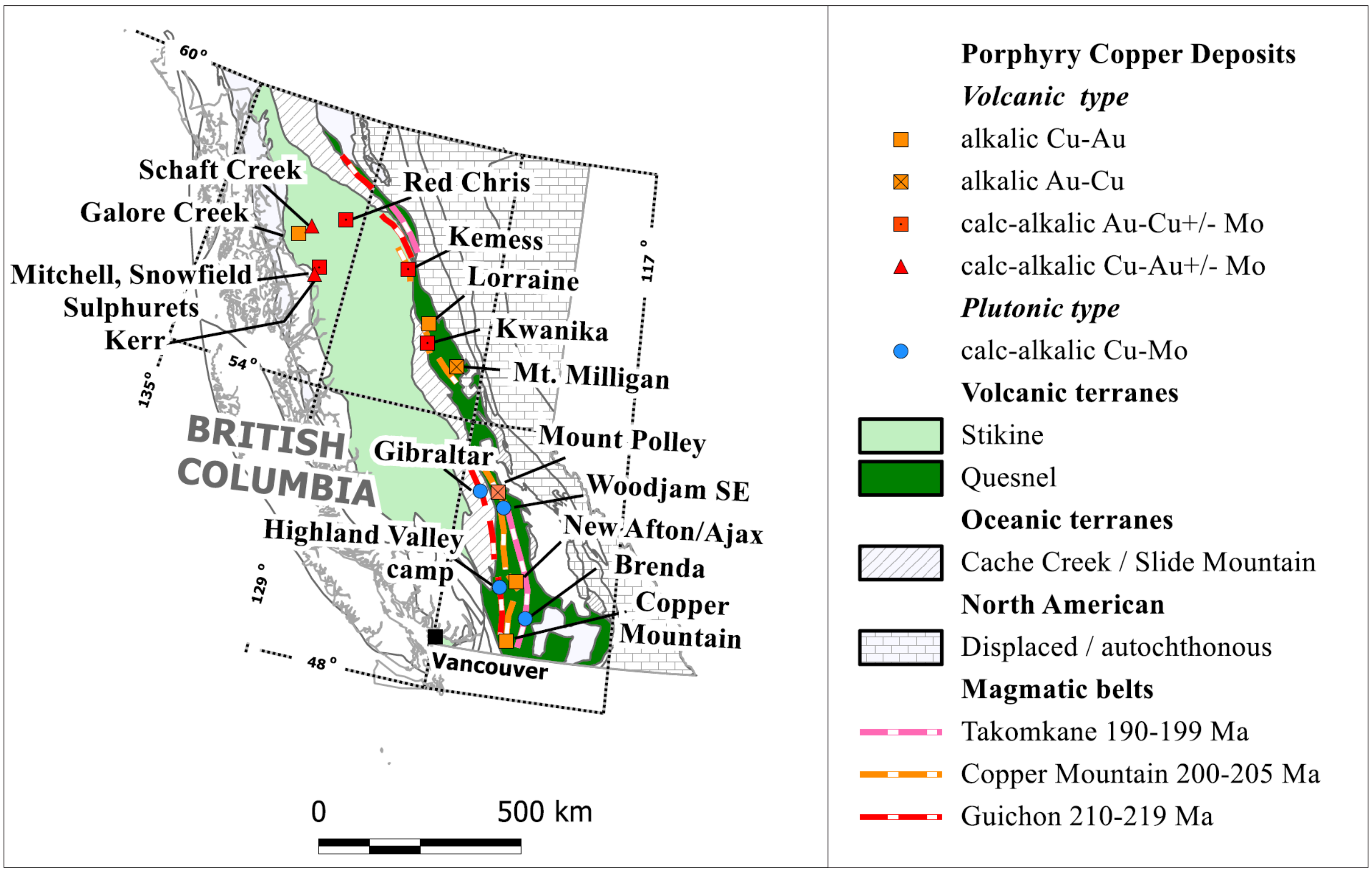
All previous NI 43-101 compliant mineral resource estimates (Simpson, 2010 and Simpson 2018) were confined to the CCS Zone.

8 DEPOSIT TYPES

8.1 Kwanika Deposit Types

Porphyry Cu-Au deposits in British Columbia occur as pre-accretionary deposits in both the Quesnel and Stikine Terranes, and also in post-accretionary settings. They are classified into alkalic, transitional and calc-alkalic sub-types, based on the composition of the host rocks, Cu:Au metal ratios, alteration types, and presence or absence of quartz stockworks (e.g., MacMillan et al., 1995). British Columbia hosts at least one major example of each porphyry sub-type (Figure 8-1). The Central and South Zones at Kwanika have characteristics compatible with models for porphyry deposit formation, although the characteristics of the two zones are different and their genetic relationship, if any, remains unknown.

Figure 8-1: Porphyry Deposits in British Columbia



Source: Osatenko et al., 2020

The Central Zone deposit at Kwanika has characteristics of both alkalic and calc-alkalic porphyry sub-types. It is similar to the classic alkalic porphyry model in that the mineralization is associated with a monzonite that contains abundant alkalic feldspar but only minor quartz. Mineralization, however, is related to a strong quartz stockwork, which is more compatible with the calc-alkalic sub-type. The Central Zone deposit may be transitional between the alkalic and calc-alkalic sub-types.

The South Zone deposit at Kwanika is a structurally controlled porphyry deposit hosted by quartz monzonite to quartz monzodiorite, and mineralization is related to quartz veins and includes significant concentrations of Mo. These features are consistent with the calc-alkalic porphyry sub-type. Structural control is implicated by a close association of Cu-Au-Ag-Mo mineralization with zones of brittle deformation that have been inundated by intense K-spar ± silica flooding. The West and East faults that bound the deposit are interpreted to be both the causes of this brittle deformation and conduits for fluid flow.

8.2 Stardust Deposit Types

The current exploration concept for the Stardust property is based on a model proposed by Sillitoe and Bonham in 1990 (Figure 8-2). The model links porphyry, skarn, carbonate replacement, vein, and sediment hosted types of mineralization. Any one or several of these deposit types can be present in a mineralized system (Hanson, 2007). According to the model, Cu-Au-bearing garnet skarns occur as replacements of the limestone host rocks adjacent to a mineralized porphyry stock. Outboard of the skarn zones, structurally and stratigraphically controlled carbonate replacement massive sulphides deposits (CRD) occur as mantos and chimneys. Sulphosalt veins can occur outboard of the CRD or overlie them in leakage zones. The distal end member mineralization style in this system is the sediment hosted Au-As-Sb (Carlin-type) deposit (Hanson, 2007).

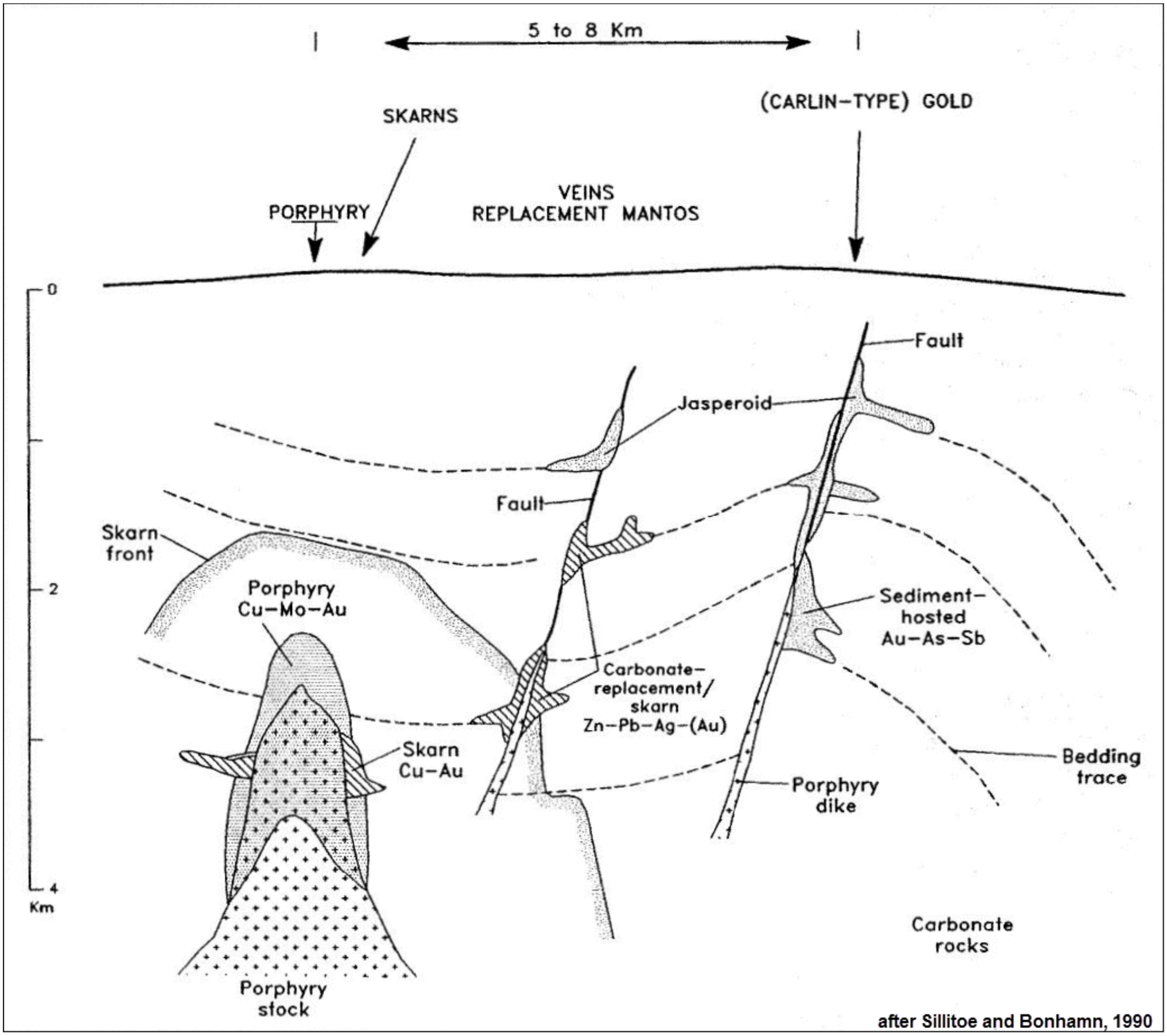
A conceptual model for the Stardust property showing the relative positions of the various mineralized zones is illustrated in Figure 8-3.

8.2.1 Carbonate Replacement Deposits

CRDs are epigenetic, intrusion related, and high-temperature sulphide-dominant Pb-Zn-Ag-Cu-Au-rich deposits. These CRD's typically grade from lenticular or podiform bodies developed along stock, dyke, or sill contacts to elongate-tubular to elongate-tabular bodies referred to as chimneys and/or mantos depending on their orientation. Limestone, dolomite, and dolomitized limestones are the major host rocks. Mineralized material grade outward from sulphide-rich skarns associated with unmineralized or porphyry-type intrusive bodies to essentially 100% polymetallic massive sulphide bodies. Both sulphide and skarn contacts with carbonate host rocks are razor sharp and evidence for replacement greatly outweighs evidence for open-space filling or syngenetic deposition (Titley & Megaw 1985). In reduced, high to low-temperature systems, proximal to distal metal zoning generally follows: Cu (Au, W, MO), Cu-Zn (Ag), Zn-Pb-Ag, Pb-Ag, Mn-Ag, Mn, and Hg. This zoning may be very subtle and large scale (Prescott 1916; Morris 1968; Megaw 1990) or tightly telescoped and smaller scale (Graf 1997).

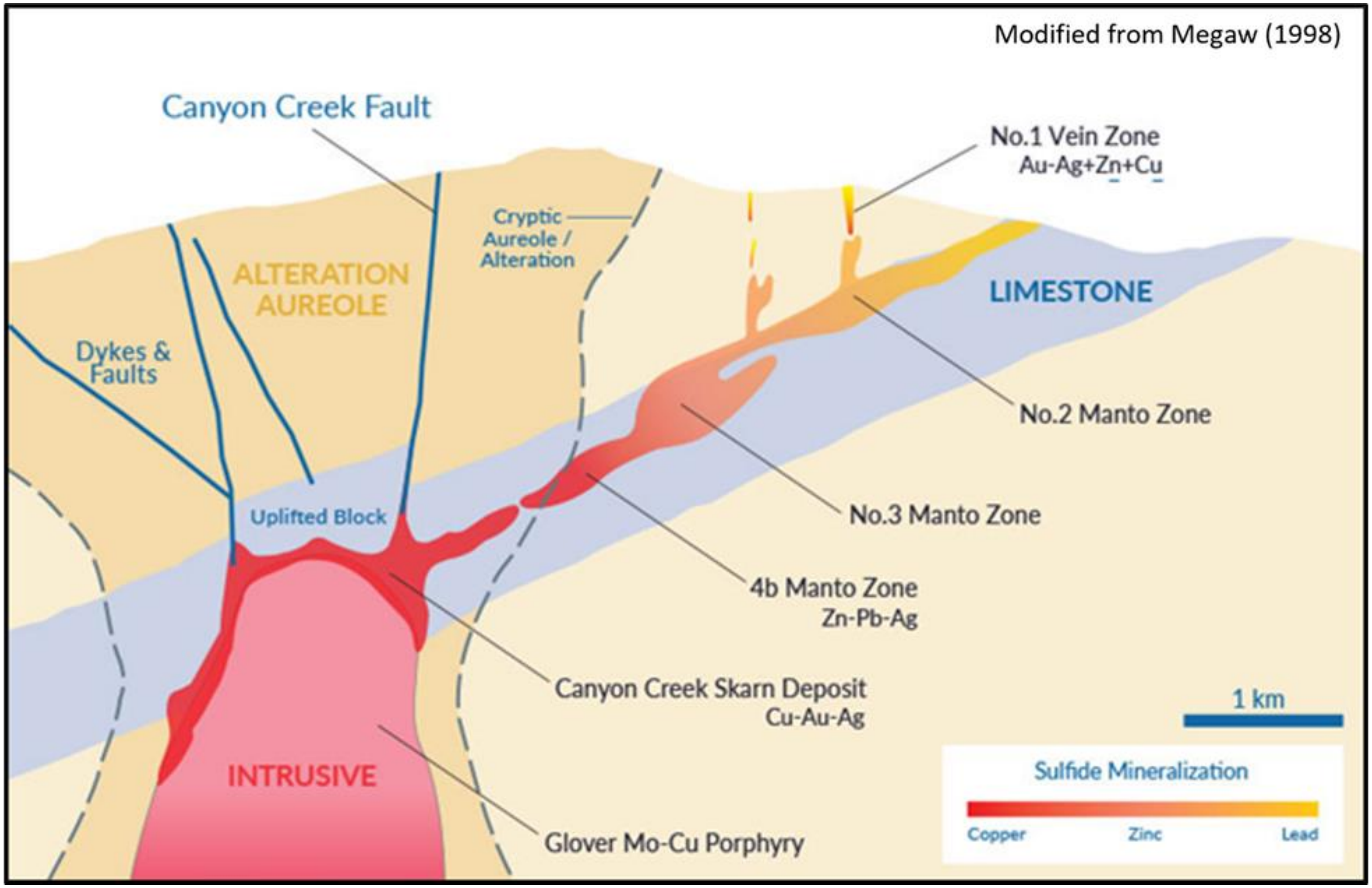
CRD mineralization is associated with polyphase intrusions that evolve from early intermediate phases towards late, highly evolved felsic intrusions and related extrusive phases. The intrusions most closely related to mineralization are usually the most evolved phases and these are not exposed in many districts. However, they are often encountered when the system is explored to depth.

Figure 8-2: Schematic Model of Possible Links Between Porphyry Districts and Sedimentary Deposits



Source: Stilltoe & Bonhamn, 1990.

Figure 8-3: Stardust Conceptual Model



Source: Megaw, 1998.

CRD exploration is difficult enough that considerable care should be taken in selecting a target district/deposit prior to high-cost detailed exploration. CRDs are typically metallurgically docile, amenable to low-cost mining methods and the environmental footprint is minimal.

Many features of CRDs tend to be well-zoned at district, deposit, and hand-sample scales. The most important zonations are:

- ore and gangue mineralogy and metal contents
- deposit geometry
- intrusive geometry and composition
- structural controls on mineralization
- alteration
- isotopic characteristics of wall rocks.

In general, the largest systems show the best-developed zoning and repetition of zoning and paragenesis. Zoning tends to be most extensive in the elongate manto and chimney systems where individual zones may extend over kilometrers vertically and laterally (Megaw 1990, 1998). Zoning in large stock contact skarn systems is typically more compressed because of telescoping and repeated overprinting (Graf 1997). In all cases, multi-phase mineralization is a reliable indicator of large systems.

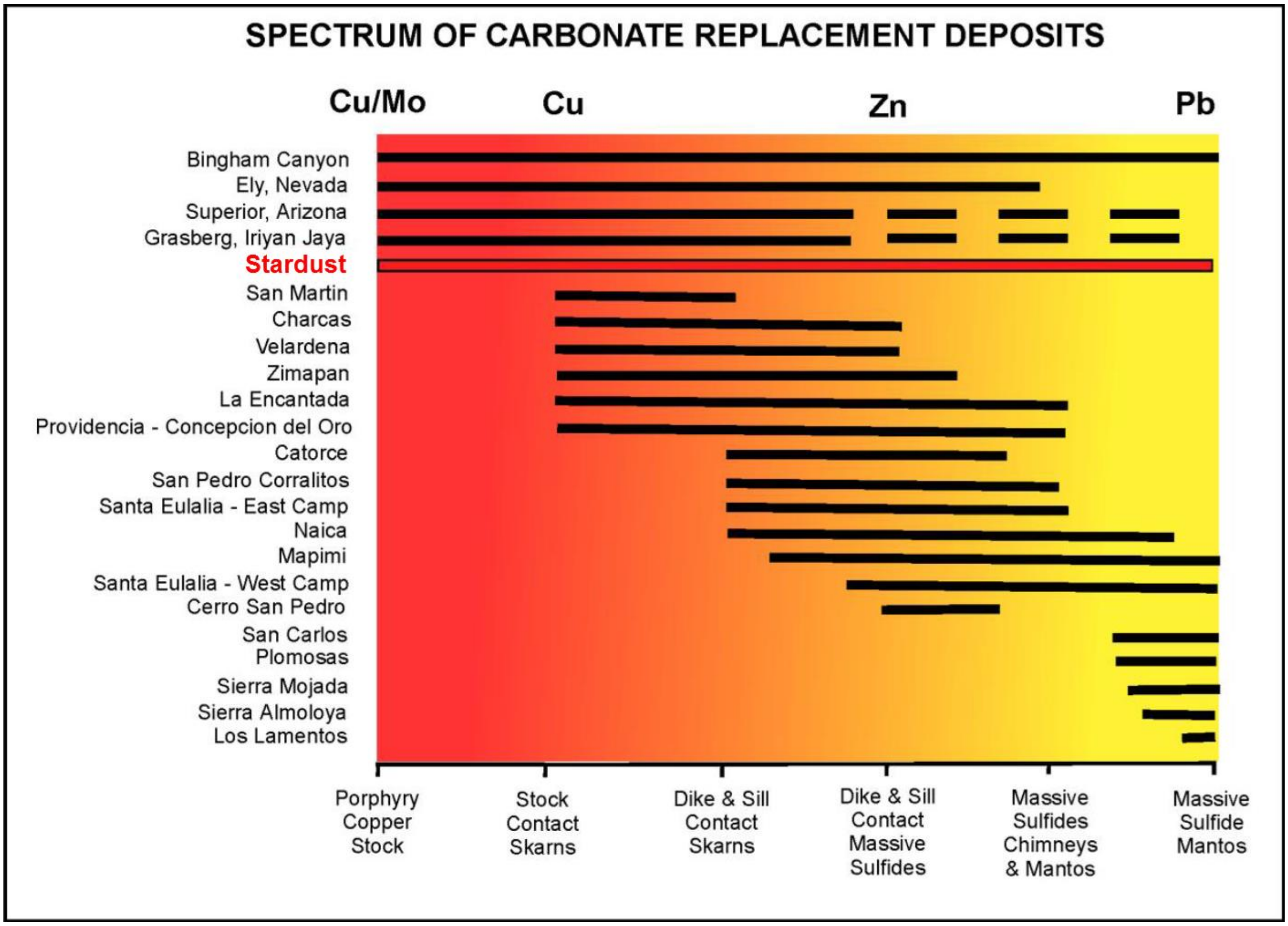
The evolution of CRD-skarn systems in time and space, and the gradations seen in single deposits or districts suggests that the various manifestations of the deposit type can be considered part of a spectrum as illustrated in Figure 8.4 (Einaudi et al. 1982; Megaw et al. 1988; Titley 1993; Megaw et al. 1998) including:

- stock contact skarns: formed against either barren or productive (i.e., porphyry copper or molybdenum) stocks
- dyke and sill contact skarns
- dyke and sill contact massive sulphide deposits
- massive sulphide chimneys
- massive sulphide mantos
- epithermal veins (in some cases).

This conceptual framework allows examination of the mineralization, alteration, intrusion types, host rock and other characteristics of a given deposit and determining where it lies within the spectrum. Examination of the composition, geometry, and controls on intrusion emplacement, if possible, is essential to determining district zoning and level of exposure. Perhaps most importantly, understanding of the host rock tectono-stratigraphy can allow rapid determination of the potential for more mineralization in the host section at depth or laterally in the known favourable beds, or in previously unconsidered host units.

Structural fabrics are the dominant control variable on mineralization in CRDs, as they control intrusion emplacement and channel fluids into favourable host strata. Most CRDs lie in fold-thrust belts on major structural domes, arches, anticlines, synclines or homoclines, and most districts have structural grains controlled by faulting and fracturing related to regional deformation (Megaw et al. 1988). Deposits are often elongate and parallel district-wide structural trends but may not be restricted to a given structure over great lengths.

Figure 8-4: Spectrum of Carbonate Replacement Deposits



Source: Megaw, 2001.

Intrusive stocks commonly occur beneath or adjacent to the most proximal portions of CRD systems, although in many cases they do not crop out. Where intrusions are exposed, they are generally less than 5 km^2 in areal extent. These stocks are generally polyphase with compositions grading from early diorite to late granite. Texturally, these intrusions range from equigranular to porphyritic and massive to highly fractured depending on age and proximity to paleosurface. The central stocks may be barren, contain porphyry copper or molybdenum systems, or have marginal zones with porphyry copper or molybdenum affinities (Megaw, 1998). In many systems, the early phases of the intrusion have associated skarnoid or barren skarn, whereas skarn and mineralization are related to later, more highly differentiated phases (Meinert, 1995 and 1999; Graf, 1997; Megaw and others, 1998).

Dykes and sills characterize the intermediate reaches of CRDs and there is often evidence for multiple dyke/sill emplacement events (Megaw 1990). These intrusions may be compositionally homogeneous (Megaw 1990) or there may be compositional evolution between dyke/sill phases (Graf 1997). Textures range from porphyritic to aphanitic, locally with narrow gradations between textural domains (Megaw 1990). Chimney and replacement veins are the most common types of mineralized material associated with these intrusions, although mantos locally occur along sill contact.

The distal zones of CRDs are characterized by massive sulphide bodies lacking an associated intrusion. These commonly have the form of high angle to vertical slab-like replacement veins or elongate pipe-like chimneys or low angle to horizontal tabular or elongate tongue-shaped mantos, generally crudely stratabound. Mantos may be developed entirely within selected beds or groups of carbonate beds, or may occur with one or more non-reactive, relatively impermeable sedimentary or intrusive rock contacts.

Development of carbonate rock alteration in CRDs, like mineralization, is highly variable in type and in scale. The major alteration types are:

- Skarnoid or hornfels: These are typically very fine-grained, mineralogically simple, calc-silicate and silicate assemblages formed through thermal metamorphism without significant addition of outside components. Skarnoid typically forms from a limestone or shaly limestone precursor, whereas hornfels forms from shale or limy shale precursors. Hornfels and skarnoid commonly develop in the thermal aureole around the largest volume (often early) intrusive phase and may aid in ground preparation for later metasomatic events. Hornfels mineralogy may be zoned with respect to the thermal centre, commonly with pyroxenes proximal and biotite more distal. Skarnoid and hornfels often contain abundant fine-grained pyrite or pyrrhotite, but seldom significant amounts of metal sulphides unless it has been overprinted by subsequent hydrothermal events.
- Skarn: Skarns are fine to very coarse-grained, often mineralogically complex, calc-silicate or calcic-iron silicate assemblages formed through metasomatism with significant addition of outside components. Endoskarn is skarn formed at the expense of intrusive rock, exoskarn is skarn formed at the expense of wallrocks to the intrusion - most commonly carbonates. Skarn commonly develops around lesser volume, more fluid-rich intrusive phases and may overprint hornfels or skarnoid to varying degrees. Anhydrous talc-silicate minerals (dominantly pyroxenes and garnets) characterize the early “prograde” skarn phase generated during rising temperatures related to magma emplacement. Hydrous talc-silicate minerals (dominantly amphiboles, chlorites, and clays) formed at the expense of predecessor prograde minerals characterize the later “retrograde” skarn assemblage. Retrograding occurs as temperatures drop and variable amounts of magmatic fluids and groundwater invade the skarn zone. Skarns are said to be mineralized when they contain sulphide minerals of economic interest. Said sulphides may be co-deposited with the calc-silicates, but more commonly are introduced along structures that cut the skarn, replacing skarn minerals and unaltered wallrocks. Complex mineralized skarn systems typically show multiple intrusive phases and a repetition of sulphides replacing talc-silicates presumably reflecting successive intrusive and hydrothermal events. In some systems, different compositions of skarn and sulphides characterize each phase (Megaw and others, 1998).
- Marbleization and Recrystallization: These are present in virtually all CRD systems and range from narrow zones around mineralization to zones hundreds of metres wide (Titley & Megaw 1985; Megaw et al. 1988).
- Silicification or Jasperoid development: These consist of fine-grained silica replacements of carbonate rocks, with or without appreciable amounts of metals, and are very common in the peripheries of some CRD systems (Titley & Megaw 1985; Megaw et al. 1988; Megaw 1990).

8.2.2 Porphyry Cu-Mo-Au Deposits

Porphyry copper deposits are large, low-grade, intrusion-related deposits which provide the major portion of the world’s copper and molybdenum and to a lesser degree gold. The deposits are formed by a shallow magma chamber of hydrous, intermediate composition at depths of less than five kilometres. When the magma crystallizes, fluids are released; the fluids’ movement upwards through overlying rocks results in hydrothermal alteration and deposition of sulphide minerals both as disseminations and as stockwork mineralization. There is a clear spatial and genetic association between the intrusion and the alteration zones at a regional and local scale.

The defining characteristics that distinguish porphyry deposits are:

- Large size
- Widespread alteration

- Structurally controlled mineralized material superimposed on pre-existing host rocks
- Distinctive metal associations
- Spatial, temporal, and genetic relationships to porphyritic intrusions

These deposits in British Columbia typically occur in the Intermontane Belt, which is host to the Quesnellia, Cache Creek, and Stikinia Terranes, and based on the composition of the host rocks comprising three specific types: Alkalic, Transitional, and Calc-Alkalic.

The Glover Stock is an intrusion of Eocene age emplacement (circa 51-52 Ma by U-Pb zircon dating; Ray et al., 2002). It is inferred to be emplaced between at a relatively shallow 1.1 to 1.9 km depth as supported by field structural relationships and fluid inclusion work (Ray et al., 2002; Dunne and Ray, 2002) and less than 5 km (Megaw, 2001). The stock is a multi-phase composite intrusive complex and most of its rocks are weakly to strongly feldspar hornblende biotite porphyritic. Compositionally it ranges from mafic diorite-monzodiorite to leucocratic monzonite-quartz monzonite (Ray et al., 2002).

The Glover Stock shows many features prospective to host porphyry-style mineralization. Molybdenite±chalcopyrite-bearing veinlets are associated with several generations of veins containing quartz, K-feldspar, sericite, pyrite, and tourmaline (Ray et al., 2002). Alteration assemblages include pervasive albitic or potassic (K-feldspar, sericite, and biotite), silicic, pyritic, and argillic. A fluid inclusion study supports a combination of highly saline and dilute fluids that show a transition from high-pressure lithostatic conditions during porphyry emplacement to lower pressure hydrostatic conditions during vein formation (Megaw, 2001). Such a transition may be indicative of a long-lived shallow emplacement. ‘Pebble’ dykes logged in drill core are similar to breccia dykes seen in major porphyry systems. These breccias are interpreted to record violent volatile release events coincident with the transition from lithostatic to hydrostatic conditions (Megaw, 1990; Fournier, 1999; Jones and Gonzalez-Partida, 2001).

Porphyry-related alteration styles include:

- Tourmaline-rich greisen along numerous structures cutting the biotite diorite in LD20O1-30.
- Potassic alteration consisting of secondary biotite selvages on mineralized veinlets secondary euhedral and/or “shreddy” biotite affecting primary biotite and hornblende and secondary K-feldspar flooding.
- Weak to pervasive sericitic alteration of intrusion
- Widespread chloritized and epidotized hornblende and feldspar
- Mineralization of the intrusions consists of crosscutting veinlets including:
- Quartz-K-feldspar-pyrite veinlets
- Quartz-K-spar-pyrite-chalcopyrite veinlets
- Quartz-K-spar-pyrite-molybdenite veinlets
- Hornblende replaced by specularite replaced by magnetite with interstitial chalcopyrite.
- Open sigmoidal cavities

8.2.3 Comments

The current mineral resource estimate is for the CCS Zone, regarded as a skarn-hosted carbonate replacement deposit and the exploration programs have been planned on this basis.

9 EXPLORATION

9.1 Kwanika

This section describes exploration work since Serengeti acquired the property in 2004. While drilling is referenced in this section, more detailed descriptions of drilling campaigns and results are discussed in Section 10 of this report.

In 2005, Serengeti (now NorthWest Copper) conducted a 530 line-km airborne magnetic/radiometric survey and collected 11 rock samples on the Kwanika and Germansen properties (Osatenko, 2005). The airborne survey identified a small magnetic anomaly on the east side of the known South Zone porphyry copper-gold deposit, with similar anomalies trending to the north-northwest of the deposit, as well as to the south. Six of these anomalies are associated with weak K/Th radiometric anomalies. The copper, gold, and molybdenum values in rock samples associated with the deposit outcrops along Kwanika Creek ranged from 507 ppm to 10,740 ppm copper, 22 ppb to 416 ppb gold, and 2 ppm to 533 ppm molybdenum.

In 2006, the discovery holes into the Central Zone starting with K-06-09 (0.69% Cu and 0.54 g/t Au over 111 m) were drilled by Serengeti Resources; historical drilling was concentrated at the known South Zone, so these were the first holes into the Central Zone. Over the season Serengeti drilled 10 diamond drillholes for 1,874 m. Additionally, Peter E. Walcott and Associates Geophysics (Walcott) was contracted by Serengeti to conduct ground-based IP and magnetics surveys in the vicinity of the Central and South Zone deposits. A total of 26.9 line-km of survey was competed. The results outlined a significant IP signature over the Kwanika deposit as well as a continuation of this IP anomaly undercover to the north-northwest.

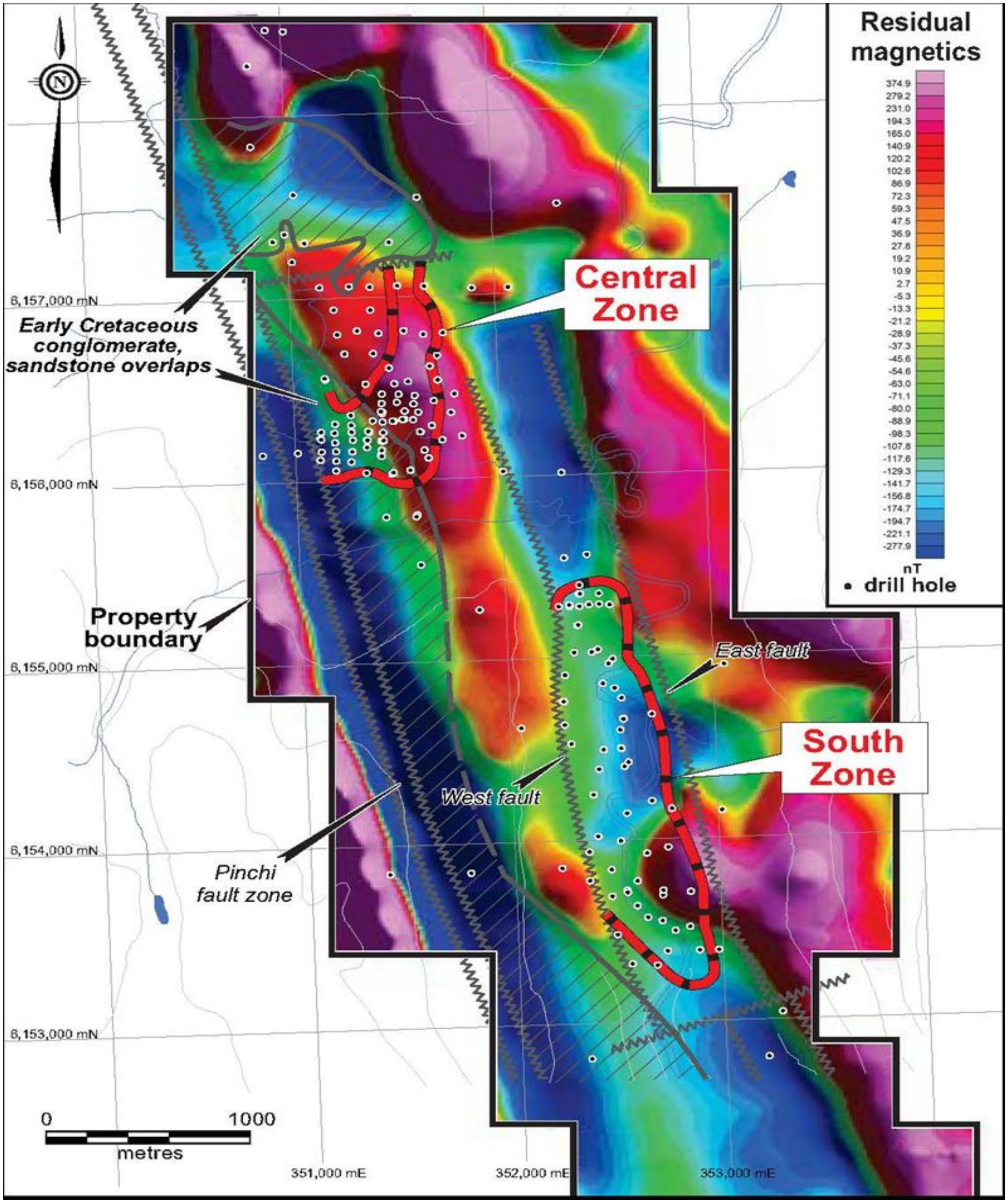
Following the success of the drilling in the Central Zone in 2006, Serengeti followed up with 47 diamond drillholes in 2007, totalling 22,415 m. Concurrently, a regional airborne magnetic and electromagnetic (EM) survey, totalling 320 line-km over the Kwanika property was carried out by Serengeti (Figure 9-1). The purpose of the survey was to test for zones of conductive sulphide mineralization, outline any porphyry-style intrusive complexes, and aid in geology and structure interpretation. The survey identified multiple magnetic high and low resistivity anomalies throughout the property. The anomalies generated were coincident with and demonstrated a north-northwest trend that is seen in the South and Central Zone deposit areas.

This was followed by another 49 diamond drillholes for 26,553 m in 2008. During this time, Walcott was again contracted to conduct 70 line-km of 100 m spaced lines of IP surveys from south of the two known deposits to the northern boundary of the Kwanika property (Figure 9-2). The shape of this anomaly is directly coincident with the outline of the currently known, near surface (i.e., within around 200 m) copper-gold mineralization in the Central and South Zone deposits.

Drilling on the property resumed in 2016 with 5 diamond drillholes totalling 2,446 m and an additional 2.4 line-km of surface IP work. In August of 2016, Serengeti contracted McElhanney to fly a LiDAR survey over the Central and South Zones of the Kwanika project. The resulting data was used to create a high-resolution topographic surface. There is no work recorded in 2017.

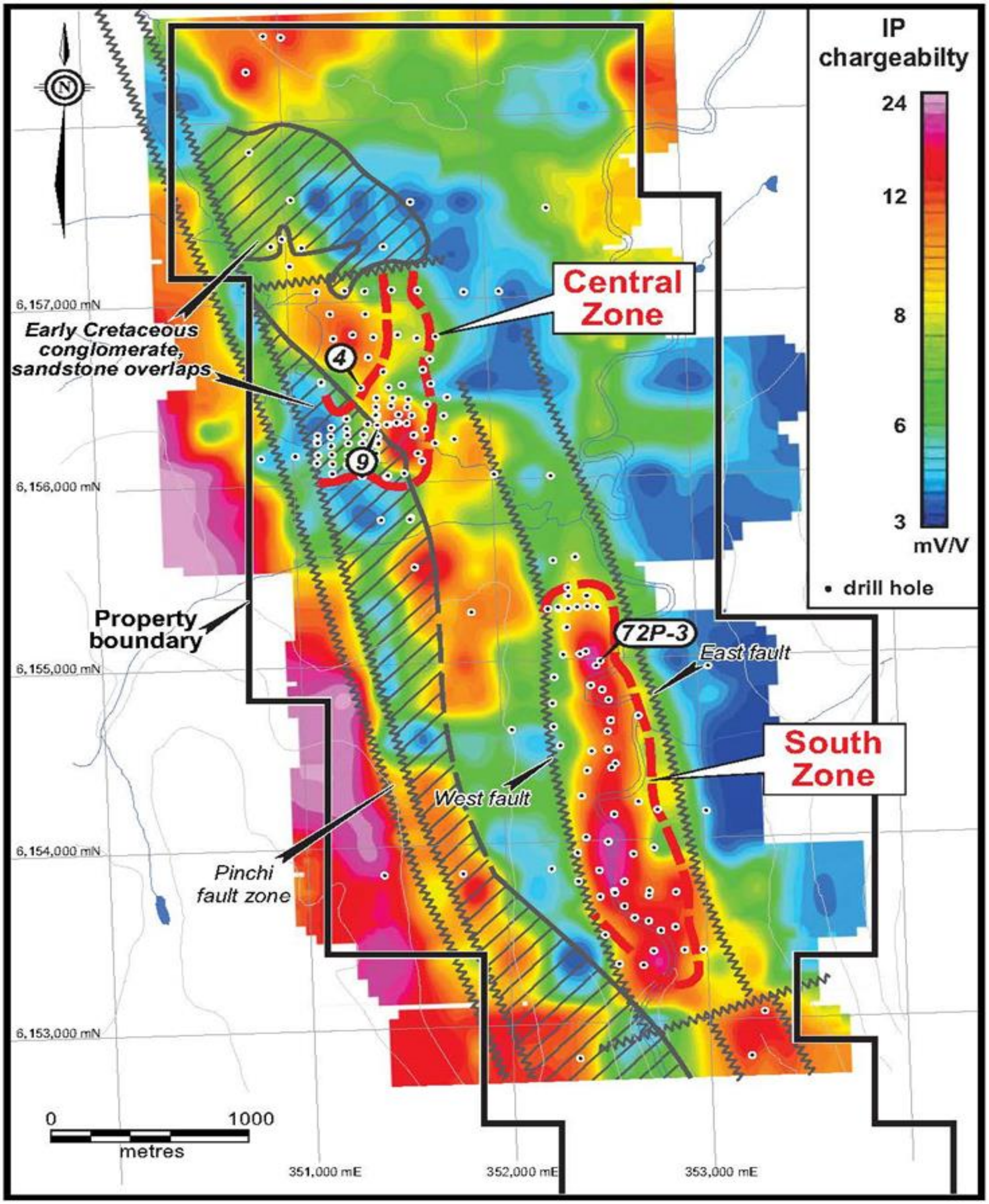
During the 2018 field season, Serengeti drilled 21 drillholes for 7,411 m to support engineering and economic studies. This was followed by no work in 2019 and an additional 4,355 m in 9 drillholes during 2020 as well as 15 line-km of ground IP survey.

Figure 9-1: Residual Magnetics from 2007 Airborne Magnetic and EM Survey over the Kwanika Property



Source: NorthWest Copper, 2022

Figure 9-2: IP-Chargeability from 2008 IP Survey over the Kwanika Property



Source: NorthWest Copper, 2022

From 2009 to 2012, Serengeti drilled 17 (6,249 m), 28 (7,619 m), 5 (1,724 m), and 4 (2,446 m) diamond drillholes, respectively, with 3 line-km of IP in 2012. This was followed by a period of exploration quiescence for 3 years.

Following the creation of NorthWest Copper Corporation with merger of Sun Metals and Serengeti in early 2021, NorthWest Copper drilled 20 holes for 8,696 m.

In 2021, NorthWest Copper collected 385 soil samples, 238 silt samples, and 100 rock samples. The 385 soil samples were collected in the South Creek and East Tsayta zones. The soil sample program was designed to test geophysical and geochemical anomalies. Samples were taken along 29 lines running roughly East-West. The lines were spaced either 100 or 200 metres apart, with the samples being spaced 50 metres apart on the tighter lines and 100 metres apart on the other lines. Sample locations were field located using a handheld GPS. The stream sediment/silt sampling program was designed to sample all mapped streams on the western block of Kwanika claims. The 238 silt samples were taken typically at the bottom of streams, where the tributary met up with a stream of a higher order. Sample locations were field located using a handheld GPS. The sampling geologist recorded information including stream width, stream flow, surrounding vegetation, silt colour, environment, slope, contamination, bank type, sample site and clast information for each sample locations. The 100 rock samples were collected from eight different prospects. Sample locations were field located using a handheld GPS. The sampling geologist recorded information including location, prospect, and field descriptions including lithology, mineralization, alteration, weathering, colour, magnetism and grain size were noted for each rock sample. Results of the 2021 surface geochemistry programs demonstrated several anomalies in the areas of known mineralization and supports the idea that surface soil/silt sampling is a good method for direct targeting in this region.

Also in 2021, Walcott conducted 12 line-km of ground IP on four lines. A pulse type system consisting of three main units, a receiver, a transmitter and a generator was used during the survey. The horizontal positions of the survey stations were recorded using a Garmin GPSmap 64CSx. Walcott also completed 2,450 line-km of heliborne magnetic surveys over the western extent of the Kwanika property. The lines flown during the survey were oriented at 060 with a spacing of 75 metres, orthogonal tie lines were spaced at 750 metres. A stinger system which consists of a C-824 Cesium Magnetometer, a Mag-13 Fluxgate and an Optilogic RS-400 Laser Range Finder was mounted on an ASTAR helicopter. Analysis of these surveys remains ongoing as of the date of this report and will be utilized in future exploration targeting.

In 2022, SkyTEM Canada Inc. (SkyTEM) completed 1,880 line-km of electromagnetic and magnetic survey over the main Kwanika property. Flight line spacing was 100 m by 1,000 m with an azimuthal direction of 061/241, and tie lines were spaced 1,000 m and flown in a 051/331 azimuthal direction. The airborne instrumentation comprising a SkyTEM312M system includes a time domain electromagnetic system, a magnetic data acquisition system and an auxiliary data acquisition system containing two inclinometres, two altimetres and two DGPS' all mounted on a frame ~40 m beneath an ASTAR helicopter. Analysis of these surveys remains ongoing as of the date of this report and will be utilized in future exploration targeting.

Several Mineral Resource Estimates and corresponding Technical Reports have been produced with the continued exploration and updating of the geologic interpretation. Prior to the current Mineral Resources described in this report, Kwanika Central was last updated in 2019 by Moose Mountain Technical Services of Cranbrook, B.C. Canada, and Kwanika South was last updated in 2016 by SRK in Vancouver, B.C. Canada. Those estimates are no longer considered current due to the additional exploration work carried out on the project since then.

9.2 Stardust

Historic exploration work on the property as outlined in Section 6 has been described in previous Technical Reports (Simpson, 2010 & Simpson 2018).

Sun Metals and NorthWest Copper have carried out four exploration programs between 2018 and the end of 2021.

9.2.1 Topographic Survey & Imagery

On June 23rd, 2018, McElhanney Consulting Services Ltd. (McElhanney) of Vancouver, B.C. performed a Light Detection and Ranging survey (LiDAR) coupled with an aerial photo acquisition over 88 km^2 of the Stardust property. LiDAR data was collected using the Optech Galaxy scanner mounted in a twin-engine Piper Navaho.

Raw data was processed by McElhanney and included the extraction of 1-metre contours and digital elevation model (DEM) bare earth hill-shade images.

9.2.2 Geological Mapping and Prospecting

In 2018, a significant effort was made to compile, and field validate historic geology maps and outcrop locations. An updated property geology map is presented in Figure 7-3. Because of limited exposure in many locations, relationships between various rock types were often difficult to determine. In these areas, locations of outcrops were noted and lithologies were checked against historical maps to check the validity. Special attention was given to the identification of carbonate strata since it is necessary for CRD mineralization.

9.2.3 Geochemical Sampling

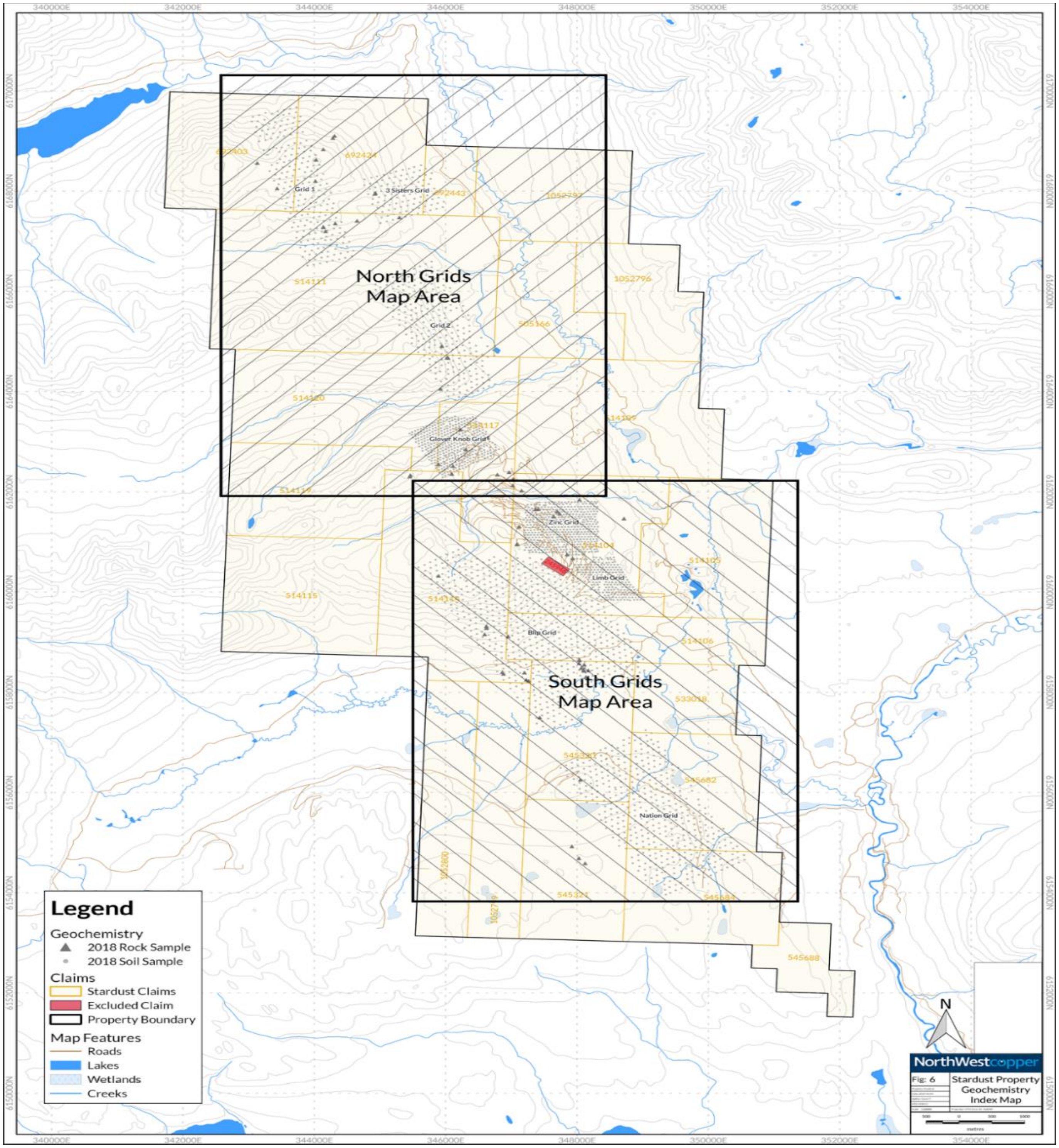
In 2018, a total of 2,804 soil samples were collected over 8 separate grids (Figure 9-3). The soil sample grids were designed to test potential targets previously identified by Aurora Geoscience in 2012 and Sun Metals in 2017 based historic geochemical and geophysical programs. Grids were orientated to be perpendicular to the strike of local stratigraphy and sampling locations were specified prior to field collection. Sample and line spacing were either 50 or 100 m apart depending on the specific grid. Alternating lines within a grid were offset by either 50 or 100 m depending on the specific grid. Sample locations were field located using a handheld GPS. Sampling targeted B and C Horizon soils. Sample depth, soil horizon, and soil colour data were recorded for each sample. Detailed soil sampling procedures are presented in Section 11.

The 2,804 soil samples taken in 2018 were integrated into a historical database of 6,264 samples, making a total database of 9,068 samples.

The 2018 soils sampling and prospecting program demonstrated that much of the historical work is accurate and supports the idea that soil sampling is a good method for direct targeting in this region. This was illustrated by the discovery of a new manto in the GD zone that is seen in drillholes DDH18-SD-415 and DDH18-SD-417. Additionally, the results reaffirm that the zone with most compelling surface geochemistry is south of the Glover intrusive complex, where the different manto zones crop out.

A total of 77 rock samples were collected during the 2018 exploration program during the field mapping program. Because of limited exposure in many locations, sampling was not carried out on a systematic grid and results are not considered to be representative of the property as a whole. Location, source, source size, and field descriptions including rock type and visible mineralization were recorded for each rock sample. Detailed rock sampling procedures are presented in Section 11.

Figure 9-3: Stardust 2018 Geochemical Sampling Grids



Source: NorthWest Copper, 2022.

9.2.4 Geophysics

9.2.4.1 2018 Airborne Geophysics

From June 27 to July 17, 2018, Geotech Ltd. (Geotech) of Aurora, Ontario carried out a helicopter-borne geophysical survey. Principal geophysical sensors included a versatile time domain electromagnetic (VTEM™plus) system and a horizontal magnetic gradiometer with two cesium sensors. Ancillary equipment included a GPS navigation system and a radar altimetre. A total of 1,128 line-km of geophysical data were acquired during the survey.

Sun Metals tested 4 different VTEM anomalies with 5 diamond drillholes. All the conductors were identified with the sole exception of Anomaly C.

9.2.4.2 2018 Borehole Geophysics

SJ Geophysics Ltd. (SJ) of Delta, B.C. completed a Volterra bore hole electromagnetic and magnetic (BHEM) survey on diamond drillhole DDH18-SD-421 during September 27th to September 30, 2018. In3D Geoscience Inc. (in3D) of Vancouver, B.C. completed data post processing of the collected by SJ. Preliminary modelling suggests mineralization intersected in DDH18-SD-421 dips to the west and shows greater coupling to the south.

9.2.4.3 2019 EM Ground Survey

SJ Geophysics Ltd. (SJ) of Delta, B.C. completed Volterra fixed-loop surface electromagnetic (EM) surveys during June 13 to September 3, 2019. The survey consisted of 31 lines spaced 100 m apart for a total of 71.85 line-km surveyed. The geophysical contractor in3D Geoscience Inc. (in3d) of Gabriola Island, B.C. completed processing of data.

Results from the surface EM survey showed good correlation between anomalous EM response and known zones of near surface mineralization. The survey was not effective at identifying deeper mineralization.

9.2.4.4 Magnetotelluric Survey

SJ of Delta, B.C. completed a Volterra surface magnetotelluric (MT) survey from September 5 to September 7, 2019. The survey consisted of two near orthogonal 3 km lines. in3D of Vancouver, B.C. completed post processing of data.

The MT survey results did not correlate well with known zones of near surface mineralization or mapped lithologies, nor did it identify significant geophysical anomalies at depth.

9.2.4.5 2019 Borehole Geophysics

SJ of Delta, B.C. completed Volterra BHEM surveys on 17 diamond drillholes during June 21 to December 4, 2019; in3D of Vancouver, B.C. completed post processing of data.

Results from the BHEM survey showed particularly good correlation between strongly anomalous EM response and increased logged sulphide abundance in diamond drill core. The surveys were also proven to be effective at detecting lateral sulphide mineralization, proximal to surveyed drillholes.

9.2.4.6 2020 Borehole Geophysics

SJ of Delta, B.C. completed Volterra BHEM surveys on 2 diamond drillholes during September 24 to October 1, 2020.

Results from the BHEM survey showed particularly good correlation between strongly anomalous EM response and increased logged sulphide abundance in diamond drill core. The surveys were also proven to be effective at detecting lateral sulphide mineralization, proximal to surveyed drillholes.

10 DRILLING

10.1 Kwanika

In the resource area, a total of 95,255 m of diamond drilling in 228 holes was carried out on the Kwanika property from July 2006 to September 2021. Drilling on the Central Zone totalled 76,156 m in 166 holes, while drilling at the South Zone totalled 19,099 m in 62 holes. The results of this drilling have achieved three main goals:

- Measured, Indicated and Inferred Mineral Resources have been delineated on the Central Zone deposit, which was initially discovered by Serengeti in late 2006
- An Inferred Mineral Resource was delineated on the South Zone deposit
- Several geophysical anomalies on the Kwanika property were tested to explore for possible extensions of the Central Zone deposit.

All but the first five drillholes up to the 2018 drill program were surveyed for downhole azimuth and dip using a Reflex EZ-shot generally at 50-60 m intervals. During the 2018 drill program, a Reflex Gyro, a north-seeking gyroscope, was used at 10 m intervals, either during drilling or upon completion. The 2020 drill program used DeviAlign, a north-seeking gyroscope, azimuth, and dip along the length of the hole were collected using continuous downhole measurements. For the 2021 drill program an Axis Champ Gyro, and north-seeking gyroscope, was used at 30 m intervals after completion of the drillholes.

For all programs from 2006 to 2010 and in 2018, All North Consultants Limited was contracted to carry out a differential GPS (DGPS) survey of the drillhole collar locations on the Kwanika property. Drilling from 2011 to 2012 was surveyed using handheld GPS units and drilling collars from 2016 were surveyed using a Reflex APS GPS unit. For drillhole collar locations in 2020 they were surveyed with handheld GPS. During the 2021 drill program, drill collars were survey suing a Trimble R2 M2 Single GNSS Receiver (> ±1 m).

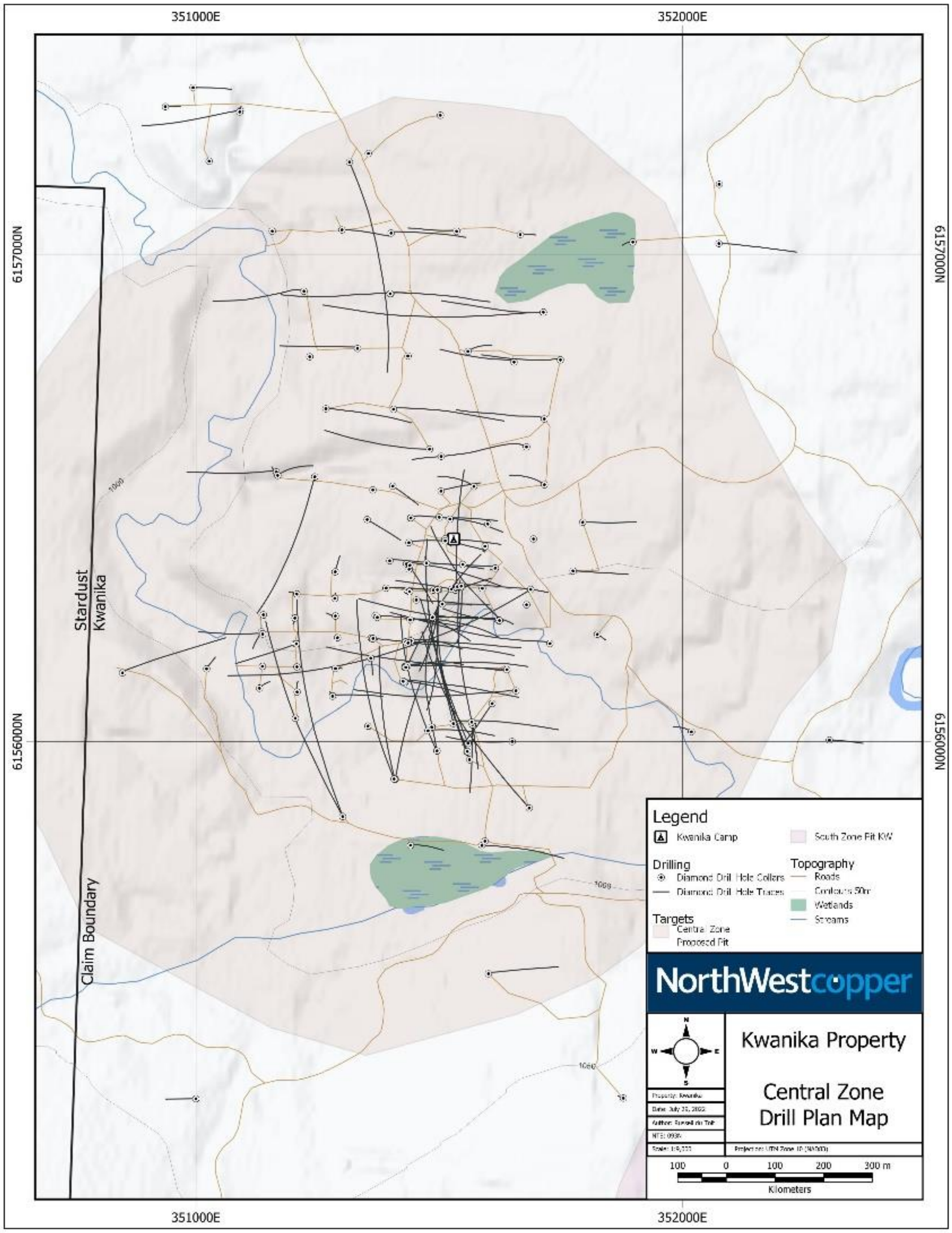
A LiDAR survey flown in 2016 has been used to verify all collar elevations. SRK compared the drillhole collar elevations to the new 2016 LiDAR surface topography and found that the elevations for some of the holes were not in agreement with the high accuracy surface. Collars have been adjusted to conform to the 2016 LiDAR topography.

Core recovery for all drill programs was good to excellent with overall recoveries greater than 95%. Areas of poor or no recovery normally occurred in fault zones. Due to the multiple orientations of the mineralized zones and the limitations of surface drilling, none of the drill intercepts approximate the true thickness. True thickness must be calculated for each intercept based on the angle of the drillhole to the specified zone. Refer to Section 14 of this report for representative cross sections of the deposits.

All drill core was logged for geological and geotechnical characteristics. Geotechnical logging included rock quality designation (RQD), magnetic susceptibility, and specific gravity, and some point load testing. The core was also photographed, sampled, and split by diamond saw or core splitter. The majority of drill core collected on the Kwanika property was NQ (4.76 cm diameter) size. In rare cases, BQ size (3.64 cm diameter) core was drilled when core size had to be reduced due to ground conditions. HQ and HQ3 size (6.35 cm diameter) core were drilled for geotechnical drilling in the 2018 drilling campaign and at the top of several holes that were collared in the sedimentary basin in the Central Zone, as well as for deep drilling in the 2016 and 2021 drilling campaigns.

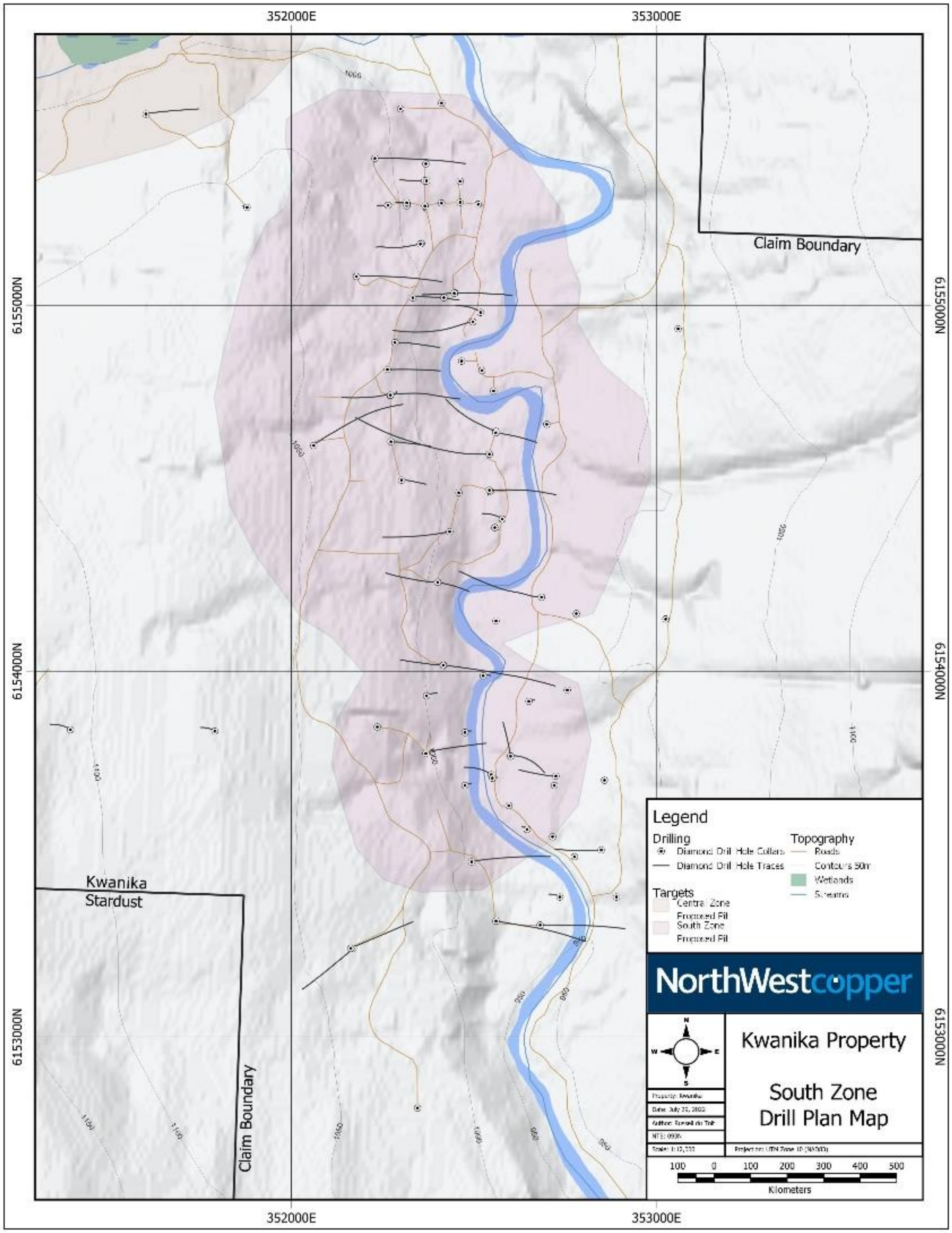
The core is currently stored in conex bins or cross-piled and palletized at the Kwanika camp. Figure 10-1 and Figure 1-2 show the drilling in the Central and South zones respectively.

Figure 10-1: Plan Map of Kwanika Central Zone Drilling



Source: NorthWest Copper, 2022.

Figure 10-2: Plan Map of Kwanika South Zone Drilling



Source: NorthWest Copper, 2022.

10.1.1 Historical Drilling

The South Zone area at Kwanika was drill tested during the period 1965 to 1991 by 30 diamond and percussion drillholes. The historical data are not included in this data compilation and Resource Estimate. Drilling has confirmed and expanded this mineralized zone with drilling that replaces the historical data. The historical drillholes prior to 2006 are discussed in the History section and are not included in the resource estimation for the South Zone.

10.1.2 Serengeti Diamond Drilling Campaigns

10.1.2.1 2006 A

In the summer of 2006 five diamond drillholes (K-06-01 to K-06-05, 660 m) were drilled to follow-up on an IP anomaly. These holes confirmed the copper grade of the previously known mineralization and identified a new zone some distance to the north of the South Zone.

10.1.2.2 2006 B

In November and December 2006, five diamond drillholes (1,215 m) were drilled in the vicinity of hole K-06-04, resulting in the discovery hole for the Central Zone, K-06-09 (0.69% Cu and 0.54g/t Au over 111 m).

10.1.2.3 2007-2008

Subsequent to the discovery of the Central Zone deposit in 2006, Serengeti initiated the third phase of the diamond drill program to define the new deposit. An all-weather, 30-man-camp was constructed in March 2007. Coast Mountain Geological Ltd. (CMG), a Vancouver-based geological consulting firm, was contracted to manage the drill project. Diamond drilling was carried out by Cyr Drilling International Ltd. of Winnipeg, Manitoba.

The Phase III drill program on the Kwanika property was conducted from March 2007 to August 2008. During this period, a total of 113 diamond drillholes, with an aggregate length of 53,615 m, were drilled on the property. These drillholes were primarily designed to delineate the mineralization in the Central Zone, explore the South Zone, as well as to test geophysical anomalies and possible extensions to the Central Zone mineralization.

Examples of significant drill intersections encountered in this phase of Central Zone drilling include K-07-15 (0.60% Cu and 0.72 g/t Au over 328 m) and K-08-113 (0.76% Cu and 1.39 g/t Au over 279 m). The significant grades and widths of copper and gold mineralization encountered confirmed the existence of a previously unknown porphyry copper-gold deposit.

The South Zone drilling campaign during 2007 and 2008 comprised 18 diamond drillholes for an aggregate length of 5,582 m. Several holes in the South Zone encountered a strongly mineralized copper-gold-molybdenite-silver porphyry system that had not been fully recognized by past exploration Examples of drill intersections include K-08-110 (0.27% Cu, 0.24 g/t Au, and 0.007% Mo over 240 m) and K-08-116 (0.39% Cu, 0.10g/t Au, and 0.013% Mo over 114 m).

10.1.2.4 2009

This phase of drilling was conducted from June to September 2009. During this period, a total of 17 diamond drillholes were completed on the property with an aggregate length of 6,249 m. This phase of exploration was primarily designed to follow-up several encouraging intersections obtained during 2008 drilling in the underexplored South Zone area. Significant drill intersections encountered included:

- K-09-124 (0.41% Cu, 0.05 g/t Au, and 0.019% Mo over 212 m)
- K-09-126 (0.51% Cu, 0.14 g/t Au, and 0.024% Mo over 150 m)

10.1.2.5 2010

The Phase V drill program on the Kwanika property was conducted from June to August 2010. During this period, a total of 28 diamond drillholes were completed on the property with an aggregate length of 7,619 m. This phase of exploration consisted of step-out drilling intended to expand the existing South Zone resource reported in March 2010. A series of infill drillholes were also completed in order to gain further understanding of the mineralization associated with the West Fault. The Phase V drilling was successful in both expanding the mineralized envelope to the north of the historical resource area of the South Zone deposit and adding important geological information to the exploration model.

10.1.2.6 2011

From June to July of 2011 a total of 5 drillholes were completed with an aggregate length of 1,727 m. This phase of exploration was carried out to test IP-chargeability and Ah-horizon soil exploration targets to the east and northeast of the Central Zone.

10.1.2.7 2012

The Phase VII drilling program was completed in August to September of 2012. During this period, a total of 4 drillholes were completed to an aggregate length of 1,472 m. Holes K-12-174 to K-12-176 tested IP-chargeability targets to the north of the Central Zone deposit. One additional drillhole was drilled at the south end of the property to test a deep IP-chargeability anomaly. Three line-km of IP was also completed in 2012 to test the existence of a chargeability anomaly to the east of the Central Zone resource area.

10.1.2.8 2016

This drilling campaign took place from July to August of 2016 during a joint exploration program funded by Daewoo Minerals Canada Corporation. A total of three deep drillholes were completed with an aggregate length of 2,584 m to test the deep roots of the Central Zone as well as an IP-chargeability anomaly to the north of the Central Zone. Hole K-16-177 penetrated the Central Zone producing significant results within the deposit. Highlights included:

- K-16-177 (0.79% Cu, 0.91 g/t Au over 385 m)

K-16-179 tested the northern deep extent of the Central Zone and showed significant grade at depth indicating the potential for further deep exploration. K-16-178 tested the northern deep chargeability anomaly and intersected significant lengths of highly altered andesite with moderate mineralization.

10.1.2.9 2018

This drilling campaign took place from June to September of 2018. A total of 20 drillholes were completed with a total length of 7,286 m to support detailed mine design and resource upgrading at the Kwanika Central Zone. Drill core was oriented with a Reflex ACT III tool and retrieved with split-tube core barrels to enable comprehensive geotechnical data capture for detailed underground and open pit mine engineering design. Included in the 2018 drill program were three holes to test foundation characteristics for potential TSF options (drillholes K-TSF-01, -02, -03). Additionally, downhole hydraulic testing was completed, and vibrating wire piezometres and monitoring wells were installed in 9 of the 21 drillholes to gather hydrogeological data. Holes K-18-180 to K-18-183 and K-18-187 penetrated the Central Zone producing significant results within the deposit. Highlights included:

- K-18-180 (0.64% Cu, 0.80 g/t Au over 514 m)
- K-18-181 (0.52% Cu, 0.37 g/t Au over 439 m)
- K-18-182 (0.66% Cu, 0.80 g/t Au over 500 m)
- K-18-183 (0.45% Cu, 0.73 g/t Au over 312 m)
- K-18-187 (0.59% Cu, 0.66 g/t Au over 226 m)

10.1.2.10 2020

From early August to mid-October 4,355 m of diamond drilling in 9 holes tested five exploration and resource expansion target areas. Boart Longyear's TruCore tool was used to orient the core. The first target area, Central Zone North, was considered prospective based on a previous (2016) deep-penetrating IP profile and broad anomalous Au intercept in K-16-176. The second target area, Central Fault South, was chosen to test the southern extent of high-grade Au and Cu results from K-18-190 and K-07-23. Target Area 3 was below the proposed underground shape and into the Pinchi fault at depth to the west to incrementally expand the resource at depth and westward. Target area 4 was the shallow wedge abutting the 'West Fault' in the South Zone. Finally, a target to the southeast of the Central Zone, south of the Pinchi fault, comprises a ZTEM anomaly similar in characteristics to the Central Zone.

10.1.3 NorthWest Copper Drilling Campaigns

This program took place between May and September of 2021, with 22 diamond drillholes with a total length of 9,505 m. Drill core was oriented using the Reflex ACT III tool. The goal of this drill program was to expand high-grade zones and improve grade within previously defined Central Zone resource. Additionally, it was designed to gather information to better understand the control on high-grade mineralization within the Central Zone deposit. Furthermore, two holes were drilled in the South Zone area testing for mineralization outside of the historic mineral resource. Highlights include.

- K-21-217 (2.00% Cu, 1.21 g/t Au over 235.45 m) – Central Zone
- K-21-223 (0.46% Cu, 0.05 g/t Au over 136.75 m) – South Zone

10.2 Stardust

Historical drilling on the property as outlined in Section 6 has been described and in previous Technical Reports (Simpson, 2010 & Simpson 2017).

Sun Metals, predecessor company to NorthWest Copper, has completed 3 drilling programs between 2018 and the end of 2020. The vast majority of this drilling has been in the CCS zone.

10.2.1 Historical Drilling

Prior to 1991, drill records for the property are missing or incomplete. Written accounts indicate that at least 16 holes were completed between 1966 and 1980 by Takla Silver Mines, Zapata Granby, and Noranda. Locations for these holes are uncertain or approximate and they have not used in the Mineral Resource estimation.

Statistics for the drilling completed over the entire property since 1991 are presented in Table 10-1.

Table 10-1: Stardust Drilling Summary by Year (1991-2020)

Year	Operator	Drillholes	Drilling (m)
1991	Alpha Gold	11	988
1992	Alpha Gold	30	1,520
1997	Teck/Alpha	16	3,063
1998	Teck/Alpha	14	1,105
1999	Alpha Gold	18	3,050
2000	Alpha Gold	29	4,680
2001	Alpha Gold	18	5,609
2002	Alpha Gold	19	7,790
2003	Alpha Gold	42	7,908
2004	Alpha Gold	21	6,010
2005	Alpha Gold	17	5,153
2006	Alpha Gold	56	9,909
2007	Alpha Gold	34	8,898
2008	Alpha Gold	5	2,140
2009	Alpha Gold	17	6,367
2010	Alpha Gold	14	3,9870
2017	Lorraine Copper	3	344
2018	Sun Metals	23	6,877
2019	Sun Metals	28	14,024
2020	Sun Metals	16	11,975
2021	NorthWest Copper	3	1,666
	Total	**434**	**101,088**

Note: total may differ due to rounding.

10.2.2 2018 Drilling

The 2018 diamond drill program began on August 3rd and was completed on September 27th. Drilling was conducted by Matrix Diamond Drilling of Kamloops, B.C. using two Zinex A5 skid mounted drills. A total of 23 bore holes were drilled from 15 sites, for a sum of 6,877 m. All core drilled was NQ diameter. Drill site locations are shown in Table 10-2 and Figure 10-3 Stardust 2018 Drillhole Locations.

A total of 1.1 km of new road was constructed to connect new drill pads to existing roads. Minor repairs of existing roads were also carried out. Road construction and repair was carried out by Gleyzay Holdings Ltd. using various excavators and bulldozers.

Table 10-2: Stardust 2018 Drillhole Locations.

Hole ID	UTM East	UTM North	Elevation (m)	Azimuth (⁰)	Dip (⁰)	Length (m)
DDH18-SD-406	346728	6161707	1408	65	-50	208
DDH18-SD-407	346728	6161707	1408	65	-60	224
DDH18-SD-408	346759	6161720	1404	65	-50	154
DDH18-SD-409	346705	6161794	1382	61	-50	176
DDH18-SD-410	346705	6161794	1382	60	-65	186
DDH18-SD-411	346849	6161847	1361	79	-60	374
DDH18-SD-Abandoned	347353	6161755	1313	70	-50	40
DDH18-SD-412	347349	6161753	1307	70	-53	283
DDH18-SD-413	346854	6162159	1359	102	-64	432
DDH18-SD-414	347348	6161753	1306	80	-70	272
DDH18-SD-415	347532	6161650	1358	80	-50	298
DDH18-SD-416	346855	6162158	1359	102	-70	463
DDH18-SD-417	347634	6161529	1372	80	-50	115
DDH18-SD-418	346905	6161482	1463	70	-50	283
DDH18-SD-419	346905	6161482	1463	70	-70	337
DDH18-SD-420	345996	6162579	1588	90	-55	502
DDH18-SD-421	346766	6162123	1362	73	-68	718
DDH18-SD-422	346893	6162248	1391	76	-50	307
DDH18-SD-423	346891	6162239	1390	76	-65	348
DDH18-SD-424	346879	6162282	1397	76	-58	341
DDH18-SD-425	346185	6161881	1405	253	-50	244
DDH18-SD-426	346185	6161880	1405	231	-49	234
DDH18-SD-427	346878	6162282	1397	70	-66	335

Note: total may differ due to rounding

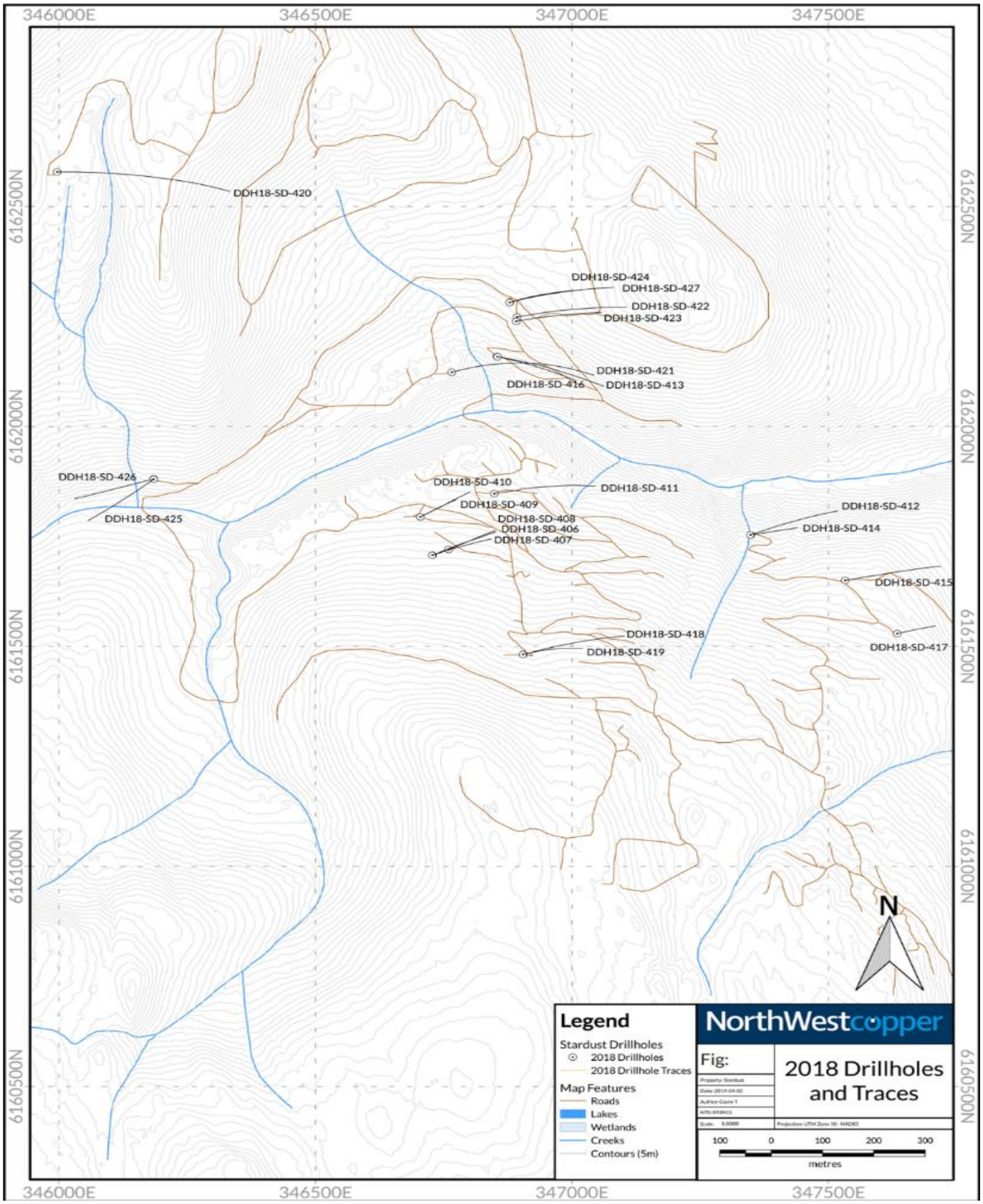
Drilling targeted copper-gold-silver-zinc-lead mineralization at the Canyon Creek Skarn, Glover Stock, and GD Zones as well as VTEM geophysical targets identified as Anomalies A, B, and C.

Drilling results from the 2018 season show similar grade and width when compared to historical drilling with the expectation of the DDH18-SD-421 which encountered a much longer massive sulphide intercept than previous drilling. This intercept has been termed the ‘421 Zone’.

Three different holes were drilled in the western part of the property and encountered thick sections of stratigraphy that is interpreted to be above the prospective carbonate package. This suggests that the geology is plunging to the north and potential for covered carbonate stratigraphy closer to surface in this corridor increases to the south.

A list of significant intercepts is shown in Figure 10-3.

Figure 10-3: Stardust 2018 Drillhole Locations



Source: NorthWest Copper, 2018.

Table 10-3: Significant Intercepts - Stardust 2018 Drill Program

Hole	From	To	Interval	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Lead (%)
DDH18-SD-411	174.70	189.10	14.40	1.32	1.03	22.9	2.12	-
incl	178.20	183.90	5.70	1.57	1.38	33.1	5.20	-
DDH18-SD-411	226.75	228.90	2.15	3.81	0.75	498.4	23.31	3.71
DDH18-SD-412	42.75	50.40	7.65	0.03	1.31	62.3	0.78	0.45
DDH18-SD-413	232.50	238.00	5.50	1.72	0.93	29.1	0.01	-
DDH18-SD-413	245.00	246.00	1.00	0.02	2.52	11.1	0.09	0.07
DDH18-SD-414	63.30	63.90	0.60	0.05	0.59	382.8	21.22	3.60
DDH18-SD-415	34.60	34.90	0.30	0.01	4.23	3.2	0.04	-
DDH18-SD-415	44.60	46.80	2.20	0.28	5.25	16.4	3.79	0.21
DDH18-SD-415	55.90	60.50	4.60	0.09	4.17	34.5	1.60	0.09
DDH18-SD-416	281.70	282.70	1.00	1.70	1.25	27.2	0.01	-
DDH18-SD-417	35.70	39.00	3.30	0.01	0.21	3.9	1.35	0.04
DDH18-SD-417	50.50	57.80	7.30	0.04	0.48	7.7	7.42	0.06
DDH18-SD-418	218.80	220.20	1.40	0.03	0.88	9.5	4.60	0.02
DDH18-SD-418	224.90	225.60	0.70	0.09	0.08	6.7	25.67	-
DDH18-SD-418	233.10	234.80	1.70	0.05	4.37	15.4	4.39	0.12
DDH18-SD-418	242.80	243.20	0.40	0.03	0.11	7.6	11.79	0.01
DDH18-SD-418	249.10	252.20	3.10	0.10	5.05	55.3	5.23	0.18
DDH18-SD-421	433.80	435.00	1.20	1.07	0.16	17.4	0.01	-
DDH18-SD-421	506.60	507.30	0.70	1.29	1.45	22.3	0.02	-
DDH18-SD-421	517.00	617.00	100.00	2.51	3.03	52.5	0.41	-
incl	539.80	617.00	77.20	3.11	3.74	64.9	0.53	-
incl	539.80	576.30	36.50	3.89	4.47	84.6	1.06	-
incl	587.90	617.00	29.10	3.35	4.30	65.7	0.07	-
DDH18-SD-424	74.50	76.00	1.50	1.67	6.70	27.0	0.01	-
DDH18-SD-424	282.70	283.30	0.60	10.00	5.17	265.3	0.08	-
DDH18-SD-425	50.80	51.35	0.55	0.15	0.58	54.1	6.23	0.43
DDH18-SD-426	143.50	144.90	1.40	0.37	1.90	25.3	3.08	0.05
DDH18-SD-427	81.20	81.80	0.60	1.12	1.96	16.1	0.01	-
DDH18-SD-427	145.50	147.20	1.70	1.01	1.63	11.8	0.01	-

10.2.3 2019 Drilling

The 2019 diamond drill program began on May 23rd and was completed on December 15th. Drilling was conducted by Matrix Diamond Drilling of Kamloops, B.C. primarily using two Zinex A5 skid mounted drills, with a third A5 drill mobilizing in November. Drilling targeted copper-gold-silver-zinc mineralized skarn at the 421 Zone. A total of 28 bore holes were drilled from 7 sites, for a sum of 14,024 m. TECH Directional Services of Sudbury, Ontario provided directional drilling and bore hole surveying services utilizing the DeviDrill Directional Core Barrel system. Use of the directional drilling system allowed for deep targets to be hit with a high degree of precision. Core drilled was NQ diameter except in sections drilled using the DeviDrill system where AQ diameter core is recovered. Drill site locations are shown in Table 10-4 and Figure 10-4. Note: total may differ due to rounding.

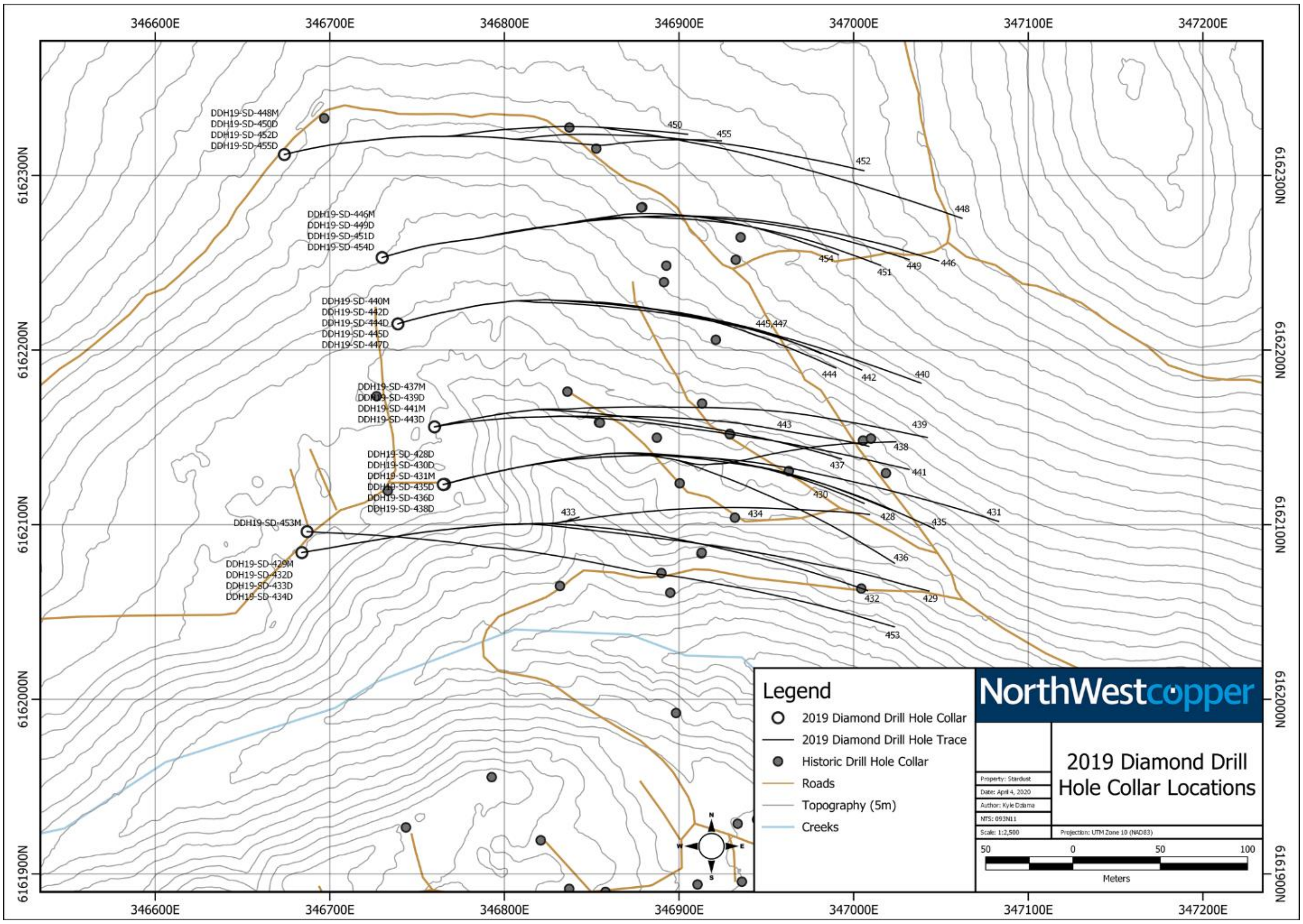
A new road of 0.8 km length was constructed to connect new drill pads to existing roads. Minor repairs of existing roads were also carried out. Road construction and repair was carried out by Gleyzay Holdings Ltd. of Takla Landing, B.C. using a Caterpillar 330 excavator.

Table 10-4: Stardust 2019 Drillhole Locations

Hole ID	UTM East	UTM North	Elevation (m)	Azimuth (⁰)	Dip (⁰)	Cut-off Depth (m)	Length EOH (m)
DDH19-SD-428D	346766	6162123	1362	76	-68	434.3	725
DDH19-SD-429M	346684	6162084	1368	80	-67.5	n/a	725
DDH19-SD-430D	346766	6162123	1362	76	-68	418.8	710
DDH19-SD-431M	346765	6162123	1362	76	-67	n/a	662
DDH19-SD-432D	346684	6162084	1368	80	-67.5	308.2	755
DDH19-SD-433D	346684	6162084	1368	80	-67.5	362.7	415
DDH19-SD-434D	346684	6162084	1368	80	-67.5	387.4	761
DDH19-SD-435D	346765	6162123	1362	76	-67	219.1	673
DDH19-SD-436D	346765	6162123	1362	76	-67	253.6	677
DDH19-SD-437M	346760	6162156	1368	75	-73	n/a	627
DDH19-SD-438D	346765	6162123	1362	76	-67	338.2	638
DDH19-SD-439D	346760	6162156	1368	75	-73	178.6	797
DDH19-SD-440M	346739	6162215	1380	80	-76	n/a	794
DDH19-SD-441M	346760	6162156	1368	80	-78	n/a	746
DDH19-SD-442D	346739	6162215	1380	80	-76	249.4	767
DDH19-SD-443D	346760	6162156	1368	80	-78	290.1	770
DDH19-SD-444D	346739	6162215	1380	80	-76	328.5	811
DDH19-SD-445D	346739	6162215	1380	80	-76	314.5	806
DDH19-SD-446M	346730	6162253	1386	73	-75	n/a	811
DDH19-SD-447D	346739	6162215	1380	80	-76	761.5	884
DDH19-SD-448M	346674	6162312	1405	76	-73.5	n/a	905
DDH19-SD-449D	346730	6162253	1386	73	-75	415.8	860
DDH19-SD-450D	346674	6162312	1405	76	-73.5	527.5	639
DDH19-SD-451D	346730	6162253	1368	73	-75	223.8	900
DDH19-SD-452D	346674	6162312	1405	76	-73.5	316.0	929
DDH19-SD-453M	346687	6162096	1368	94	-65	n/a	670
DDH19-SD-454D	346730	6162253	1368	73	-75	444.4	963
DDH19-SD-455D	346674	6162312	1405	76	-73.5	440.2	813

Note: total may differ due to rounding.

Figure 10-4: Stardust 2019 Drillhole Locations



Source: NorthWest Copper, 2019.

Drilling results from the 2019 season confirmed the presence of a large, mineralized skarn system at depth in the 421 Zone. Seventeen diamond drillholes intersected significant copper-gold-silver-zinc mineralization. These results expanded the zone in all directions from mineralization previously intersecting in DHH18-SD-421.

Mineralization is hosted in skarn alteration within a pre-mineral parasitic anticline fold hinge of a broad anticline along the contact of overlying siliciclastic sedimentary rocks and underlying carbonates. The trend of the fold hinge is interpreted to be plunging down at 20° – 30° to the north-northwest. Intensity and thickness of skarn replacement appears to be increasing to the north and down plunge, this implies the source of the fluids in the system are to the north and/or below the 421 Zone. Additionally, the decrease in thickness of mineralized intercepts on sections 6162275N and 6162325N suggests an east – west trending fault(s) may down drop to the north offsetting mineralization.

DDH19-SD-453M is the most southerly test of the 421 Zone and intersected strong copper-gold-silver mineralization. This indicates mineralization remains open in the south as well as both up and downdip in this area.

Results from DDH19-SD-452D show that high-grade copper-gold-silver mineralization is present and open in this northerly part of the system.

A list of significant intercepts is shown in Table 10-5.

Table 10-5: Significant Intercepts - Stardust 2019 Drill Program

Hole	From (m)	To (m)	Interval (m)	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)
DDH19-SD-428D	493.45	635.8	142.35	1.22	1.28	21.8	0.41
incl.	562.8	595.0	32.2	2.47	2.37	47.4	1.61
incl.	604.95	619.05	14.1	3.45	4.12	57.9	0.44
DDH19-SD-429M	564.0	654.05	90.05	1.08	1.4	21.6	0.22
incl.	586.5	593.0	6.5	4.61	7.05	60.2	1.68
incl.	649.45	654.05	4.6	2.96	5.31	131.8	1.65
DDH19-SD-430D	490.6	512.6	22.0	1.53	1.02	24.6	0.03
DDH19-SD-430D	546.0	653.0	107.0	1.64	1.77	28.6	0.03
incl.	572.2	630.3	58.1	2.49	2.61	44.3	0.04
DDH19-SD-432D	680.15	691.95	11.8	0.61	0.54	11.1	0.01
DDH19-SD-436D	502.6	548.15	45.55	1.44	1.18	27	0.04
incl.	542.3	548.15	5.85	5.13	3.78	91	0.18
DDH19-SD-436D	598.4	623.25	24.85	3.13	4.85	93.5	0.28
incl.	609.2	618.2	9.0	6.04	9.13	183.7	0.6
DDH19-SD-437M	537.6	624.0	86.4	1.65	1.56	28.8	0.28
incl.	585.7	607.0	21.3	3.13	2.14	51.4	1.08
DDH19-SD-438D	564.4	572.9	8.5	3.09	3.47	72	0.08
DDH19-SD-438D	594.0	597.05	3.05	1.08	1.26	21.8	0.02
DDH19-SD-439D	637.0	657.5	20.5	1.17	0.96	20.4	0.01
DDH19-SD-439D	714.5	724.45	9.95	0.78	0.7	97.1	0.28
DDH19-SD-440M	582.0	591.0	9.0	1.26	1.91	32.8	0.01
DDH19-SD-440M	708.9	724.8	15.9	2.38	2.68	66.6	0.1
DDH19-SD-441M	609.25	650.8	41.55	2.33	2.73	44.3	0.07
incl.	609.25	620.3	11.05	3.35	3.88	60.7	0.14
incl.	639.5	650.8	11.3	3.94	4.58	79.2	0.11
DDH19-SD-442D	669.75	720.7	50.95	0.64	0.67	10.6	0.01
incl.	669.75	693.2	23.45	0.92	0.92	14.4	0.01
DDH19-SD-443D	678.3	695.3	17.0	1.17	1.05	19.2	0.01
DDH19-SD-444D	735.0	738.2	3.2	1.65	1.3	29.4	0.01
DDH19-SD-444D	762.0	772.95	10.95	3.19	3.59	58.1	0.07
DDH19-SD-451D	807.0	810.7	3.7	1.64	1.36	25.8	0.01
DDH19-SD-452D	866.0	869.0	3.0	3.25	4.32	70.1	0.05
DDH19-SD-453M	540.7	567.0	26.3	1.45	1.48	22.2	0.01
incl.	553.8	557.4	3.6	3.98	3.45	66.6	0.02
DDH19-SD-453M	594.0	601.2	7.2	2.1	1.41	33.4	0.01

10.2.4 2020 Drilling

The 2020 diamond drill program began on June 26th and was completed on September 21st. Drilling was conducted by Matrix Diamond Drilling of Kamloops, B.C. using three Zinex A5 skid mounted drills. Drilling targeted copper-gold-silver-zinc mineralized skarn at the Canyon Creek, East, and 421 Zones. A total of 17 bore holes were drilled from 10 sites, for a sum of 11,975.4 m. TECH Directional Services of Sudbury, Ontario provided directional drilling and bore hole surveying services utilizing the DeviDrill Directional Core Barrel system. Core drilled was NQ diameter except in sections drilled using the DeviDrill system where AQ diameter core is recovered. Drill site locations are shown in Table 10-6 and Figure 10-5. A list of significant intercepts is presented in Table 10-7.

Around 0.1 km of new road was constructed to connect new drill pads to existing roads. Minor repairs of existing roads were also carried out. Road construction and repair was carried out by Gleyzay Holdings Ltd. of Takla Landing, B.C. using a Caterpillar 330 excavator.

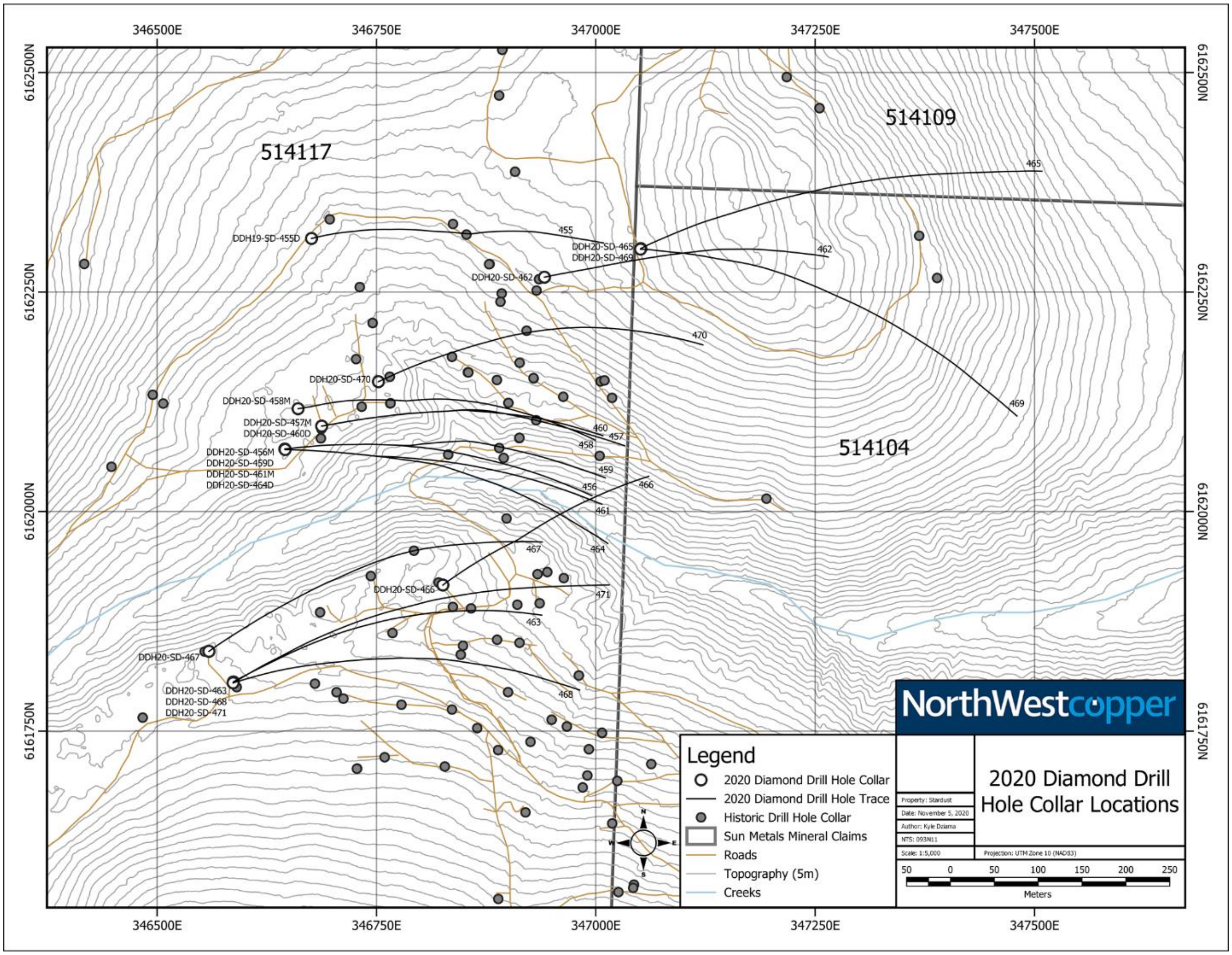
Stardust 2020 Drillhole Locations. Note: total may differ due to rounding.

Table 10-6: Stardust 2020 Drillhole Locations.

Hole ID	UTM East	UTM North	Elevation (m)	Azimuth (⁰)	Dip (⁰)	Cut-off Depth (m)	Length EOH (m)
DDH19-SD-455D	346676	6162311	1405	76	-73.5	440.2	1089
DDH20-SD-456M	346646	6162071	1366	82.5	-64	n/a	692
DDH20-SD-457M	346688	6162097	1365	81	-66	n/a	664
DDH20-SD-458M	346661	6162117	1370	80	-73.5	n/a	1038
DDH20-SD-459D	346646	6162071	1366	82.5	-64	289.4	741
DDH20-SD-460D	346688	6162097	1365	81	-66	324.0	710
DDH20-SD-461M	346646	6162071	1366	91	-59	n/a	647
DDH20-SD-462	346941	6162267	1404	60	-66	n/a	804
DDH20-SD-463	346587	6161806	1383	60	-66	n/a	893
DDH20-SD-464D	346646	6162071	1366	81	-64	194.0	707
DDH20-SD-465	347051	6162299	1430	60	-63	n/a	856
DDH20-SD-466	346826	6161916	1360	52	-60	n/a	497
DDH20-SD-467	346559	6161841	1381	53	-61	n/a	815
DDH20-SD-468	346587	6161805	1383	76	-61	n/a	833
DDH20-SD-469	347051	6162299	1430	85	-69	n/a	993
DDH20-SD-470	346752	6162148	1365	64	-61	n/a	806
DDH20-SD-471	346587	6161805	1383	58	-61	n/a	812

Note: total may differ due to rounding.

Figure 10-5: Stardust 2020 Drillhole Locations



Source: NorthWest Copper, 2020.

Table 10-7: Significant Intercepts - Stardust 2020 Drill Program

Hole ID	From (m)	To (m)	Interval (m)	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)
DDH19-SD-455D	903.8	905.8	2.0	1.05	1.26	26.5	0.02
DDH20-SD-456M	635.3	654.9	19.6	0.59	0.55	13.3	0.02
incl.	635.3	638.2	2.9	2.15	1.78	49.2	0.04
DDH20-SD-457M	505.7	549.7	44.0	1.57	1.08	28.2	0.01
incl.	535.8	549.7	13.9	3.05	2.12	53.6	0.01
DDH20-SD-459D	675.0	679.8	4.8	0.92	0.81	16.2	0.01
DDH20-SD-460D	588.0	628.4	40.4	1.74	1.41	26.6	0.01
incl.	588.0	604.0	16.0	3.12	2.55	48.2	0.01
DDH20-SD-461M	493.4	498.45	5.05	0.90	0.74	11.3	0.02
DDH20-SD-463	823.8	833.4	9.6	0.58	0.36	11.0	0.01

Hole ID	From (m)	To (m)	Interval (m)	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)
DDH20-SD-464D	499.0	506.3	7.3	1.18	1.07	14.4	0.02
DDH20-SD-464D	614.25	618.7	4.45	5.58	5.99	190.5	0.12
DDH20-SD-466	373.35	390.8	17.45	1.37	1.70	39.7	0.03
incl.	384.35	389.85	5.5	3.02	3.83	87.2	0.07
DDH20-SD-467	775.85	779.2	3.35	0.78	0.85	20.3	0.03
DDH20-SD-468	614.0	635.0	21.0	0.45	0.28	4.9	0.01
DDH20-SD-468	657.1	658.85	1.75	1.28	0.60	13.5	0.01
DDH20-SD-469	236.75	247.2	10.45	0.53	0.44	40.2	0.02
incl.	238.65	244.05	5.4	0.88	0.58	66.0	0.03

The 2020 drilling combined with Sun Metals previous drilling in 2017-2019, as well as historical drilling on the property was used to re-interpret the geological model and mineralized domains. The structural framework that controls mineralization is currently interpreted to be a series of parasitic folds and thrust faults formed where faults and associated fault propagation folds create the architecture and plumbing system for the skarn alteration, fluid flow, and base metal mineral deposition. Zone thickening is seen at the intersection lineation between the faults and certain stratigraphic horizons. Dilatational offset within the structures creates northerly plunging mineralized material chutes within the larger mineralized structure. The most prospective stratigraphic horizon for hosting the high-grade zones is the carbonate unit that is deposited stratigraphically below the clastic sediment unit and above the limestone clast tuff unit.

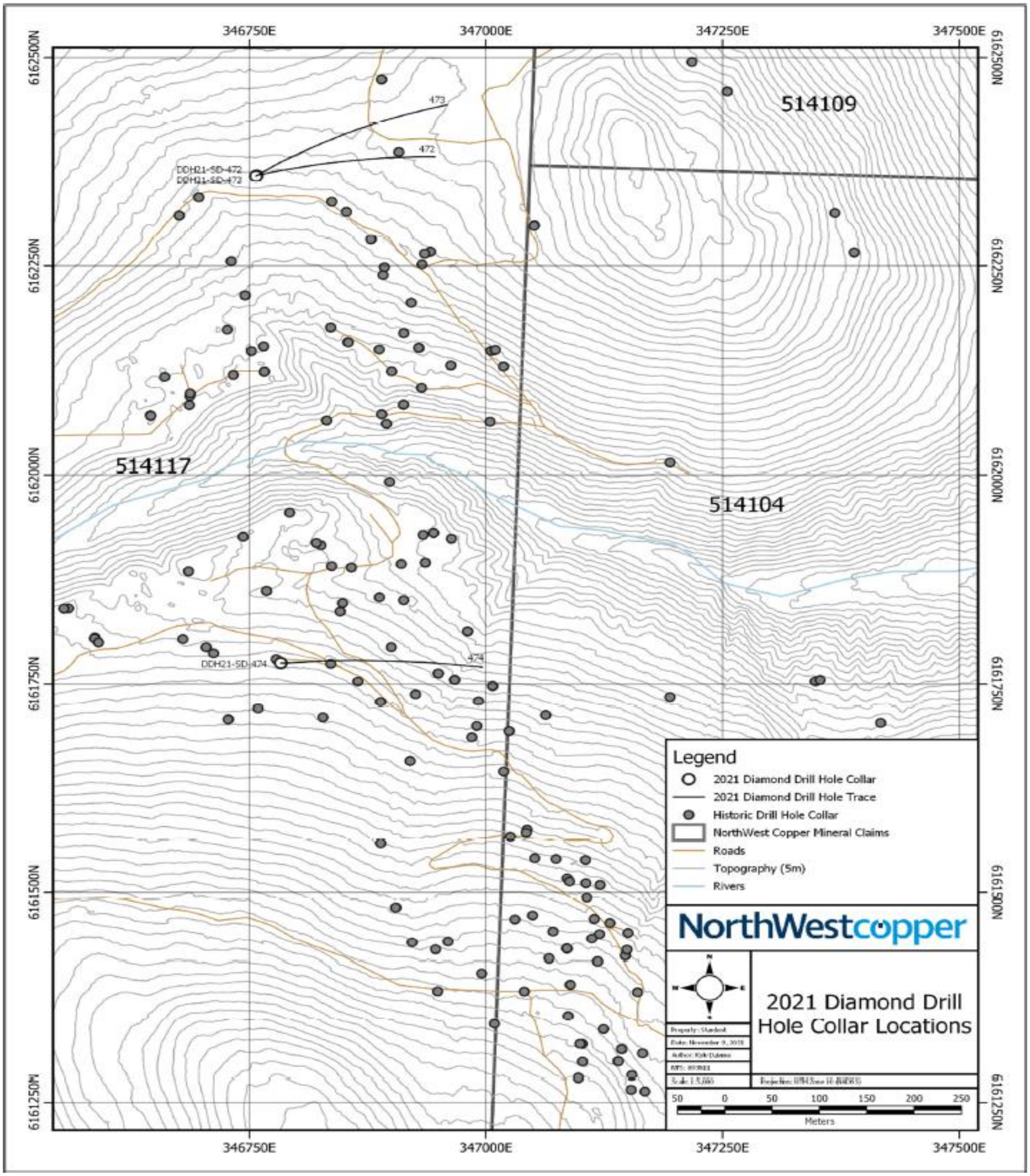
10.2.5 2021 Drilling

The 2021 diamond drill program began on September 5th and was completed on September 26th. Drilling was conducted by Matrix Diamond Drilling of Kamloops, B.C. using one Zinex A5 skid mounted drill. Drilling targeted copper-gold-silver-zinc mineralized skarn at the Canyon Creek zone. A total of 3 bore holes were drilled from 2 sites, for a sum of 1,665.5 m. Core drilled was NQ diameter. Drill site locations are shown in Table 10-8 and Figure 10-6. The core is currently stacked beside the Stardust core shack.

Around 0.1 km of new road was constructed to connect new drill pads to existing roads. Minor repairs of existing roads were also carried out. Road construction and repair was carried out by Kazaco Contracting Ltd. of Williams Lake, B.C. using a Komatsu 210 excavator. Current drill roads are shown on Figure 10.6.

All drill core samples were submitted to Bureau Veritas Mineral Laboratories in Vancouver, B.C. for analysis. Detailed sampling procedures are presented in Section 11. A list of significant intercepts is shown in Table 10-9.

Figure 10-6: Stardust 2021 Drillhole Locations



Source: NorthWest Copper, 2021.

Table 10-8: Stardust 2021 Drillhole Locations

Hole ID	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)	Length EOH (m)	Section
DDH21-SD-472	346758	6162359	1407	75	-68	478	6162375N
DDH21-SD-473	346757	6162358	1407	55	-73	672	6162375NW
DDH21-SD-474	346783	6161775	1393	86	-68	515.5	6161750N

Table 10-9: Significant Intercepts - Stardust 2021 Drill Program

Hole ID	From (m)	To (m)	Interval (m)	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)
DDH19-SD-455D	903.8	905.8	2	1.05	1.26	26.5	0.02
DDH20-SD-456M	635.3	654.9	19.6	0.59	0.55	13.3	0.02
incl.	635.3	638.2	2.9	2.15	1.78	49.2	0.04
DDH20-SD-457M	505.7	549.7	44	1.57	1.08	28.2	0.01
incl.	535.8	549.7	13.9	3.05	2.12	53.6	0.01
DDH20-SD-458M	No Significant Value						
DDH20-SD-459D	675	679.8	4.8	0.92	0.81	16.2	0.01
DDH20-SD-460D	588	628.4	40.4	1.74	1.41	26.6	0.01
incl.	588	604	16	3.12	2.55	48.2	0.01
DDH20-SD-461M	493.4	498.45	5.05	0.9	0.74	11.3	0.02
DDH20-SD-462	No Significant Value						
DDH20-SD-463	823.8	833.4	9.6	0.58	0.36	11	0.01
DDH20-SD-464D	499	506.3	7.3	1.18	1.07	14.4	0.02
DDH20-SD-464D	614.25	618.7	4.45	5.58	5.99	190.5	0.12
DDH20-SD-465	No Significant Value						
DDH20-SD-466	373.35	390.8	17.45	1.37	1.7	39.7	0.03
incl.	384.35	389.85	5.5	3.02	3.83	87.2	0.07
DDH20-SD-467	775.85	779.2	3.35	0.78	0.85	20.3	0.03
DDH20-SD-468	614	635	21	0.45	0.28	4.9	0.01
DDH20-SD-468	657.1	658.85	1.75	1.28	0.6	13.5	0.01
DDH20-SD-469	236.75	247.2	10.45	0.53	0.44	40.2	0.02
incl.	238.65	244.05	5.4	0.88	0.58	66	0.03
DDH20-SD-470	No Significant Value						
DDH20-SD-471	No Significant Value						

10.2.6 Core Recovery

Core recovery for the Sun Metals drill programs was good to excellent with an overall average of 95% and a median value 98%. Areas of poor or no recovery normally occurred in fault zones and small karst cavities.

10.2.7 Drillhole Location Surveys

During the Sun Metals drill programs, drillhole collars were surveyed using a Real-Time Kinematic and Differential GPS system. Elevations were derived from the LiDAR survey data described in Section 9.2.1.

10.2.8 Downhole Surveys

During the Sun Metals drill programs, downhole surveys were generally taken at intervals between 10 and 30 m, although a number of holes used 3 m intervals. The average spacing was 20 m.

Downhole survey instruments used were a Reflex EZ-GYRO and an Axis C-Gyro. The C-Gyro was used for directional drilling by TECH Directional Services.

10.2.9 Sample Length vs. True Thickness

Due to the steeply-dipping orientation of the mineralized zones and the limitations of surface drilling, none of the drill intercepts approximate the true thickness. True thickness must be calculated for each intercept based on the angle of the drillhole to the specified zone.

10.2.10 Comments

Drilling methods and drillhole design are suitable for construction of a Mineral Resource model for the CCS Zone.

Fault zones and small karst cavities have been intersected during the drill programs resulting in loss of recovery, but nothing that could materially impact the accuracy and reliability of the results.

True thickness must be calculated for each intercept based on the angle of the drillhole to the specified zone.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Kwanika

11.1.1 Core Logging

The drill programs at the Kwanika property were managed by Coast Mountain Geological (CMG) from 2006 to 2008, by Serengeti staff from 2009 to 2020 and by NorthWest Copper staff in 2021. The methodology for core handling and sampling since 2006 is described below.

The core was transported from the drills to the camp at each drilling shift change, once in the morning and once in the evening. Each morning, the core drilled during the previous day was quick-logged by a geologist. The quick log involved a brief description of lithology, alteration, and mineralogy, as well as a description of any significant structural characteristics. An approximate copper grade based on visual inspection of mineralization was assigned to each interval. Since February 2008, a handheld X-ray fluorescence (XRF) tool was used to aid in the initial grade estimation.

Once quick-logged, the core was stacked on-site pending detailed logging. The logging includes a description of the lithology, alteration, structural features, mineralogy, and veining. Sample intervals were divided based on contacts between these characteristics, to a nominal length of two metres. The overlying post-mineral sedimentary rocks, encountered at the top of many of the holes drilled in the Central Zone were not sampled unless copper mineralization was observed. Once the sampled intervals were established, each interval was assigned a unique sample number.

Geo-technicians determined the recovery, rock quality designation (RQD), specific gravity, magnetic susceptibility and conductivity of the rock as well as conducted point load testing on select intervals.

The Magnetic susceptibility and conductivity were determined using a multi-parameter probe. A reading was taken every 1.5 m directly on the core surface (every 1 m in 2021). Recovery and RQD were completed for the full length of the holes, whereas specific gravity and magnetic susceptibility were measured only for sampled intervals.

Structure related geotechnical data, were gathered from 2018 drill core. Drill core was oriented and retrieved with triple-tube core barrels to enable comprehensive geotechnical data capture for underground and open pit mine engineering design. The core was also oriented to record oriented structural data in 2020 and 2021, the difference being it was not triple-tubed.

11.1.2 Core Sampling

After logging, the core was split or cut under the supervision of project geologists. Diamond core saws were the preferred method, or a hydraulic splitter was used to split the core in zones observed by the geologists to be low grade during earlier drill campaigns by Serengeti. The diamond core saws used clean, un-recirculated water to aid in cutting, and were cleaned regularly to avoid contamination. The mechanical splitter was cleaned thoroughly after each sample was split.

Once split, half of the core was left in the core box for reference, and the other half was sent for analysis. Samples were placed in labelled plastic bags with the corresponding sample tag and sealed with zip ties or staples. Quality control samples, including standard reference materials and blanks, were also placed into a labelled plastic bag with a sample tag. These plastic bags were placed in numbered rice sacks, which were sealed by heavy duty zip ties or metal tamper-proof closures and given a numbered tamper-proof security tag.

Samples were transported via truck by a local third party expediting and freight company. To ensure that samples were not tampered with during transport to the laboratory, the number of each security tag and its associated rice sack number were recorded by the geologist at the Kwanika site. Prior to 2021, a list of each bag and its unique security tag number was forwarded to GDL/ACME/ACT/BV, which then confirmed that each security tag matched its correct rice sack. In 2021, sample bags and ID tags were recorded, tamper-proof metal closures were used to seal the bags, and the lab was asked to notify if any of the bags appeared to be tampered with. Samples were in NWST personnel possession until they were delivered to a licensed and bonded transportation company. No contractors, or non-NWST personnel delivered samples.

11.1.3 Core Preparation and Analysis

11.1.3.1 Sample Preparation and Assaying by Global Discovery Labs (2006 – 2009)

From 2006 to 2009 all assays from the Kwanika project were sent to independent lab Global Discovery Labs (GDL) in Vancouver, British Columbia. GDL did not have ISO accreditation but did participate in the Proficiency Testing Program for Mineral Analysis Laboratories (PTP-MAL). PTP-MAL is an ISO 9001:2000 accredited program that is operated by the Canadian Certified Reference Materials Project (CCRMP) and meets recognized international standards for proficiency testing providers.

Samples sent to GDL were passed through a two-stage crushing process reducing the material to 90% minus 2 mm in size. The crushed material was split in a Jones Riffle to a subsample measuring 250 g to 300 g. The samples were pulverized in a ring-and-puck mill to 95% passing a 150-mesh screen.

The shipped samples were divided into two groups: samples with an assumed grade less than 0.2% Cu and samples with an assumed grade of greater than 0.2% Cu, as determined by the project geologist. All samples were subject to aqua regia digestion and then run for 28 elements using Inductively Coupled Plasma (ICP) spectrometry (Package ICP-OES). Samples with greater than 2,000 ppm Cu or 100 ppb Au were rerun for Au, Cu, Pb, Zn and Fe by Atomic Absorption (AA). Dissolution of the samples for the base metal determinations was done using aqua regia, while for the gold it was aqua regia followed by 2, 6-Dimethyl-4-heptanone.

Samples assaying greater than 0.2 g/t Au in the ICP or AA analyses were rerun using fire assay and AA finish. These assays were carried out on a 30 g (one assay-ton) aliquot.

11.1.3.2 Sample Preparation and Assaying by Acme Labs (2009 – 2012)

From 2009 to 2012, sampling was carried out by independent lab Acme Labs which acquired GDL in July of 2009. Acme Laboratories held ISO 9001 accreditation during this time. The assay prep and processing remained the same from 2009-2012 after Acme took over GDL.

11.1.3.3 Sample Preparation and Assaying by Activation Labs (2016)

During the 2016 drilling program, independent lab Activation Labs of Kamloops, British Columbia was used to carry out assaying of the Kwanika project. Activation Labs is an ISO 17025 accredited laboratory.

Once samples were received at the lab they were weighed, and then crushed up to 90% passing 10 mesh, riffle split (250 g) and then pulverized to 95% passing minus 150 mesh. The pulverizer bowl was cleaned after each sample. Prepared samples were assayed for a suite of 38 elements including selenium by aqua regia digestions and ICP spectrometry. All Au analysis was carried out by 30 g fire assay and AA.

Samples greater than 2,500 ppm Cu were rerun by assay grade aqua regia digestion and ICP spectrometry. Au results greater than 3.0 g/t were rerun by 30 g fire assay and a gravimetric finish.

11.1.3.4 Sample Preparation and Assaying by Bureau Veritas Labs (2018-2021)

During the 2018-2021 drilling campaign, independent lab Bureau Veritas Mineral Laboratories (BV) out of Vancouver, British Colombia was used to carry out the assaying of the Kwanika project. Bureau Veritas is an ISO 17025 accredited laboratory.

At BV all rock samples were crushed, to 70% passing 2 mm, then split to 250 g samples and pulverized up to 85% passing 200 mesh. Split samples were assayed for Au using fire assay fusion with AAS finish. In 2018, samples were digested using aqua regia and analyzed with ICP- ES/MS for 34 elements. In 2020 and 2021, 4-acid digestion was utilized, with either 35 element ICP-ES or 45 element ICP-MS analysis.

11.1.4 Specific Gravity Data

Specific gravity data were collected using whole core measurements carried out on-site before core was sent for assay. This was performed using a water immersion method. The data were recorded in a density log within the drilling template or logged directly into online data base in the case of the 2021 drill program. Specific gravity has been collected since the 2007 drill program.

Specific gravity was determined by taking the weight of that sample in air and then the weight of the sample in water. The volume of the sample was determined by subtracting its weight in air from its weight in water. Specific gravity was found by dividing sample weight in air by its volume. A wax coating was not necessary for the core at Kwanika because it is not vuggy or particularly porous.

11.1.5 Quality Assurance and Quality Control

An independent assay Quality Assurance/Quality Control (QA/QC) program has been in place throughout the drilling campaigns carried out by Serengeti and NorthWest Copper since 2006. Control samples have included certified reference materials (CRMs), pulp blanks, and quarter core twin samples (field duplicates).

CRMs were prepared by CDN Resource Labs Ltd. (CDN) of Langley, B.C. or by Ore Research & Exploration P/L in Australia. Most of the standards used are certified for both copper and gold values. Two standards are not certified for gold and are deemed “Provisional”. CRMs are used to assess analytical accuracy.

Blank material comprised packets of pulverized barren material. The 2020-2021 drilling campaign used a certified blank, also prepared by CDN. Pulp blanks are used to assess contamination during assaying. During 2021, a small number of coarse blanks (unmineralized garden stone) were used to assess contamination during preparation.

Twin samples were produced by cutting the initial core sample interval in half and leaving one half in the core box. The half to be sent to the laboratory for analyses was then quartered by cutting each piece in half again and putting one quarter of the core in one sample bag and the other quarter of the core in a separate sample bag. Twin samples are generally used to assess sampling precision and mineralization homogeneity. The term ‘duplicate’ is avoided since the original and twin-sample do not occupy the same spatial position.

A total of 3,534 quality control samples, plus 184 additional check assays were completed on the 37,921 primary samples from the 2006 to 2021 drilling campaigns, representing an approximate 10% insertion rate.

11.1.5.1 2020-2021 Drilling at Central and South Zones

A total of 653 QC samples comprising blanks, CRMs, and twin samples were inserted into the stream of drill core samples submitted or assay, for an insertion rate of around 13%. The QC summary is shown in Table 11-1.

Table 11-1: 2020-2021 Central and South Zone QC Summary

Sampling Program	Count	Insertion Rate (%)
Sample Count	5,199	
Pulp Blanks	223	4.3%
Coarse Blanks	24	0.5%
Certified Reference Materials	209	4.0%
Twin Samples	197	3.8%
Total QC Samples	653	12.6%

11.1.5.2 Certified Reference Materials

A total of 209 certified reference material samples were included in the QC samples from the 2020 to 2021 drilling at Central and South Zones. Twelve different CRMs were obtained from either CDN Resource Laboratories or Ore Research & Exploration and inserted into the sample stream. The CRM names and expected values for Cu and Au are shown in Table 11-2. Early on, problems were identified with the assay value results for material CDN-CGS-23 and use of this material was discontinued.

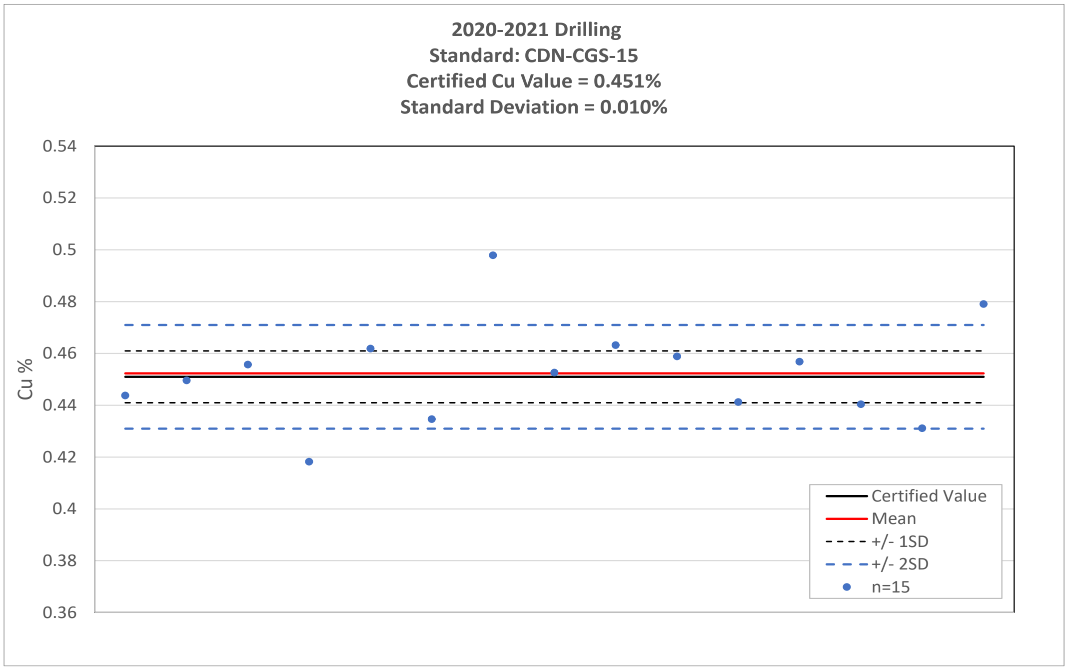
Table 11-2: 2020-2021 CRM Expected Values

Reference Material	Cu(%)	Au (g/t)	# Used
CDN-CGS-15	0.451	0.57	15
CDN-CGS-22	0.725	0.64	5
CDN-CGS-23	0.182	0.218*	1
CDN-CM-31	0.082	NA	35
CDN-CM-36	0.227	0.316	3
CDN-CM-38	0.681	0.942	22
CDN-CM-40	0.561	1.31	11
CDN-CM-42	0.529	0.576	4
CDN-CM-43	0.233	0.309	20
CDN-CM-7	0.445	0.427	6
OREAS 151b	0.180	0.065	67
OREAS 152b	0.377	0.134	20

An * indicates a provisional value

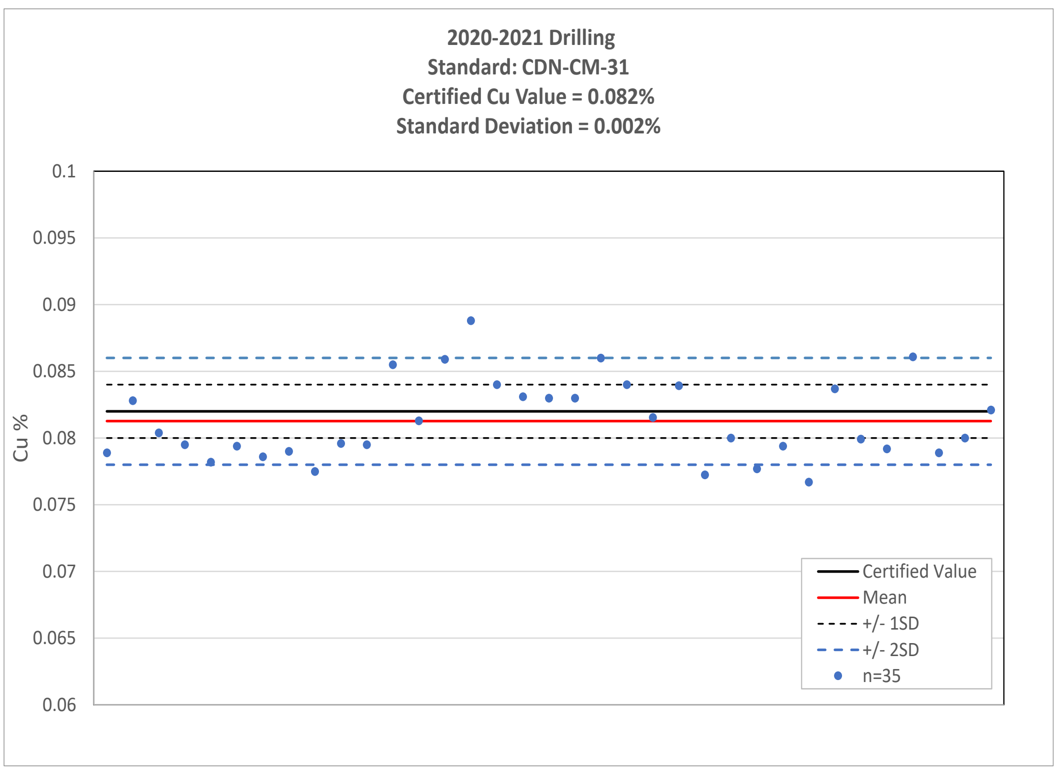
Representative control charts showing Cu assay values are presented in Figure 11-1 through Figure 11-6. Most standards show acceptable performance. Standard CDN-CM-40 shows two values far higher than the +2 standard deviation range, causing the mean to plot outside of this range. Poor performance of this standard was recognized in earlier drill programs, so its use should be discontinued. Standard OREAS151b shows many values that lie above the +2-standard deviation, with the mean being around 4% higher than the expected value, but still within the +/- 2SD range. Performance of this standard should be closely monitored.

Figure 11-1: Copper Control Chart for CDN-CGS-15



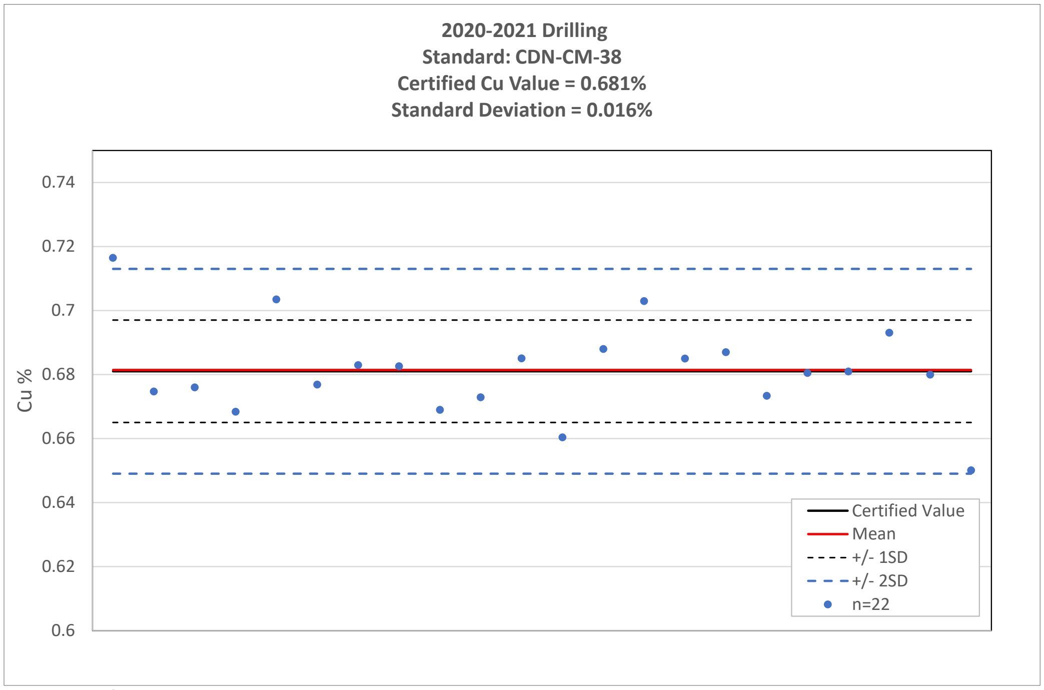
Source: Ridge Geosciences, 2022

Figure 11-2: Copper Control Chart for CDN-CM-31



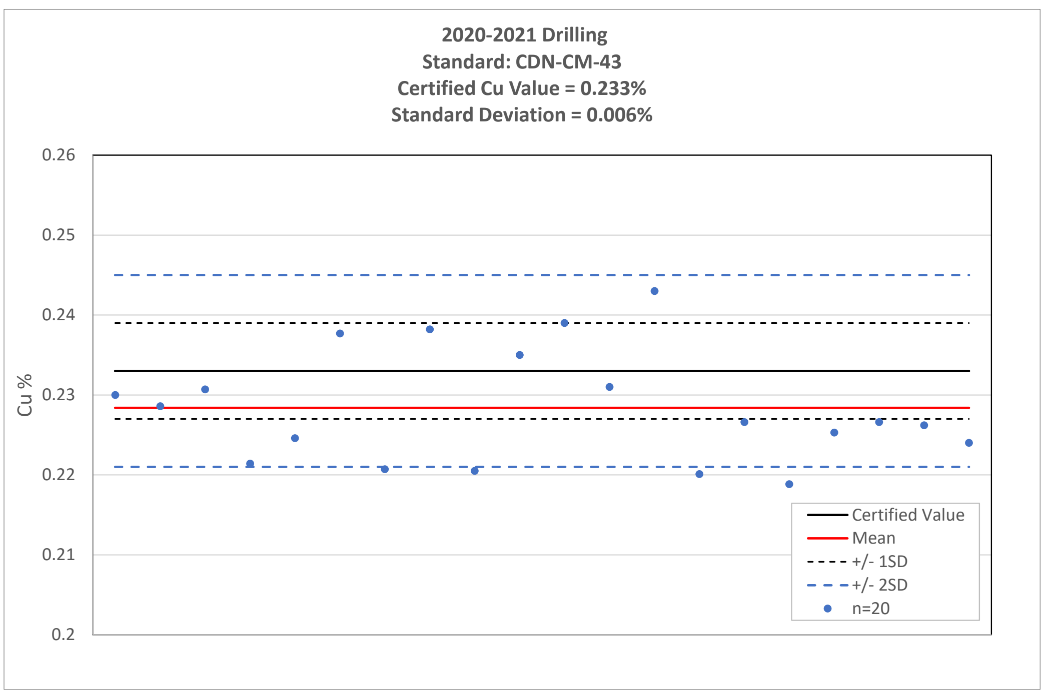
Source: Ridge Geosciences, 2022

Figure 11-3: Copper Control Chart for CDN-CM-38



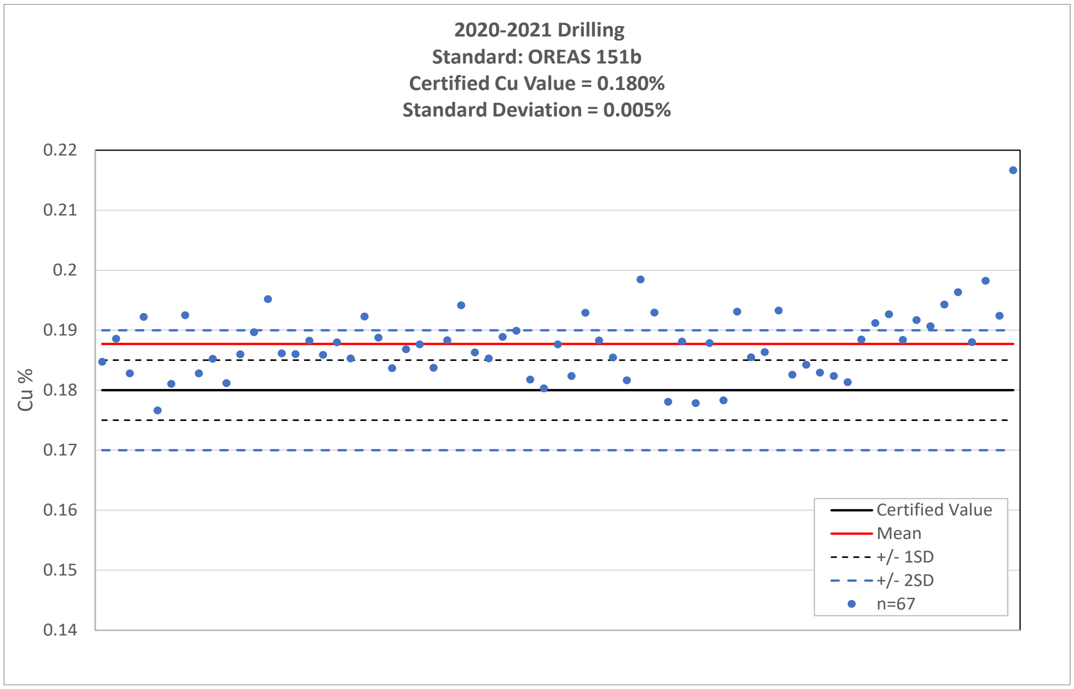
Source: Ridge Geosciences, 2022

Figure 11-4: Copper Control Chart for CDN-CM-43



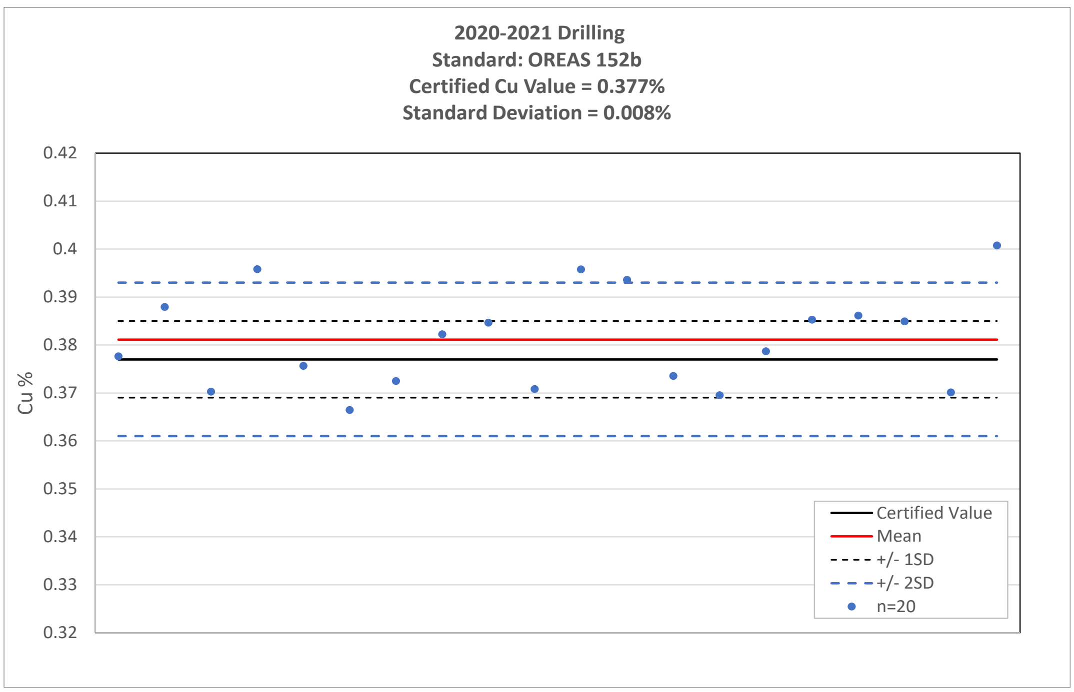
Source: Ridge Geosciences, 2022

Figure 11-5: Copper Control Chart for OREAS 151b



Source: Ridge Geosciences, 2022

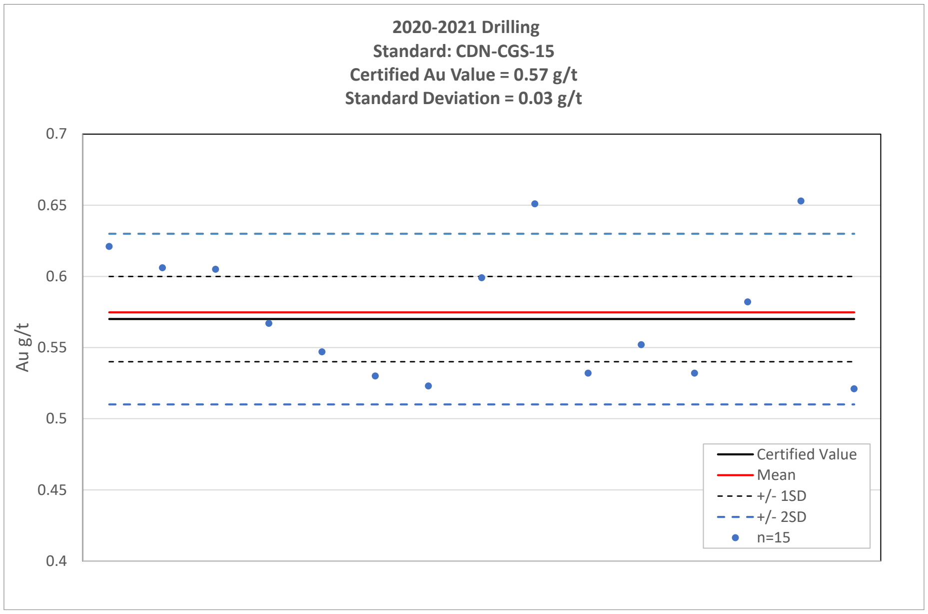
Figure 11-6: Copper Control Chart for OREAS 152b



Source: Ridge Geosciences, 2022

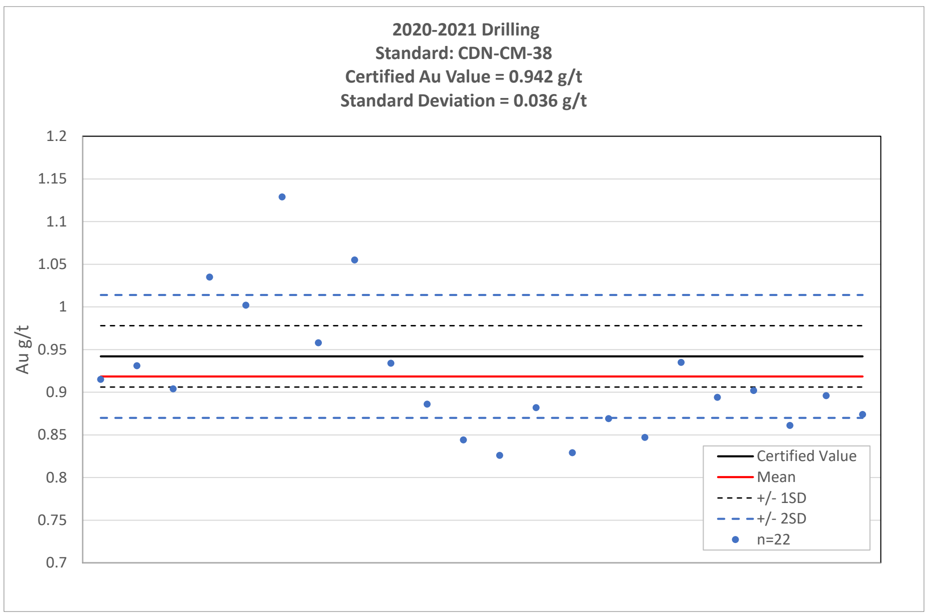
Representative control charts showing Au assay values are presented in Figure 11-7 through Figure 11-11. While some standards do have a few analyses outside of the +/-2SD range, mean values are close to the expected value. There are no significant or systematic bias noted by the CRM gold analyses, and the results are considered acceptable.

Figure 11-7: Gold Control Chart for CDN-CGS-15



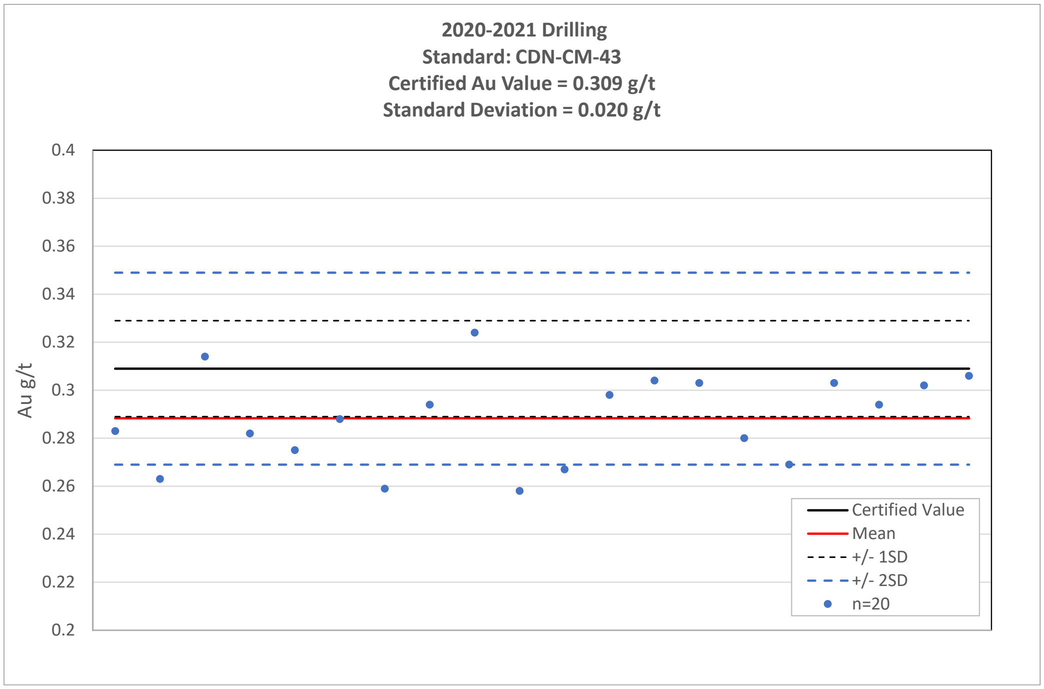
Source: Ridge Geosciences, 2022

Figure 11-8: Gold Control Chart for CDN-CM-38



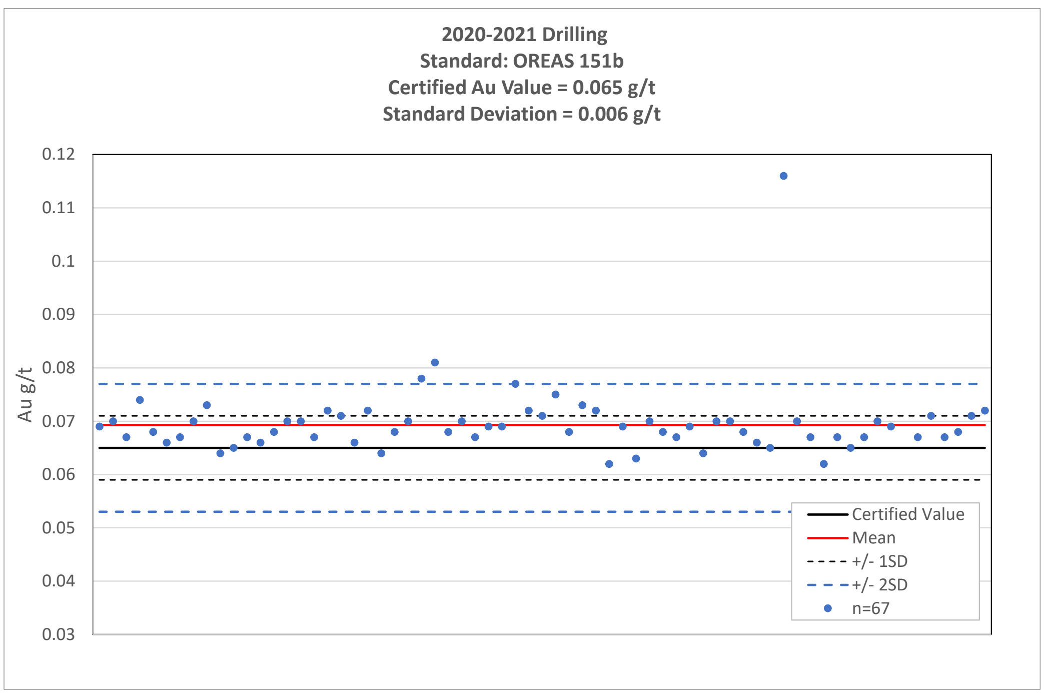
Source: Ridge Geosciences, 2022

Figure 11-9: Gold Control Chart for CDN-CM-43



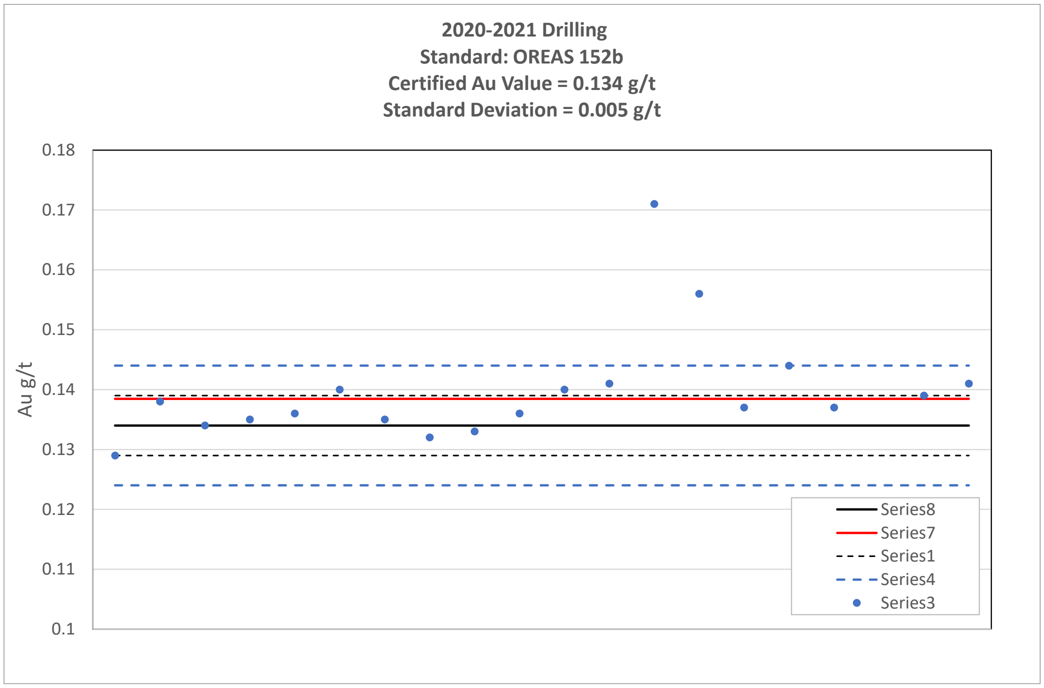
Source: Ridge Geosciences, 2022

Figure 11-10: Gold Control Chart for OREAS 151b



Source: Ridge Geosciences, 2022

Figure 11-11: Gold Control Chart for OREAS 152b

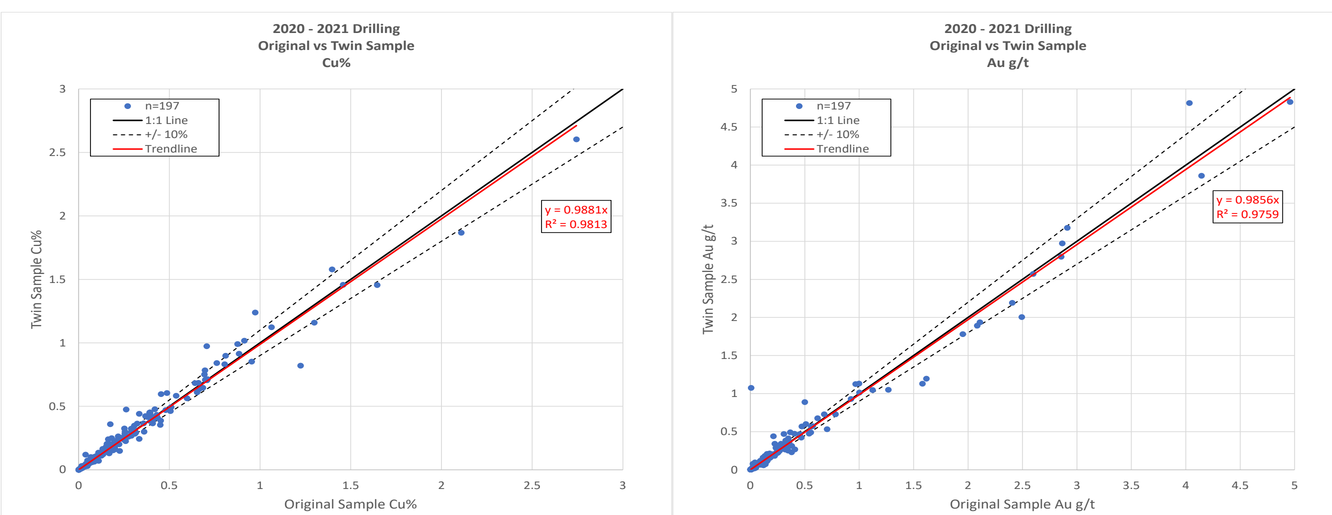


Source: Ridge Geosciences, 2022

11.1.5.3 Twin Samples

One hundred and ninety-seven pairs of quarter core were inserted into the sample stream as twin samples. The scatter plot of paired copper and gold values is shown in Figure 11-12. The values show good correlation about the 1:1 line. The comparison is acceptable for quartered core twin samples.

Figure 11-12: Comparison of Original and Quartered Core Twin Samples



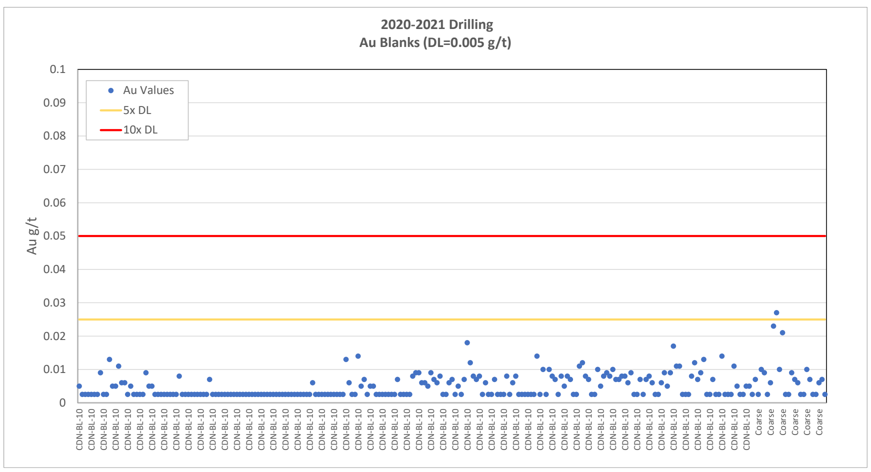
Source: Ridge Geosciences, 2022

11.1.5.3.1 Blanks

Blanks consisted of 223 certified pulp blanks (CDN-BL-10) and 24 coarse blanks (garden stone). The results of the Au assays of these blank samples are shown in Figure 11-13. Only one value exceeds 5 times the detection limit and occurs in the non-certified coarse garden stone. The results indicate that there was no contamination during the gold assay process.

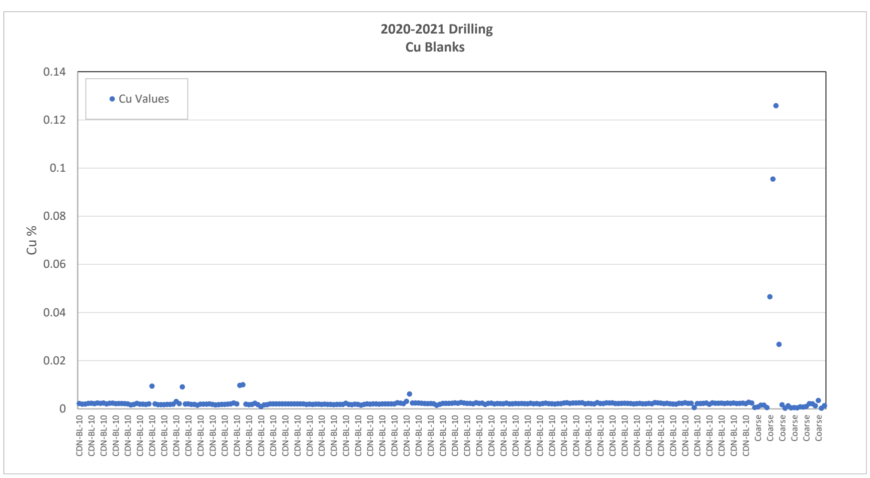
The certified blank CDN-BL-10 is not certified for copper. Most analyses are around 0.002%. Four analyses were around 0.01%, still well below any indication of possible contamination. The coarse garden stone returned four elevated values up to 0.13% copper and may not be a suitable candidate to properly assess contamination during sample preparation. Overall, results of the blanks indicate that no contamination occurred during the copper assay procedure. Results are shown in Figure 11-14.

Figure 11-13: Results of Gold Blank Analyses for 2020-2021 Drilling



Source: Ridge Geosciences, 2022

Figure 11-14: Results of Copper Blank Analyses for 2020-2021 Drilling



Source: Ridge Geosciences, 2022

11.1.5.4 2018 Drilling at Central Zone

A total of 804 QC samples comprising blanks, CRMs, and twin samples were inserted into the stream of drill core samples submitted or assay, for an insertion rate of around 25%. The QC sample summary is shown in Table 11-3.

Table 11-3: Central Zone QC Summary

Sampling Program	Count	(%)
Sample Count	3,234	
Pulp Blanks	270	8.3%
Certified Reference Materials	266	8.2%
Twin Samples	268	8.3%
Total QC Samples	804	24.9%

11.1.5.4.1 Certified Reference Materials

A total of 266 certified reference material samples were included in the QC samples from 2008 drilling at Central Zone. Seven different reference materials were obtained from CDN Resource Laboratories and inserted into the sample stream. The material names and expected values for Cu and Au are shown in Table 11-4. Early on, problems were identified with the assay results for CDN-CGS-23 and use of this material was discontinued.

Table 11-4: 2018 CRM Expected Values

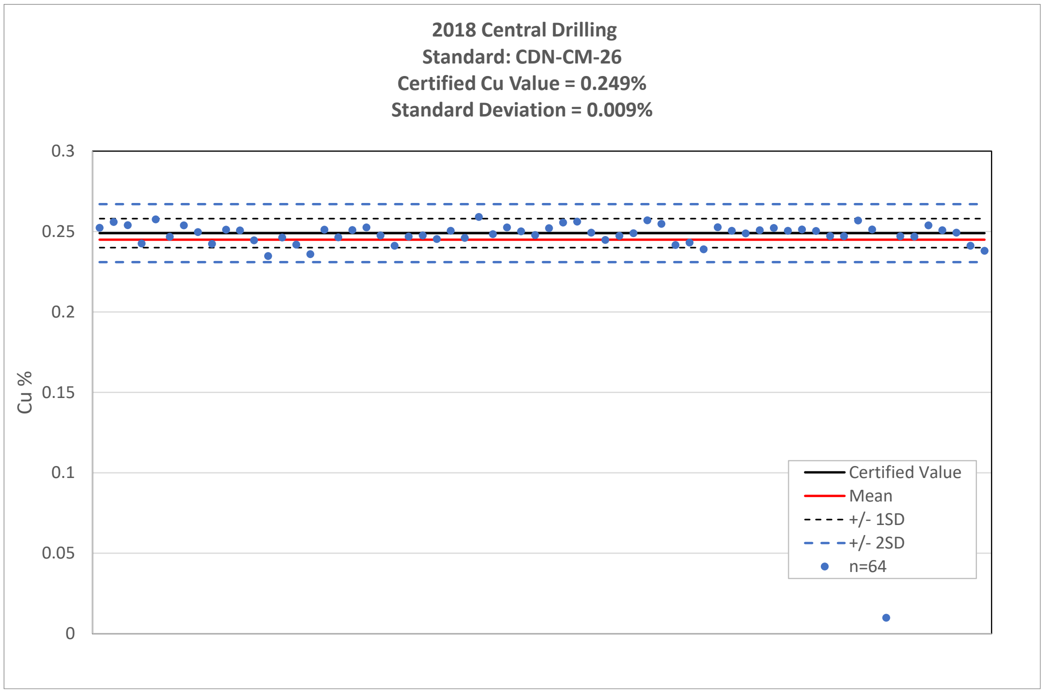
Reference Material	Cu (%)	Au (g/t)	# Used
CDN-CGS-23	0.182	0.218*	3
CDN-CM-26	0.249	0.372	64
CDN-CM-29	0.734	0.720	46
CDN-CM-38	0.681	0.942	37
CDN-CM-40	0.561	1.31	38
CDN-CM-42	0.529	0.576	44
CDN-CM-43	0.233	0.309	34

An * indicates a provisional value

Representative control charts showing Cu assay values for the standards are presented in Figure 11-15 through Figure 11-18. These figures generally show good results with most values within the ±2 standard deviation performance gates and mean values close to the expected values. CDN-CM-26 has one value well below the expected range and has likely been mislabelled as the incorrect QC type. Standard CDN-CM-40 shows multiple values far higher than the expected range, causing the mean to plot outside of this range. The anomalous results for this standard are thought to be a problem with the standard itself and not with the laboratory. Overall performance of CRM copper values is acceptable.

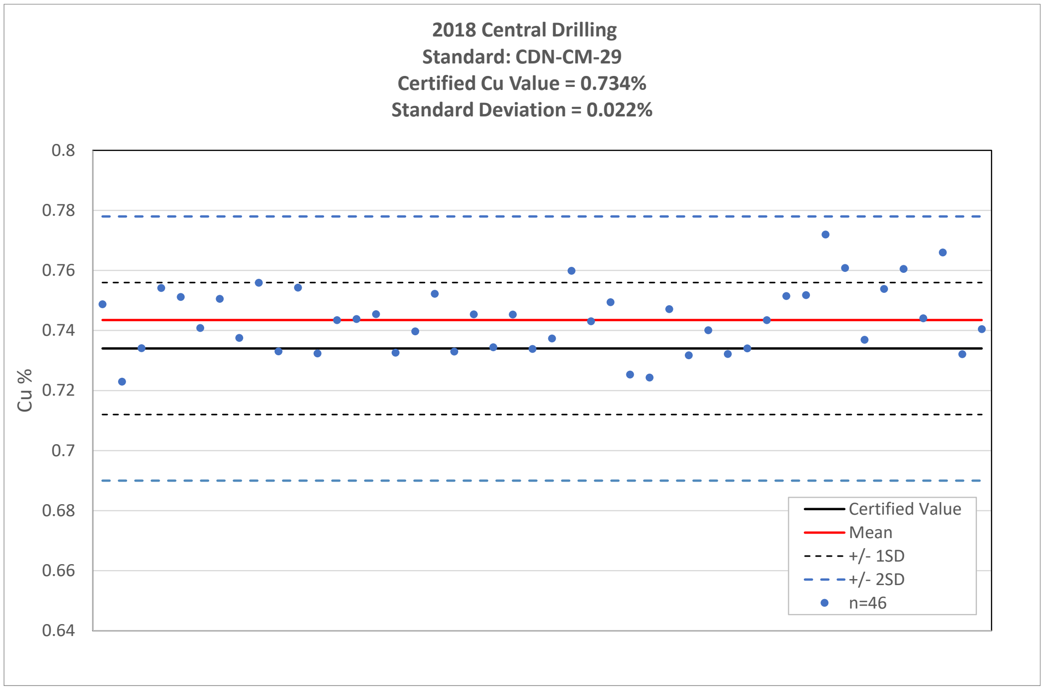
Representative control charts showing Au assay values for the standards are presented in Figure 11-19 through Figure 11-22. Assays of all six reference materials give mean values close to the expected value with a few values outside of the performance gates. The low value in CDN-CM-26 corresponds with the low copper value, which again is likely to be a mislabelled QC sample. There are no systematic biases noted in the results, and they are considered to be acceptable.

Figure 11-15: Copper Control Chart for CDN-CM-26



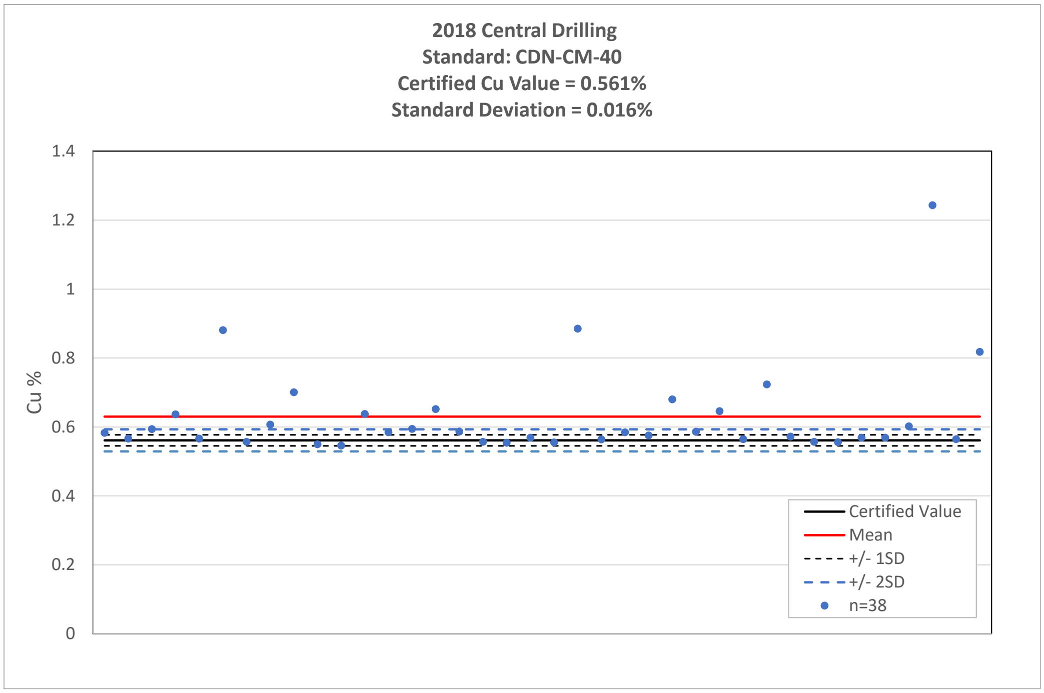
Source: Ridge Geosciences, 2022

Figure 11-16: Copper Control Chart for CDN-CM-29



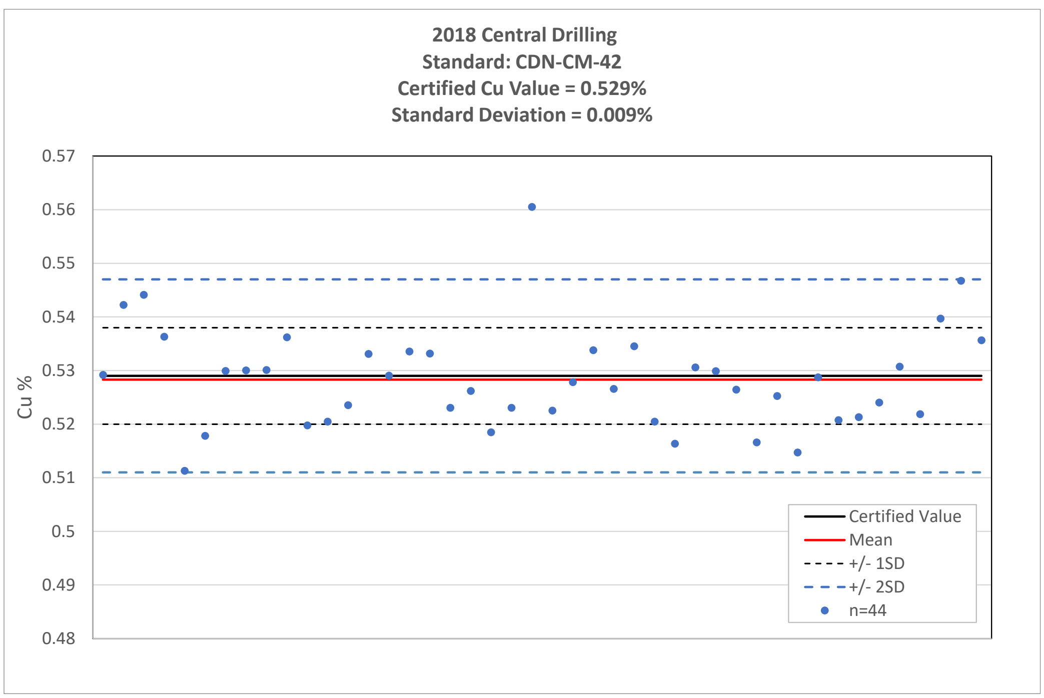
Source: Ridge Geosciences, 2022

Figure 11-17: Copper Control Chart for CDN-CN-40



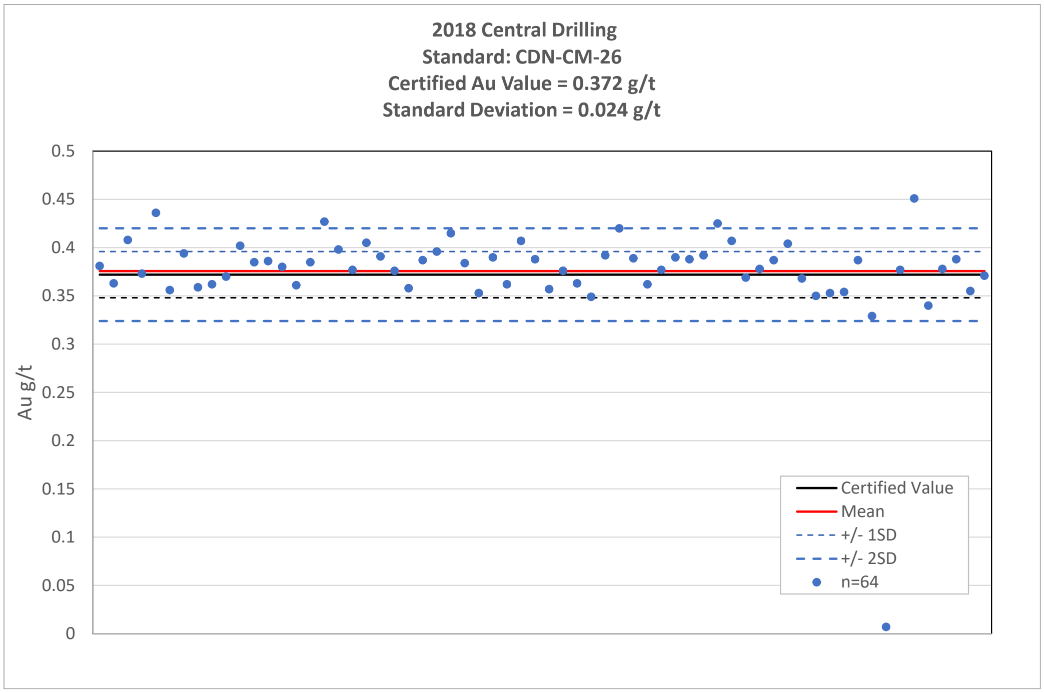
Source: Ridge Geosciences, 2022

Figure 11-18: Copper Control Chart for CDN-CM-42



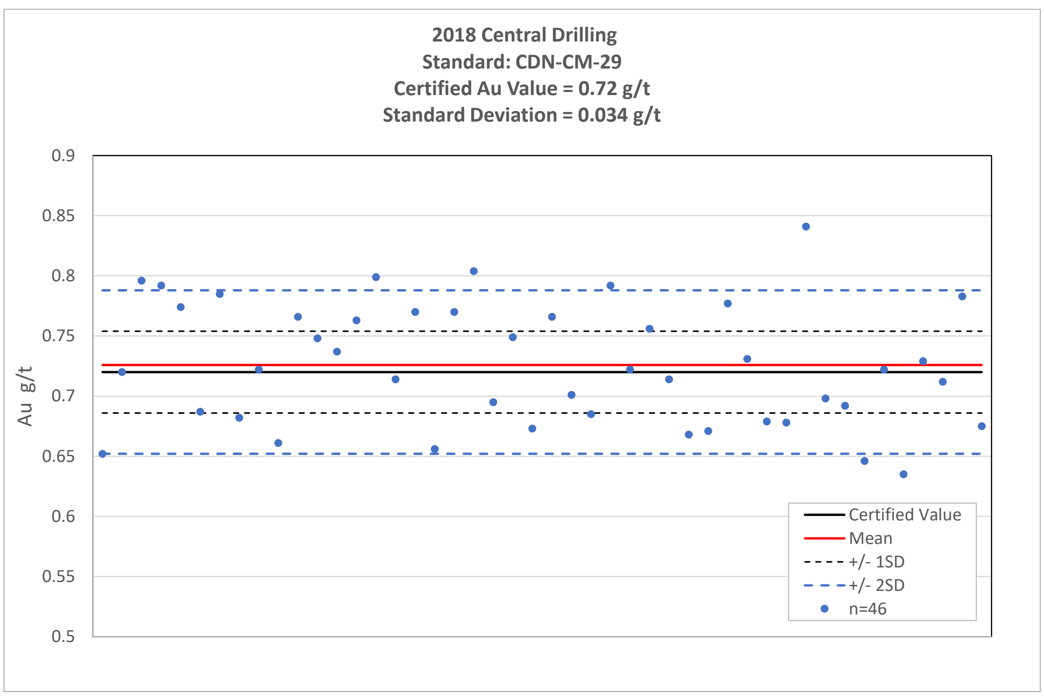
Source: Ridge Geosciences, 2022

Figure 11-19: Gold Control Chart for CDN-C-26



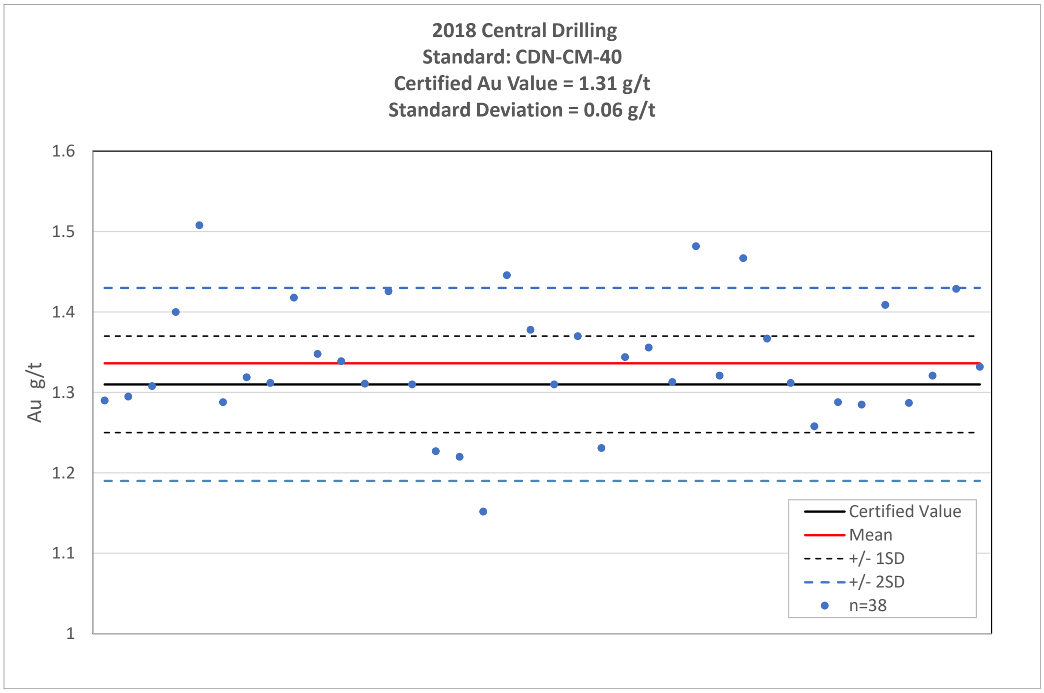
Source: Ridge Geosciences, 2022

Figure 11-20: Gold Control Chart for CDN-CM-29



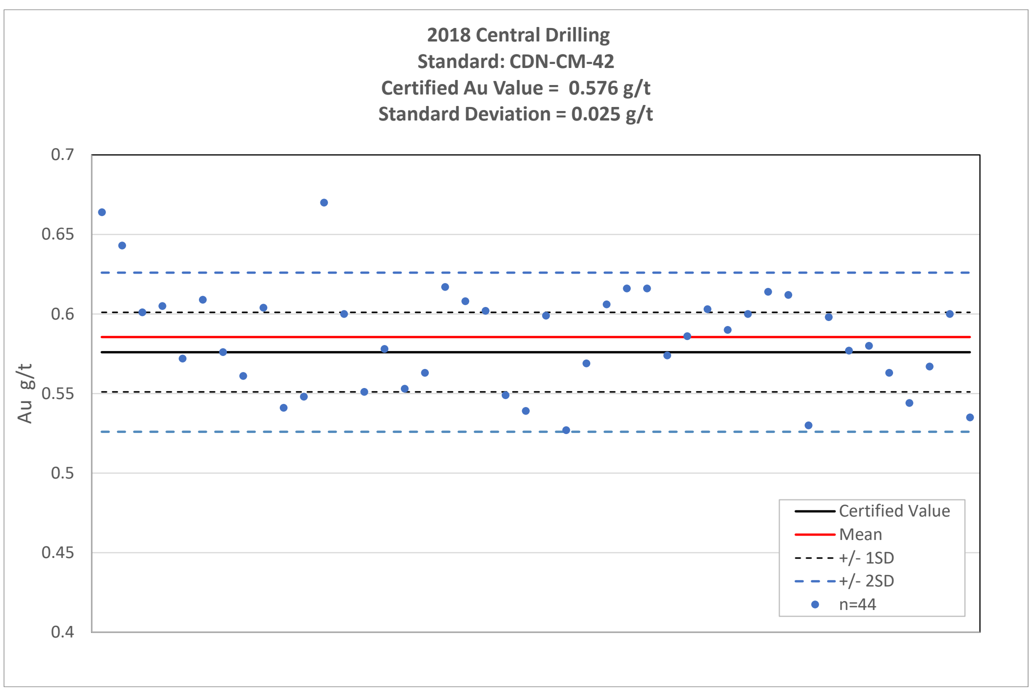
Source: Ridge Geosciences, 2022

Figure 11-21: Gold Control Chart for CDN-CM-40



Source: Ridge Geosciences, 2022

Figure 11-22: Gold Control Chart for CDN-CM-42

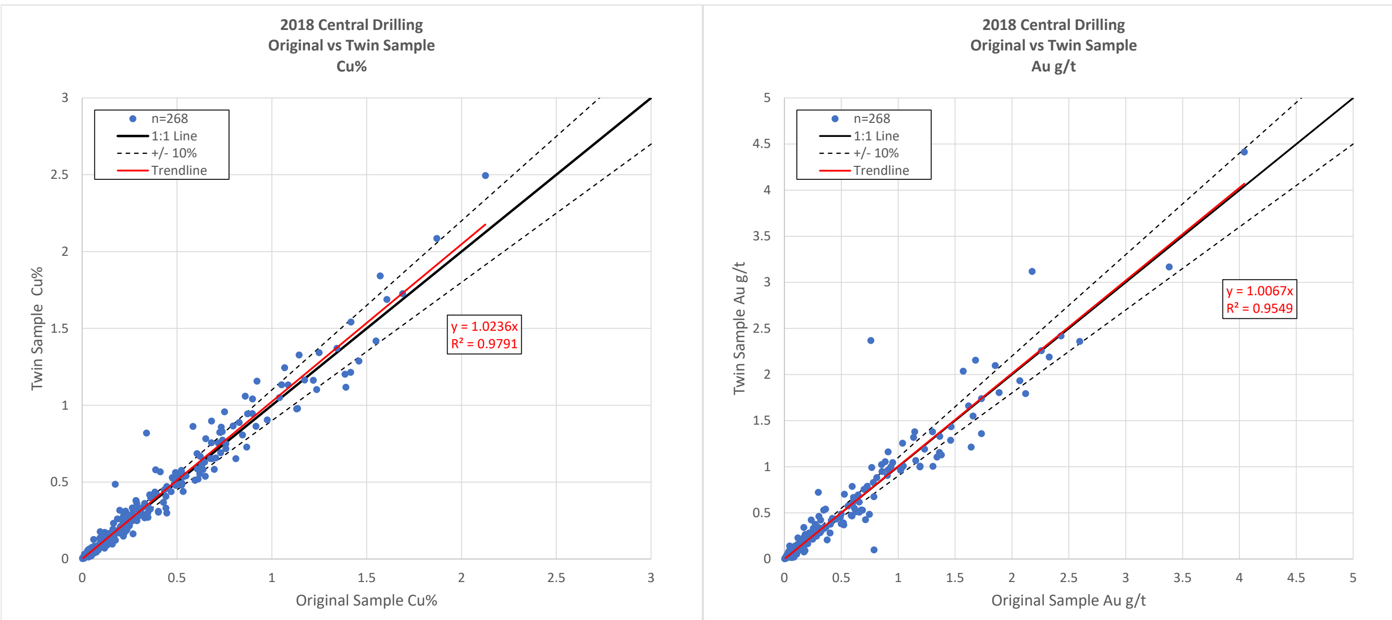


Source: Ridge Geosciences, 2022

11.1.5.4.2 Twin Samples

Two hundred and sixty-eight pairs of quarter core were inserted into the sample stream as twin samples. The scatter plot of paired copper and gold values is shown in Figure 11-23. The values show good correlation about the 1:1 line. The comparison is acceptable for quartered core twin samples.

Figure 11-23: Comparison of Original and Quartered Core Twin Samples



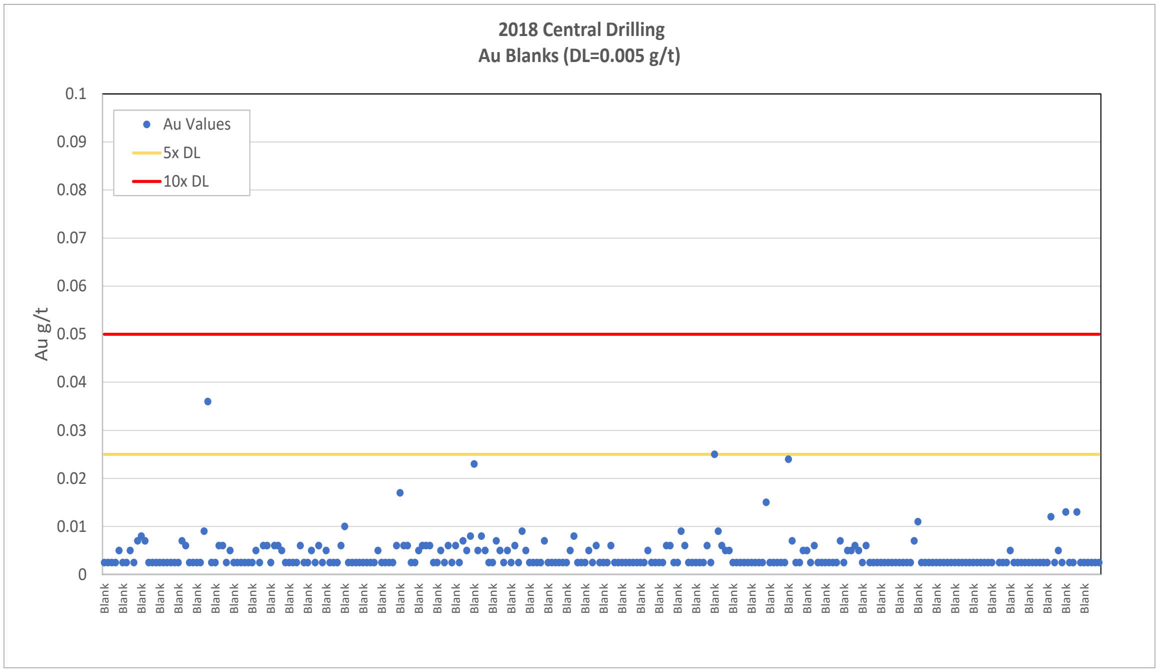
Source: Ridge Geosciences, 2022

11.1.5.4.3 Blanks

Two hundred and seventy pulp blanks were inserted into the sample stream. The results of the Au assays of these blank samples are shown in Figure 11-24. Only one value exceeds 5 times the detection limit. The results indicate that there was no contamination during the gold assay process.

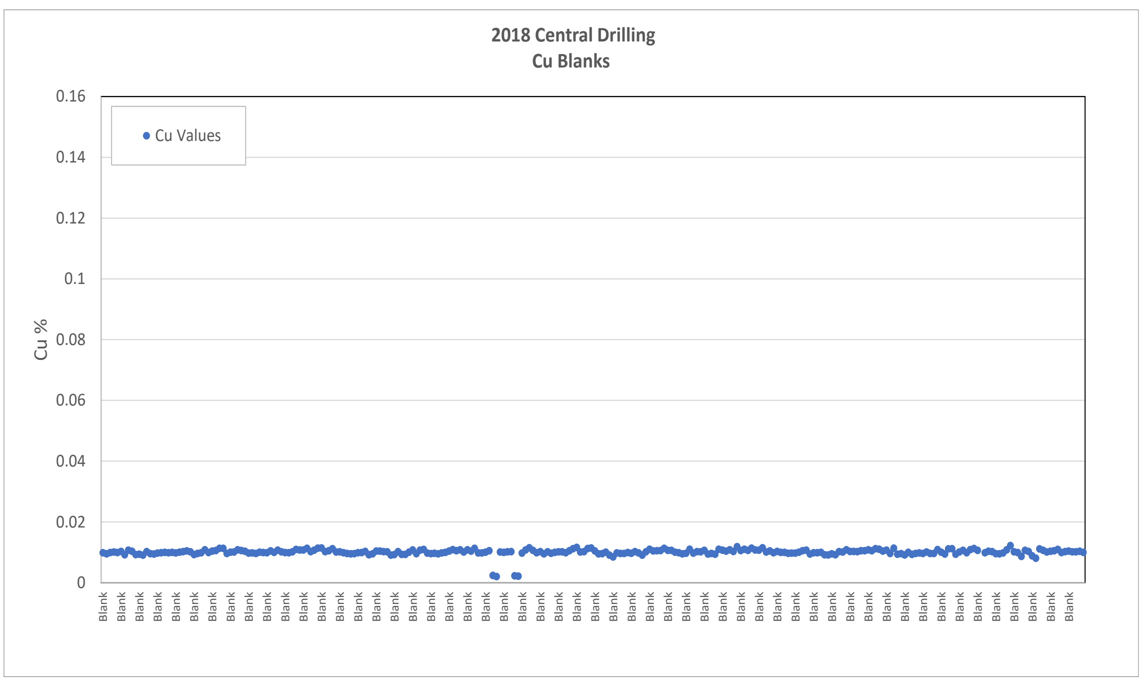
All copper analyses are around 0.01% or below, well below any indication of possible contamination. Results of the blanks are shown in Figure 11-25 and indicate that no contamination occurred during the copper assay procedure.

Figure 11-24: Results of Gold Blank Analyses for 2018 Central Zone Drilling



Source: Ridge Geosciences, 2022

Figure 11-25: Results of Copper Blank Analyses for 2018 Central Zone Drilling



Source: Ridge Geosciences, 2022

11.1.5.5 2006 – 2016 Drilling at Central Zone

A total of 1,517 QC samples comprising blanks, CRMs, and twin samples were inserted into the stream of drill core samples submitted or assay, for an insertion rate of around 7%. Details are shown in in Table 11-5.

Table 11-5: 2006-2016 Central Zone QC Summary

Sampling Program	Count	(%)
Sample Count	21,484	
Pulp Blanks	391	1.8%
Certified Reference Materials	744	3.5%
Twin Samples	382	1.8%
Total QC Samples	1,517	7.1%

Standard CDN-CSG-18 has been removed from the QA/QC analysis. The results from the lab are reported as the same number for a large portion of the samples. This is assumed to be a data entry error as described in previous technical reports. Therefore, this standard has been removed from the QA/QC analysis for this review.

11.1.5.5.1 Certified Reference Materials

A total of 744 certified reference material samples were included in the QC samples from 2006 to 2016 drilling at Central Zone. Ten different reference materials were obtained from CDN Resource Laboratories and inserted into the sample stream. The material names and expected values for Cu and Au are shown in Table 11-6. Early on, problems were identified with the assay value results for material CDN-CGS-23 and use of this material was later discontinued. Standard CDN-CSG-18 has been removed from the QA/QC analysis, as results from the lab are reported as the same number for a large portion of the samples. This is discussed in previous technical reports as being a data entry error.

2006-2016 Central Zone Drilling CRM Expected Values

Table 11-6: 2006-2016 Central Zone Drilling CRM Expected Values

Reference Material	Cu(%)	Au (g/t)	# Used
CDN-CGS-11	0.683	0.73	333
CDN-CGS-12	0.265	0.29	247
CDN-CGS-15	0.451	0.57	7
CDN-CGS-18	0.319	0.297*	84
CDN-CGS-22	0.711	0.64	10
CDN-CGS-23	0.182	0.218*	8
CDN-CM-23	0.471	0.549	21
CDN-CM-36	0.227	0.316	17
CDN-CM-5	0.319	0.294	13
CDN-CM-7	0.445	0.427	4

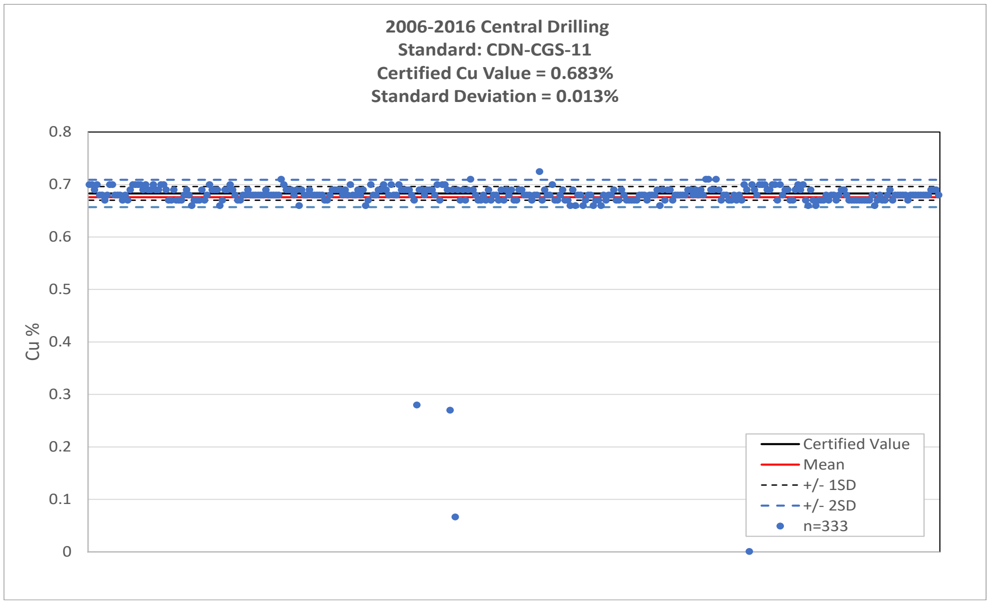
An * indicates a provisional value

Representative control charts showing Cu assay values for the standards are presented in Figure 11-26 through Figure 11-28. These figures generally show good results with most values within the ±2 standard deviation performance gates and mean values close to the expected values. Some samples fall well outside the expected ranges and are likely to be mislabelled QC sample types. CDN-CSG-15 performs poorly but is only comprised of 7 analyses. The overall performance of CRM copper values is acceptable with no significant or systematic biases noted.

Representative control charts showing Au assay values for the standards are presented in Figure 11-29.

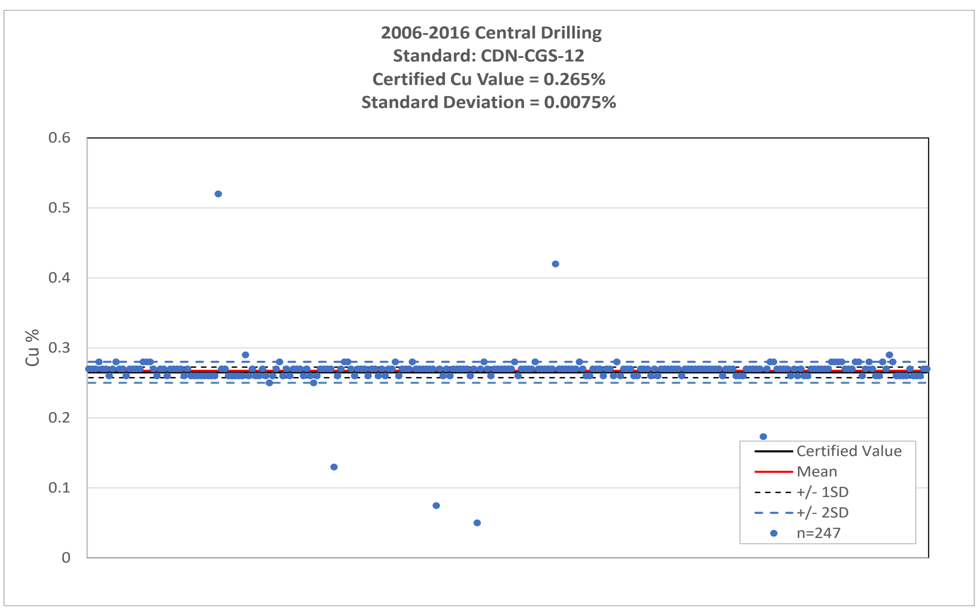
Figure 11-31. These figures generally show good results with most values within the ±2 standard deviation performance gates and mean values close to the expected values. Some samples fall well outside the expected ranges and are likely to be mislabelled QC sample types. CDN-CSG-15 performs poorly but is only comprised of 7 samples. The overall performance of CRM gold values is acceptable with no significant or systematic biases noted.

Figure 11-26: Copper Control Chart for CDN-CGS-11



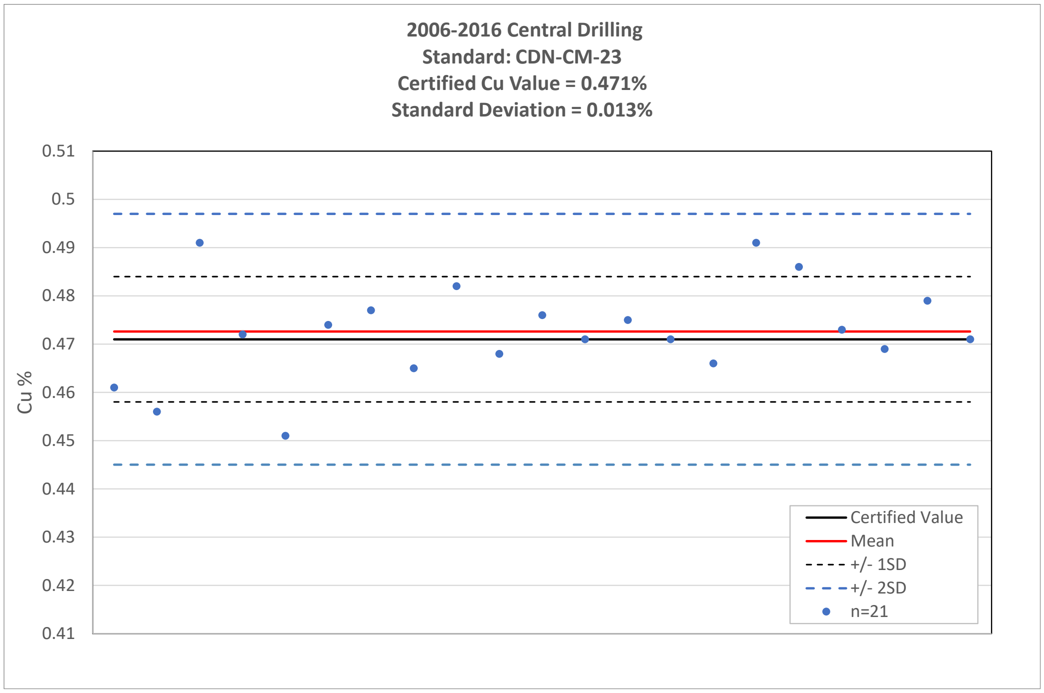
Source: Ridge Geosciences, 2022

Figure 11-27: Copper Control Chart for CDN-CGS-12



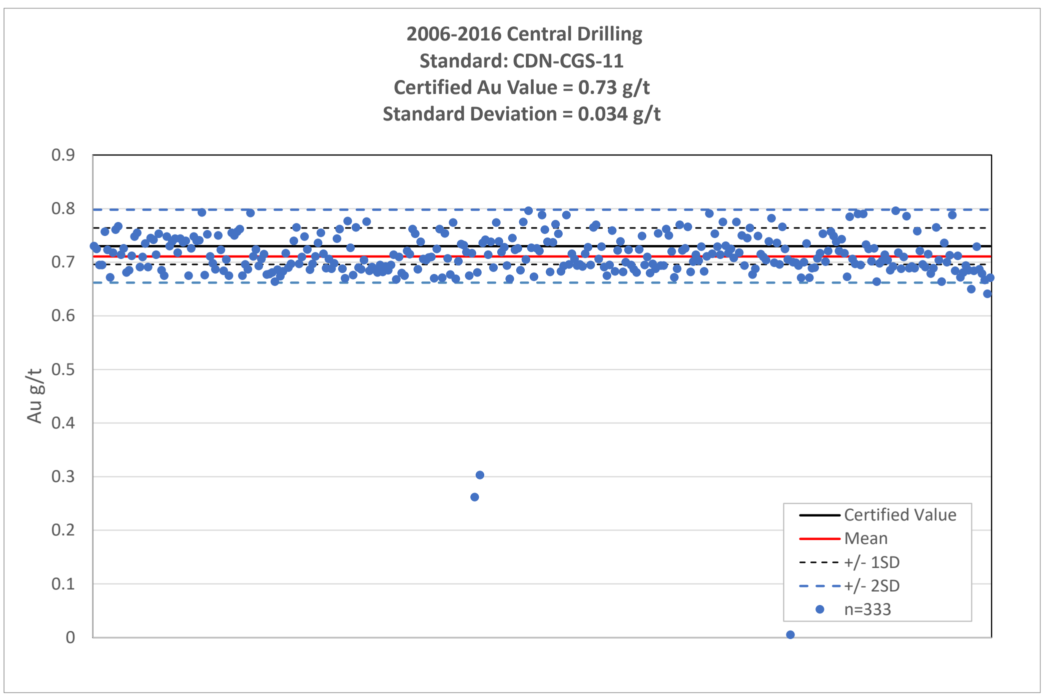
Source: Ridge Geosciences, 2022

Figure 11-28: Copper Control Chart for CDN-CM-23



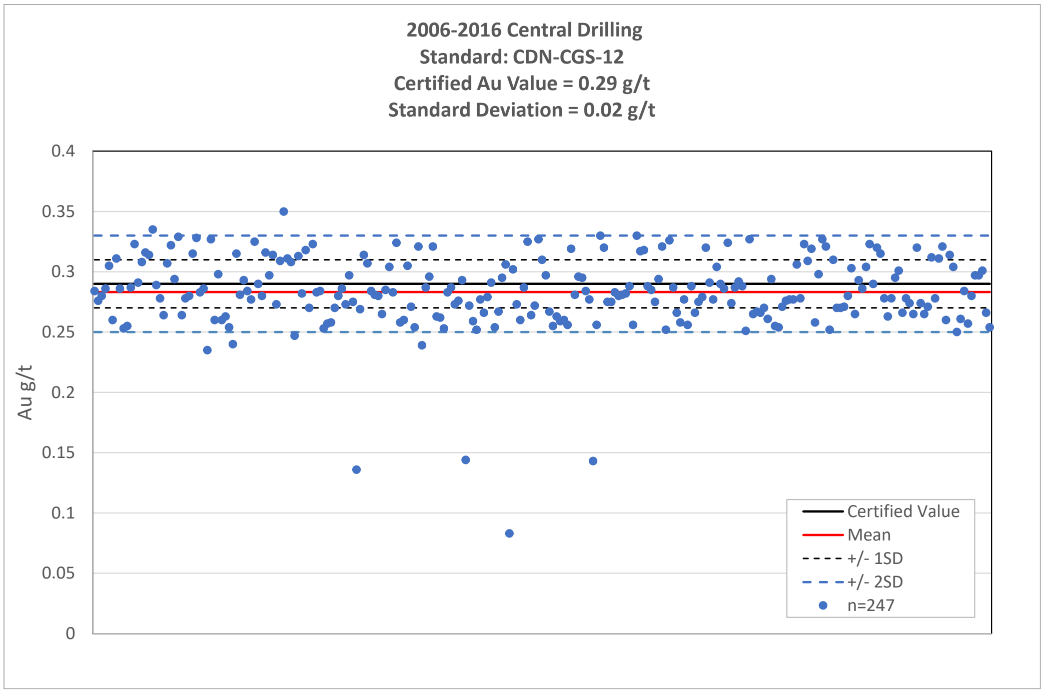
Source: Ridge Geosciences, 2022

Figure 11-29: Gold Control Chart for CDN-CGS-11



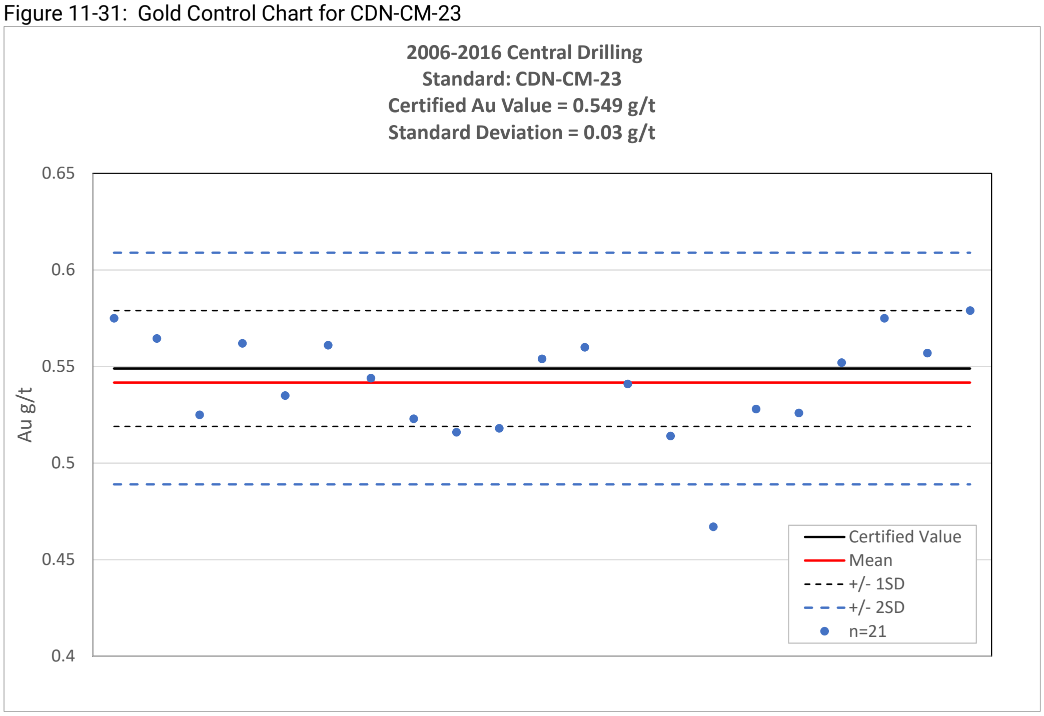
Source: Ridge Geosciences, 2022

Figure 11-30: Gold Control Chart for CDN-CGS-12



Source: Ridge Geosciences, 2022

Figure 11-31: Gold Control Chart for CDN-CM-23

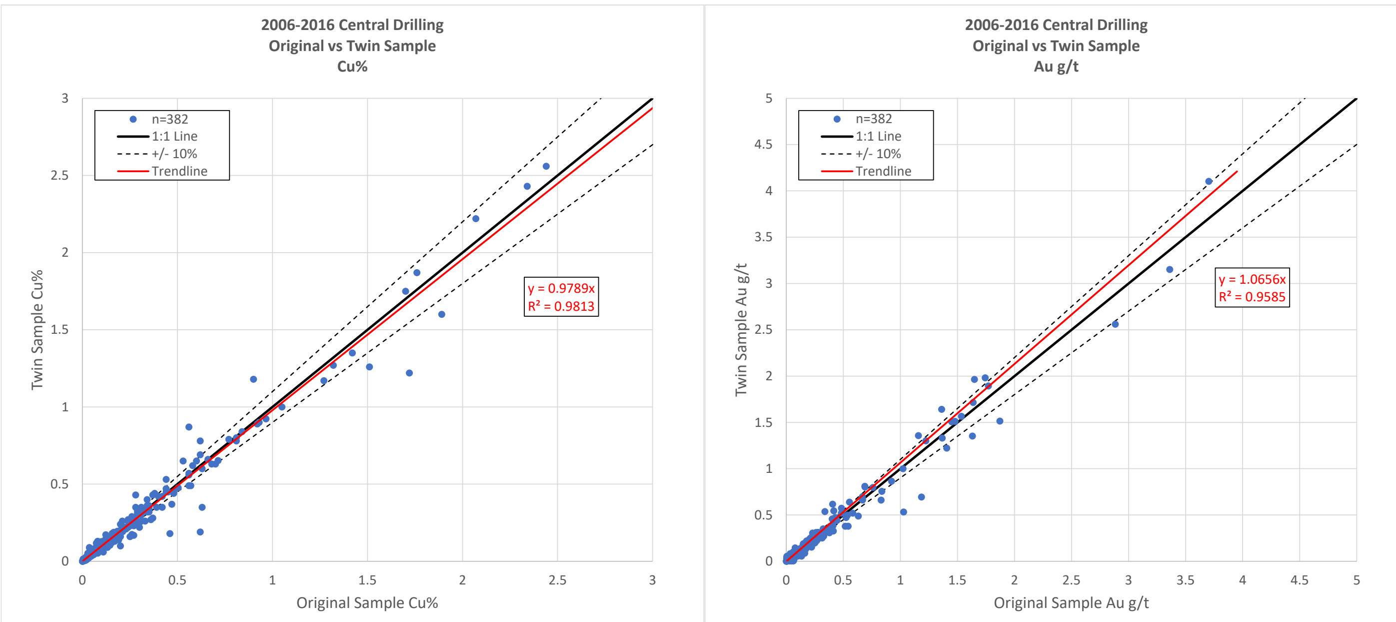


Source: Ridge Geosciences, 2022

11.1.5.5.2 Twin Samples

Three hundred and eighty-two pairs of quarter core were inserted into the sample stream as twin samples. The scatter plot of paired copper and gold values is shown in Figure 11-32. The values show good correlation about the 1:1 line. The comparison is acceptable for quartered core twin samples.

Figure 11-32: Comparison of Original vs Quartered Core Twin Samples



Source: Ridge Geosciences, 2022

11.1.5.5.3 Blanks

A total of 391 pulp blanks were included in samples from 2006 to 2016. The blank material is a pre-crushed prepackaged blank from CDN Resources Laboratories Inc. The results of the Au assays of these blank samples are shown in Figure 11-33. Seven values exceed 5 times the detection limit. The results indicate that there was no contamination during the gold assay process.

All but one copper analyses are around 0.01% or below, well below any indication of possible contamination. One sample returned 0.02% Cu. Results of the blanks indicate that no contamination occurred during the copper assay procedure and are shown in Figure 11-34

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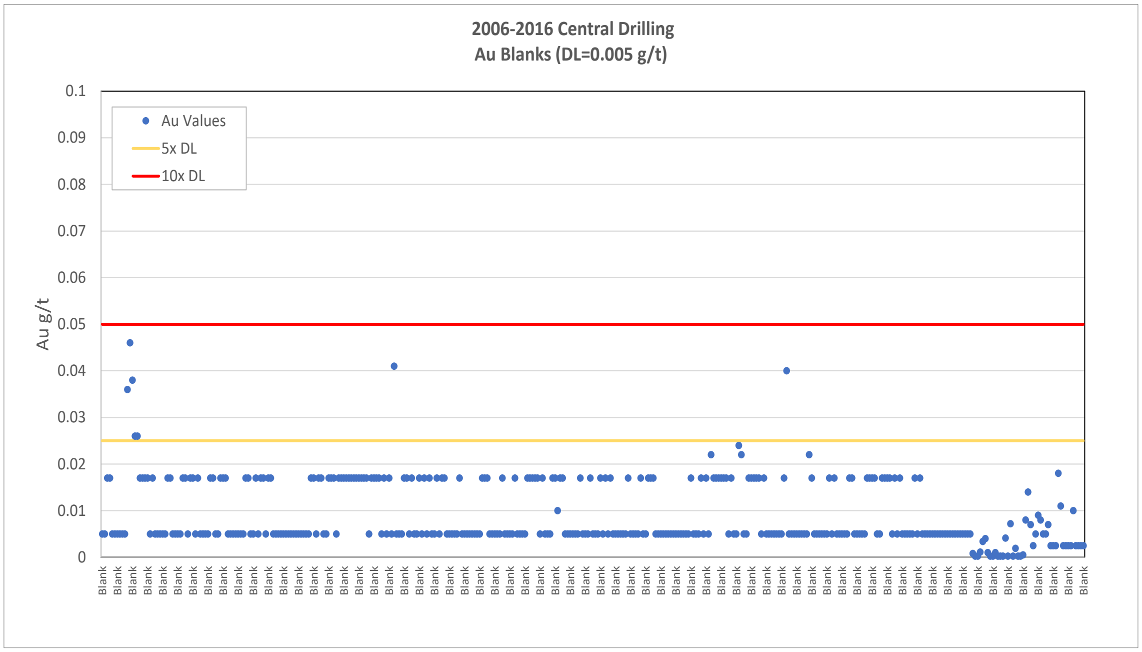
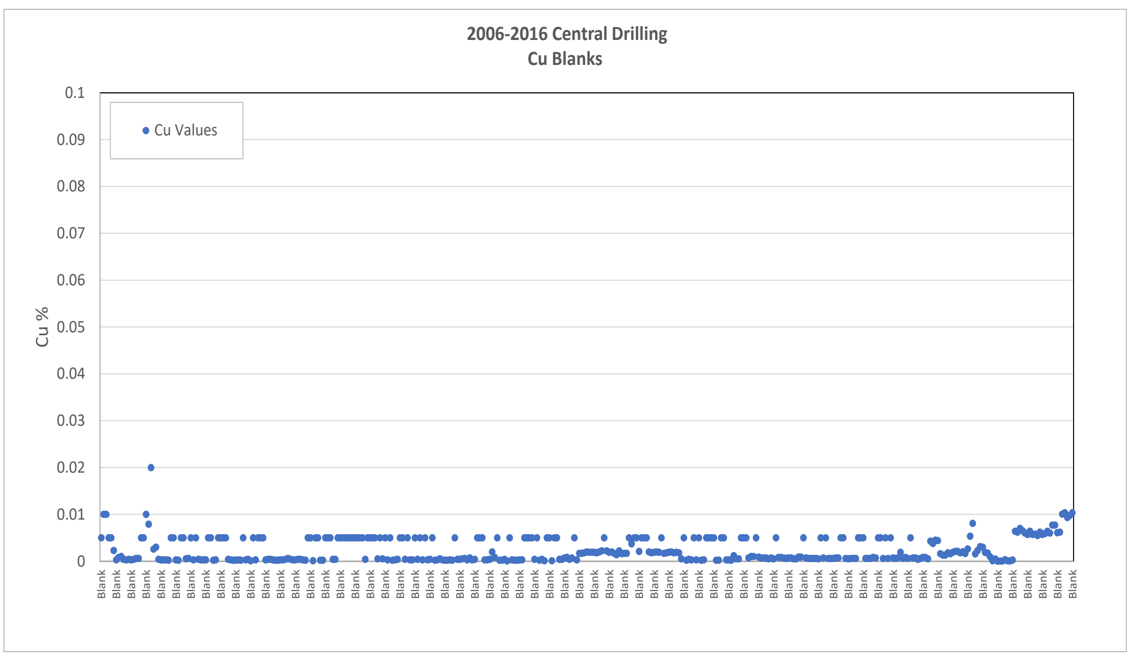
Figure 11-33: Results of Gold Blank Analyses for 2006-2016 Central Zone Drilling



Source: Ridge Geosciences, 2022

Figure 11-34: Results of Copper Blank Analyses for 2006-2016 Central Zone Drilling



Source: Ridge Geosciences, 2022

11.1.5.6 2008 – 2010 Drilling at South Zone

A total of 560 QA/QC samples comprising blanks, CRMs, and twin samples were inserted into the stream of drill core samples submitted or assay, for an insertion rate of around 7%. Details are shown in Table 11-7.

Table 11-7: 2008-2010 South Zone QC Sample Summary

Sampling Program	Count	(%)
Sample Count	8,064	
Pulp Blanks	140	1.7%
Certified Reference Materials	281	3.5%
Twin Samples	139	1.7%
Total QC Samples	**560**	**6.9%**

11.1.5.6.1 Certified Reference Materials

A total of 281 certified reference material samples were included in the QC samples from 2008 to 2010 drilling at South Zone. Five different reference materials were obtained from CDN Resource Laboratories and inserted into the sample stream. The material names and expected values for Cu and Au are shown in Table 11-8. Early on, problems were identified with the assay value results for material CDN-CGS-23 and use of this material was discontinued.

Table 11-8: 2008-2010 South Zone Drilling CRM Expected Values

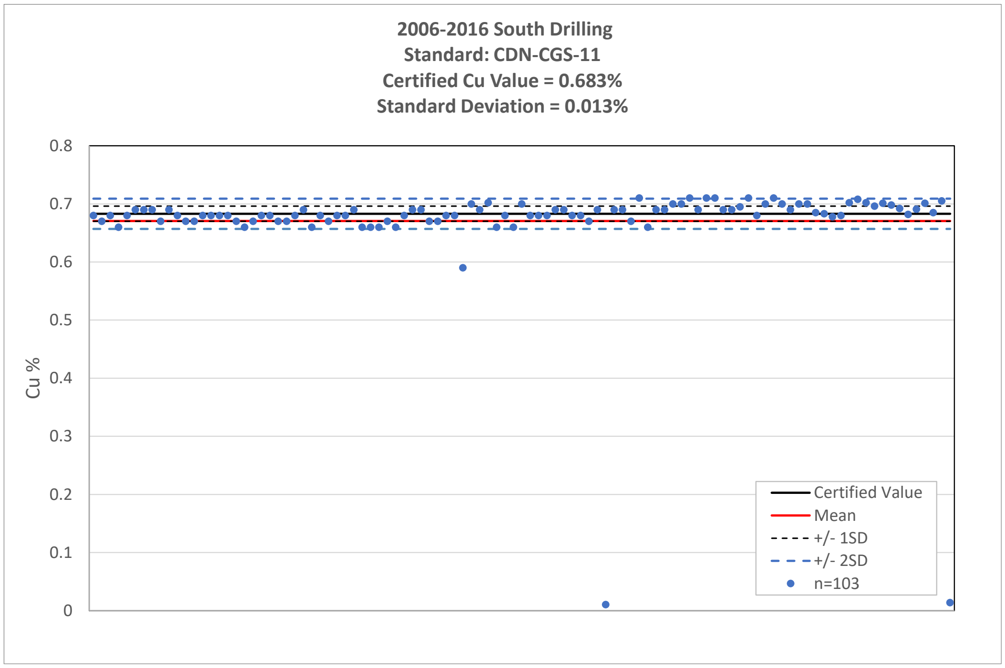
Reference Material	Cu (%)	Au (g/t)	# Used
CDN-CGS-11	0.683	0.73	103
CDN-CGS-12	0.265	0.29	20
CDN-CGS-18	0.319	0.297*	83
CDN-CGS-23	0.182	0.218*	36
CDN-CM-7	0.445	0.427	39

An * indicates a provisional value

Representative control charts showing Cu assay values for the standards are presented in Figure 11-35 through Figure 11-37. These figures generally show good results with most values within the ±2 standard deviation performance gates and mean values close to the expected values. Some samples fall well outside the expected ranges and are likely to be mislabelled QC sample types. CDN-CGS-23 did not perform well with around 40% of the samples falling outside of the ±2SD performance gates. The sample overall reported lower than expected values. The use of this material was subsequently discontinued. The overall performance of CRM copper values is acceptable with no systematic biases noted.

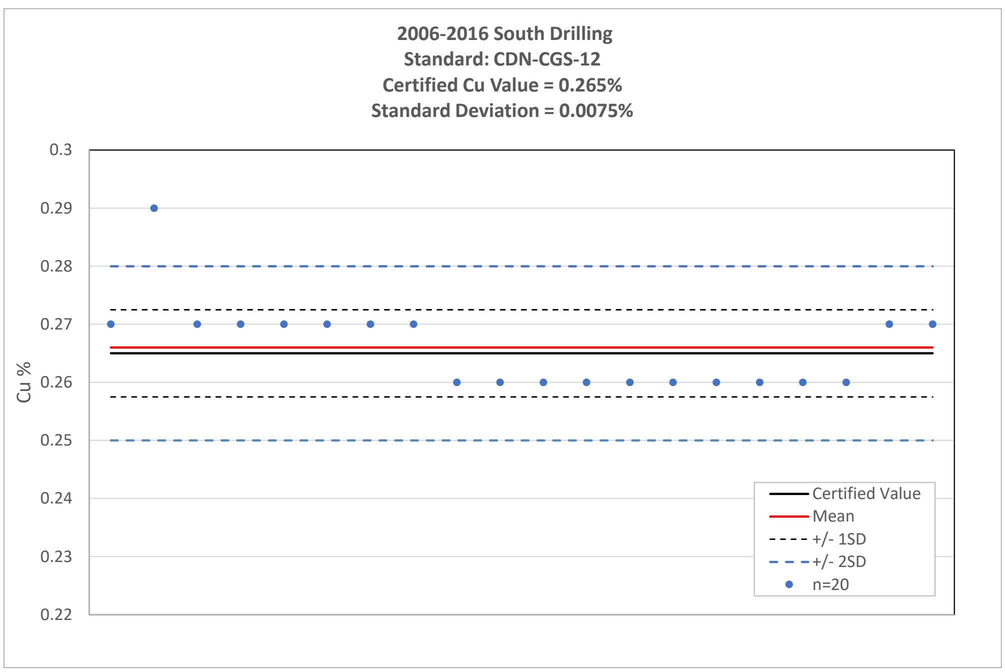
Representative control charts showing Au assay values for the standards are presented in Figure 11-38 through Figure 11-39. These figures generally show good results with most values within the ±2 standard deviation performance gates and mean values close to the expected values. Some samples fall well outside the expected ranges and are likely to be mislabelled QC sample types. The overall performance of CRM gold values is considered acceptable with no significant or systematic biases noted.

Figure 11-35: Copper Control Chart for CDN-CGS-11



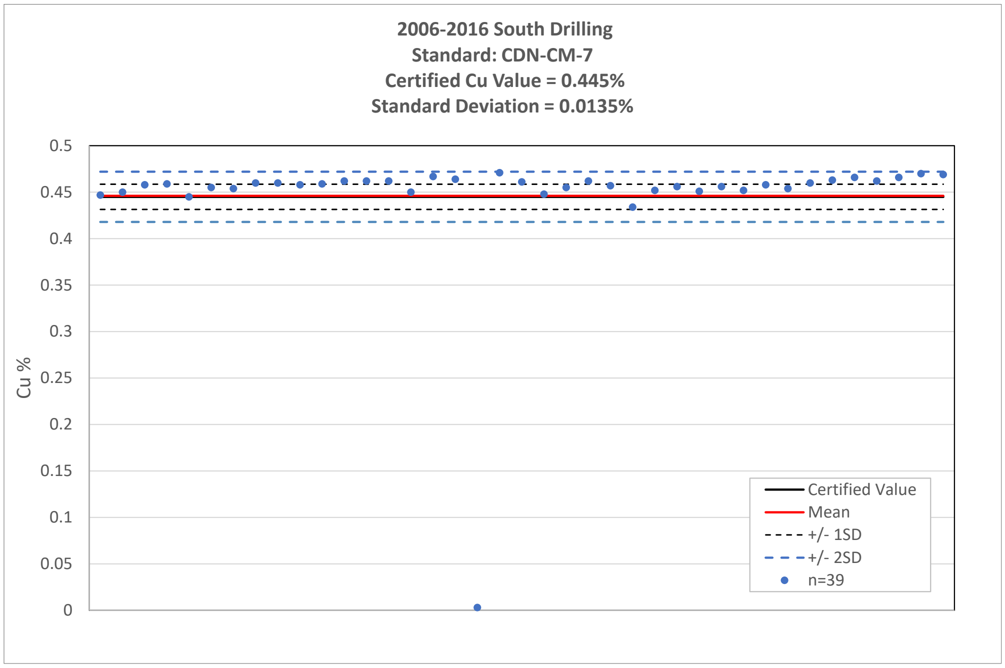
Source: Ridge Geosciences, 2022

Figure 11-36: Copper Control Chart for CDN-CGS-12



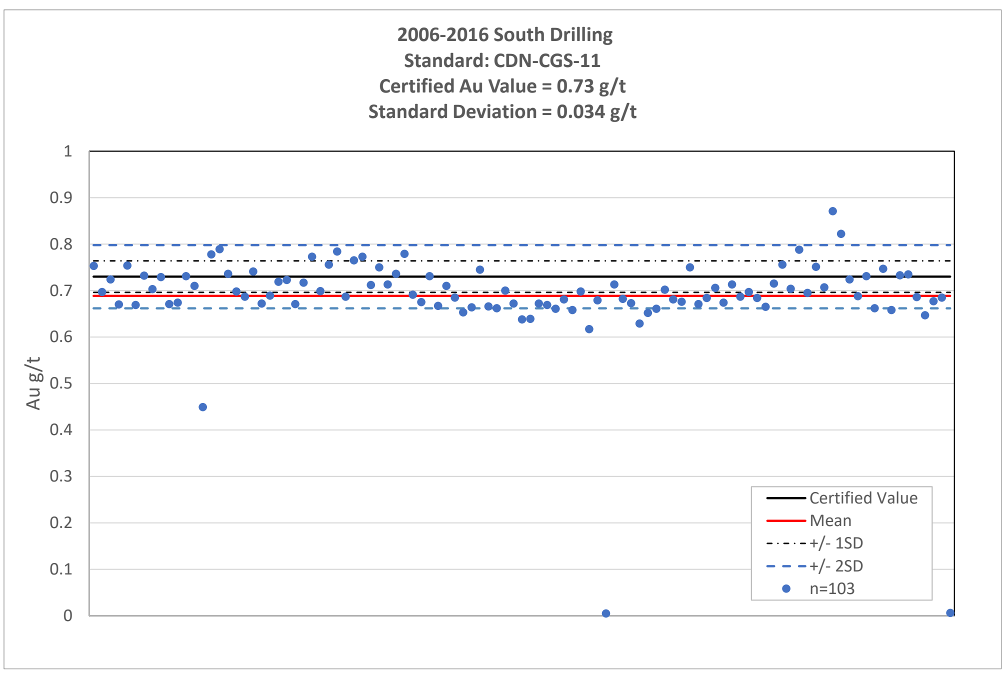
Source: Ridge Geosciences, 2022

Figure 11-37: Copper Control Chart for CDN-CM-7



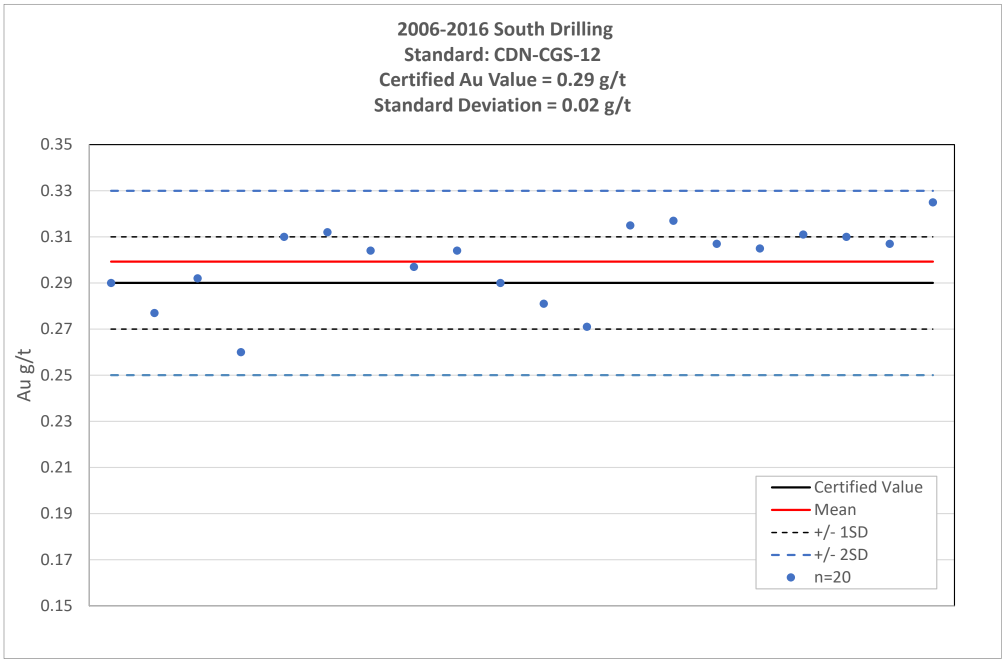
Source: Ridge Geosciences, 2022

Figure 11-38: Gold Control Chart for CDN-CGS-11



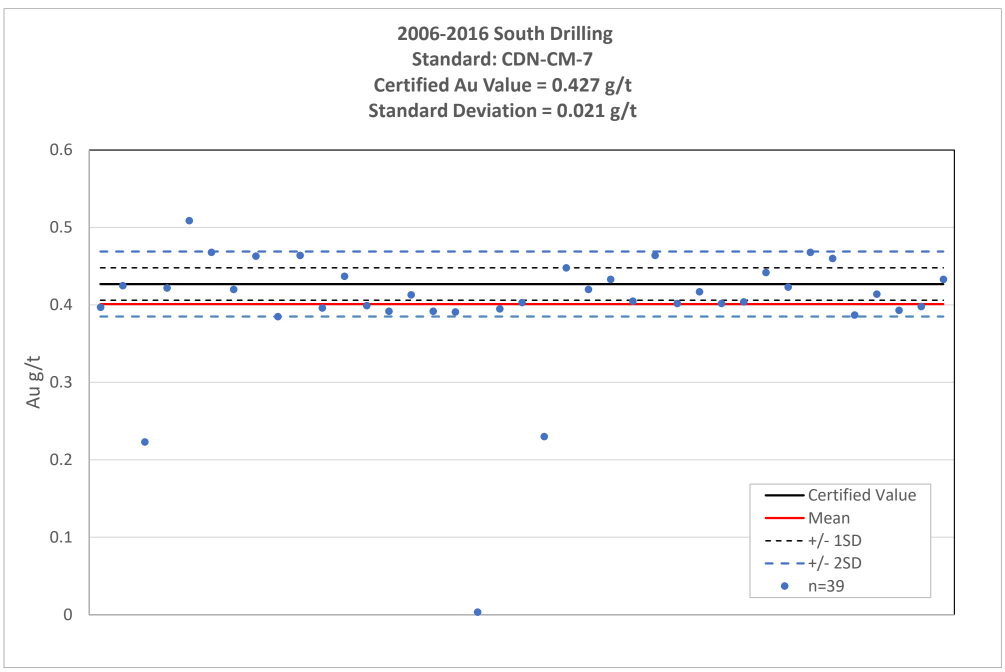
Source: Ridge Geosciences, 2022

Figure 11-39: Gold Control Chart for CDN-CGS-12



Source: Ridge Geosciences, 2022

Figure 11-40: Gold Control Chart for CDN-CM-7

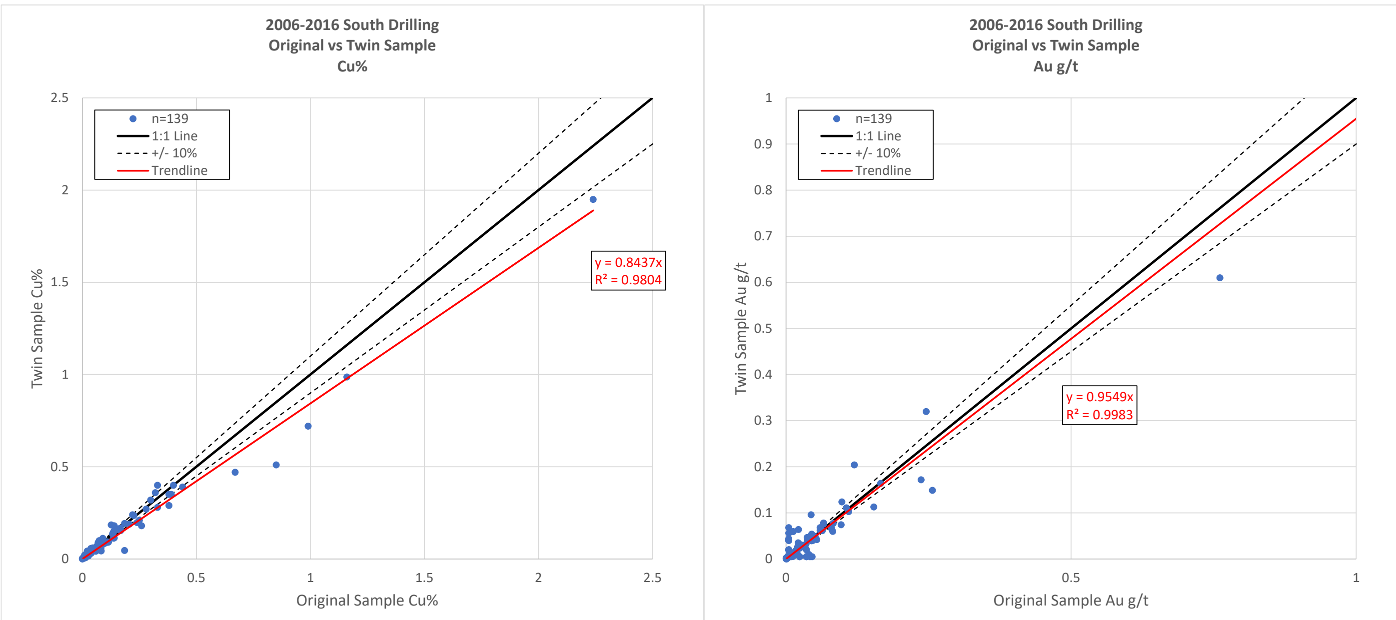


Source: Ridge Geosciences, 2022

11.1.5.6.2 Twin Samples

One hundred and thirty-nine pairs of quarter core were inserted into the sample stream as twin samples. The scatter plot of paired copper and gold values is shown in Figure 11-41. The values show reasonable correlation about the 1:1 line. A few samples with original assays values >0.5% copper returned more than 10% lower. This is not considered material, and the comparison is acceptable for quartered core twin samples.

Figure 11-41: Comparison of Original vs Quartered Core Twin Samples



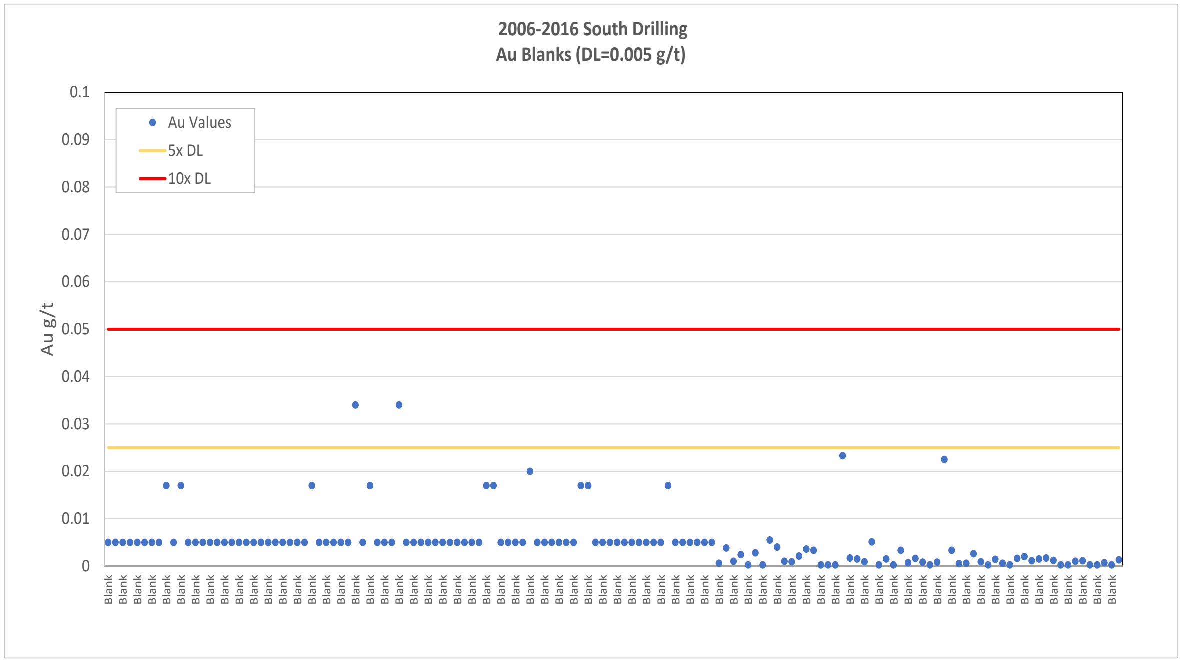
Source: Ridge Geosciences, 2022

11.1.5.6.3 Blanks

A total of 140 blanks were included in the QC samples from 2008 to 2010 drilling at South Zone. The blank material is a pre-crushed prepackaged blank from CDN Resources Laboratories Inc.

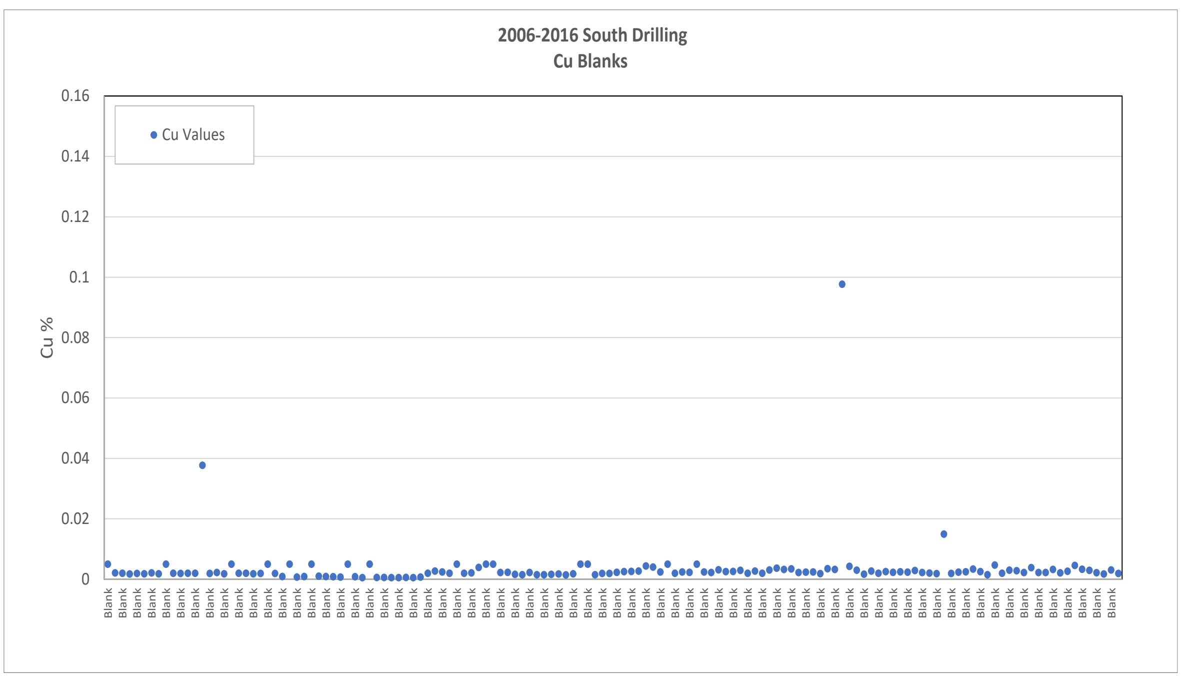
Figure 11-42 and Figure 11-43 illustrate the gold and copper blank charts. Only two gold assays return values above five times the detection limit. Three copper assays return values greater than 0.01%, with the highest value being less than 0.1%. Blanks perform very well overall with no indication of contamination.

Figure 11-42: Results of Gold Blank Analyses for 2008-2010 Drilling at South Zone



Source: Ridge Geosciences, 2022

Figure 11-43: Results of Copper Blank Analyses for 2008-2010 Drilling at South Zone



Source: Ridge Geosciences, 2022

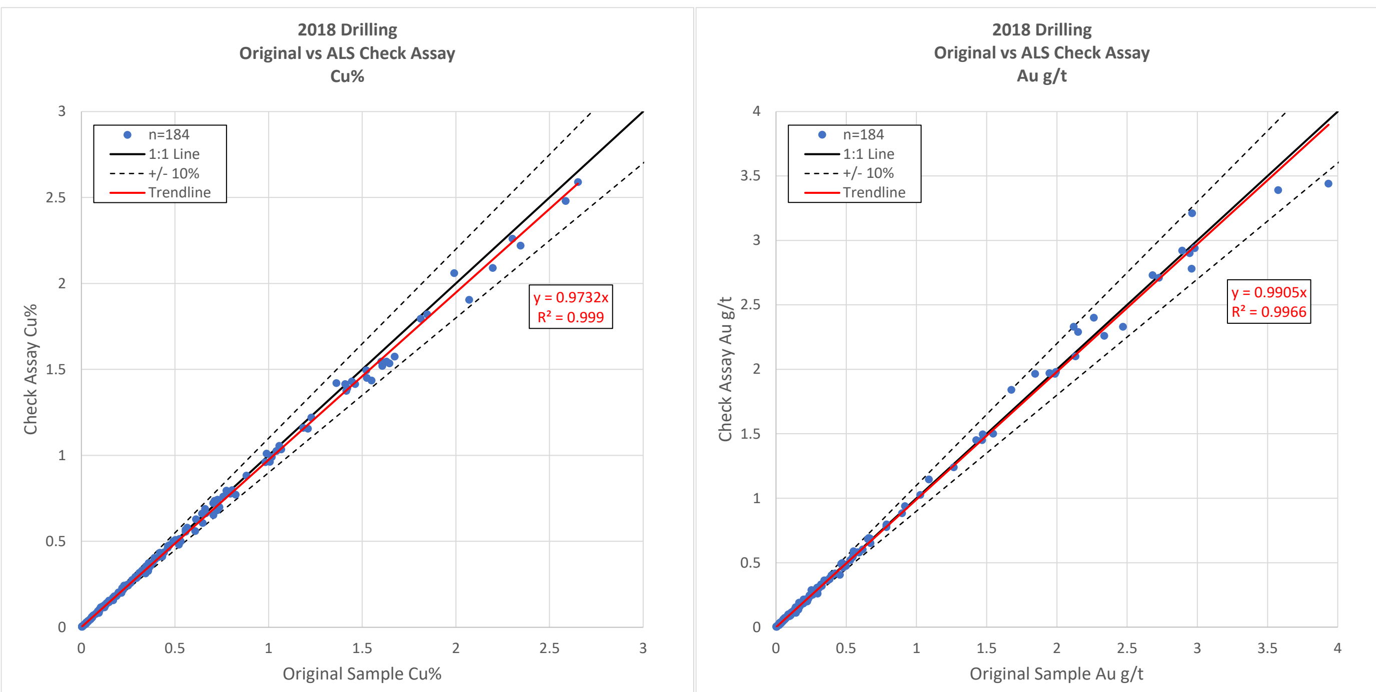
11.1.5.7 Check Assays from 2018 Drilling at Central Zone

A set of 184 primary sample pulps were processed at ALS laboratory in Vancouver, B.C. in February 2019, representing around 5.7% of the primary samples. These samples are used to assess the assay accuracy of the primary laboratory relative to a secondary laboratory.

In addition to the primary samples, 17 samples of five different CRMs were included in the check assay submittals. All samples except for one had assay values that fell within the ±2 SD from the expected value for both Cu and Au. The one sample that fell outside of this range, was a sample of CDN-CM-40, which had previously been identified as being problematic due to multiple assays by BV out of range, especially for Cu. Seven pulp blanks were also inserted into the sample stream with results showing no assays over the threshold limit for Au and assays not exceeding 0.01% for Cu.

The check assay values were compared against the assay values reported by BV. The results indicate confidence in the BV assays as demonstrated by reasonable correlation to a third-party laboratory. Figure 11-44 below show scatter plots of original versus check copper and gold values, respectively, showing good correlation between the two laboratories.

Figure 11-44: Comparison of Original vs Check Assays for 2018 Drilling at Central Zone



Source: Ridge Geosciences, 2022

11.1.6 Comment

Three CRMs (CDN-CSG-18, CDN-CGS-23, and CDN-CM-40) have been flagged as having potential problems with the standard material or data entry errors that are not attributable to the laboratory analysis. Aside from these materials, performance of the CRM analyses is adequate and does not indicate any significant or systematic bias in the copper or gold assays. Twin samples demonstrate reasonable correlation for quartered core. Blank analyses show no contamination during assaying. Check assays support the accuracy of the copper and gold values relative to a third-party laboratory. Overall performance of the quality control samples demonstrates that the quality of the database is adequate for use in the resource estimation.

The addition of coarse blanks to assess contamination during sample preparation, coarse duplicates to test sub-sampling precision, and pulp duplicates to assess analytical precision is recommended to ensure a comprehensive quality control program. Additionally, the practice of sending check assay to a third-party lab should be continued.

11.2 Stardust

11.2.1 Sampling Methods

11.2.1.1 Soil Samples

Soil samples were collected with a tree planting shovel or soil auger and placed in a kraft paper bag labelled with a sample number and containing the corresponding pre-numbered analytical tag provided by BV. In instances where field duplicate samples were taken, the sample was divided by hand and placed in a separate kraft bag with unique sample number for analysis. Kraft bags were folded shut and placed in a carboard box for shipping. Sampling targeted B and C Horizon soils. Sample locations were recorded using a handheld GPS and field marked with flagging tape labelled with the sample number.

11.2.1.2 Rock Samples

Rock samples were collected by taking selected pieces of rock from outcrop, subcrop, and float using a rock hammer. All samples were placed in a poly bag labelled with the sample number and containing the corresponding pre-numbered analytical tag provided by Bureau Veritas. Poly bags were sealed using a nylon cable tie and placed in rice bags for shipping. Sample locations were recorded using a handheld GPS, and field marked with flagging tape and an aluminum tag labelled with the sample number.

11.2.1.3 Drill Core

Drill core sample intervals were laid out and recorded by the logging geologist on site based on lithology and mineralization noted. Sample locations and associated sample numbers were marked on the core using a red lumber crayon. Pre-numbered three-part analytical tags provided by BV were stapled into the core boxes at the end of each sample.

Drill core was cut using an electric powered rock saw. Samples were cut in half lengthwise. One half was returned to its original location in the core box. The other half was placed in a poly sample bag pre-labelled with the sample number. Two sections of the analytical tag were placed in the pre-labelled polyethylene (poly) bag with the corresponding sample number. One section of the analytical tag remained stapled to the core box. In instances where field duplicate samples were taken, the sampled half core was re-sawn lengthwise to produce two quarter core samples. Each quarter core sample was placed in a separate poly bag with unique sample number for analysis. Poly sample bags were sealed using a stapler and placed in rice bags for shipping. Rice bags were sealed using numbered locking security ties.

11.2.2 Density Determinations

Specific gravity measurements were taken on 9,159 core samples from the 2018, 2019 and 2020 drill programs using the water immersion method. The measurements were carried out by Sun Metals geotechnical personnel on-site using a digital scale.

11.2.3 Analytical and Test Laboratories

All core and geochemical samples from 2017 were analyzed at Bureau Veritas Minerals Laboratory in Vancouver (BV), an ISO:9001 Certified lab. BV is independent of the Company and Sun Metals.

11.2.4 Sample Preparation and Analysis

Analytical methods used by BV are presented in Table 11-9.

Table 11-9: Analytical Methods – Bureau Veritas

Procedure	Lab Code	Description
Soil Preparation	SS80	Dry at 60°C
		Sieve to -180 µm (80 mesh)
Soil Analysis	AQ200	0.5 gram sample
		Aqua regia digestion
		ICP-MS analysis
Drill Core/Rock Preparation	PRP70-250	Crush to ≥70% passing 2 mm
		Pulverize 250 g to ≥85% passing 75 µm (200 mesh)
Drill Core/Rock Analysis	MA270	0.5 gram sample
		4 Acid digestion
		ICP-ES/ICP-MS analysis
Gold Fire Assay	FA330	30 gram sample
		Fire assay fusion
		ICP-ES analysis
Overlimit Gold/Silver	FA530	Automatic for any samples >10 ppm Au or >100 ppm Ag
		30 gram sample
		Fire assay fusion
		Gravimetric finish
Overlimit Copper	GC820	Automatic for any samples >10,000 ppm Cu
		Copper Assay by Classical Titration
Overlimit Zinc	GC816	Automatic for any samples >10,000 ppm Zn
		Zinc Assay by Classical Titration
Overlimit Lead	GC817	Automatic for any samples >10,000 ppm Pb
		Lead Assay by Classical Titration

Soil samples were dried at 60°C and sieved to 180 microns (80 mesh). Each sample was analyzed for 36 elements using modified aqua regia digestion (1:1:1 HNO_3:HCl:H_2O) and ICP-MS finish.

Rock and drill core samples were crushed to ≥70% passing 2 millimetres and pulverized to ≥85% passing 75 microns (200 mesh). Each sample was analyzed for 41 elements using multi-acid digestion with ICP-ES and ICP-MS finish. Fire assay fusion decomposition with ICP-ES analysis was also completed on each sample to determine gold-platinum-palladium content. Samples containing gold, silver, copper, zinc, or lead above the detection limit of these techniques were automatically reanalyzed. Samples containing >10 ppm gold or >100 ppm silver were reanalyzed by fire assay fusion with a gravimetric finish. Samples containing >10,000 ppm copper, zinc, or lead were reanalyzed by titration.

11.2.5 Quality Assurance and Quality Control

11.2.5.1 Drill Core QA/QC

Diamond drill core samples had standard and blank reference material inserted into the sampling series at regular intervals. The certified ranges for the blank and standards used are summarized in Table 11-10.

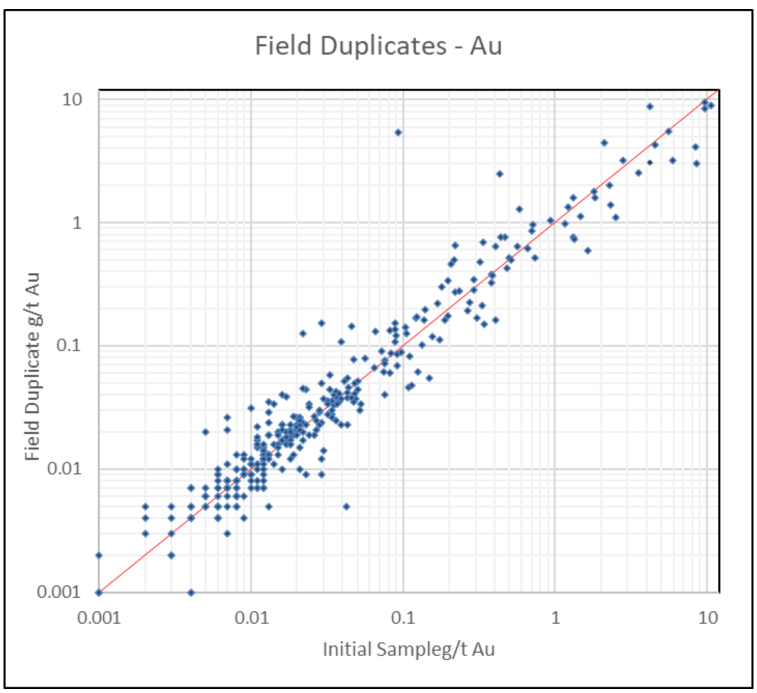
Table 11-10: Stardust Certified Reference Material Expected Values

Standard	Au	Ag	Cu	Pb	Zn
CDN-ME-1312	1.27	22.3	0.446	0.273	1.81
CDN-ME-1410	0.542	69	3.8	0.248	3.682
CDN-ME-1708	6.96	53.9	2	0.171	0.484
CDN-BL-10	<0.01				

Field duplicates were also taken at regular intervals. In sections of high-grade mineralization, the frequency of insertion of reference material and field duplicates was increased. Additional reference material samples and field duplicates were also added at the discretion of the logging geologist on site. The results indicated no significant problems with the laboratory analysis. A review of BV's QAQC data – duplicate analysis, standards, blanks, and prep washes also indicate no significant problem with the laboratory analysis.

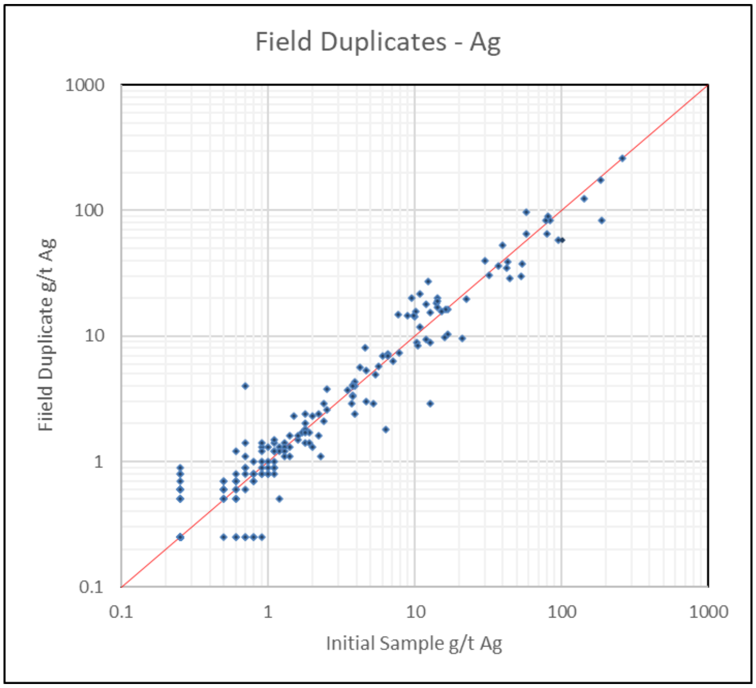
Correlation between field duplicate core samples is generally strong (Figure 11-45 to Figure 11-47). Increased variability is noted in returned gold and silver analytic results <1 ppm. Minor variability is noted in copper results throughout the range of returned results. These inconsistencies are interpreted to be due to the irregular nature of mineralization in skarn and CRD systems and local relative coarseness of commodity bearing minerals in these systems.

Figure 11-45: Log Scatterplot of Field Duplicates - Au



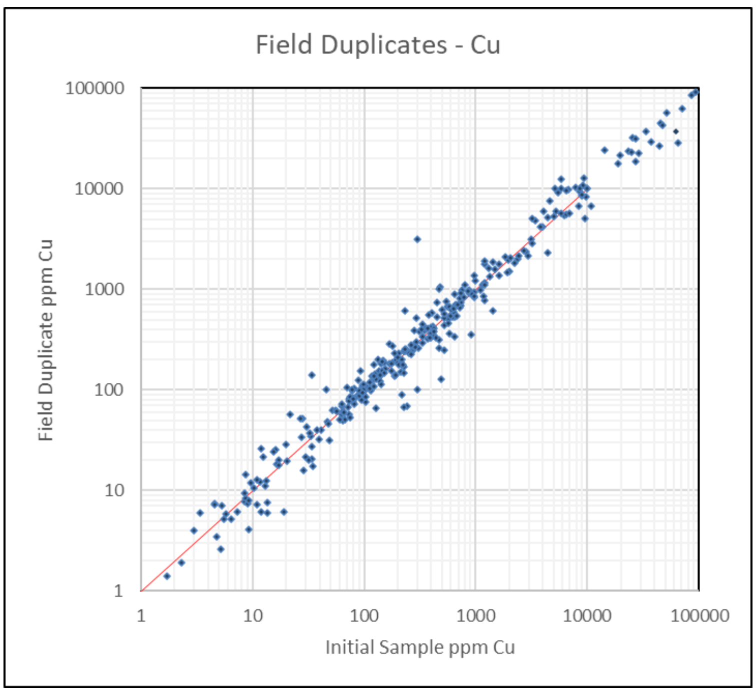
Source: Ridge Geosciences, 2022

Figure 11-46: Log Scatterplot of Field Duplicates - Ag



Source: Ridge Geosciences, 2022

Figure 11-47: Log Scatterplot of Field Duplicates - Cu



Source: Ridge Geosciences, 2022

11.2.5.2 Soil Sampling QAQC

Soil samples had blank reference material inserted into the sample sequence around every 20 samples. Field duplicates were taken around every 35 samples. The results indicated no problems with the laboratory analysis. A review of BV's QAQC data – duplicate analysis, standards, blanks, and prep washes also indicate no significant problem with the laboratory analysis.

11.2.5.3 Rock Sample QAQC

All rock samples passed BV's internal reference material and duplicate QAQC protocols. Results from duplicate analysis, standards, blanks, and prep washes indicate no significant problem with the laboratory analysis.

11.2.6 Sample Security

Drill core was brought from the drill to the core logging facility by either the drillers or the project geologist. On site, the core was kept in and around the core logging tent, where it was logged by the geologist and sample intervals laid out.

Rock and drill core samples were placed in labelled rice bags and sealed using numbered locking security ties for shipping. Rice bags were labelled with a unique identification number and list of samples contained within. Soil samples were placed in cardboard boxes labelled with a unique identification number and list of samples contained within and sealed with packing tape for shipping. Each batch of samples shipped to BV was given a unique shipment identification.

Samples were delivered by Sun Metals personnel to Bandstra Transportation Systems Ltd. (Bandstra) in Prince George, B.C. Bandstra personnel complete a certified bill of landing for each sample shipment and maintain a complete chain of custody of samples until delivered to BV.

At all times samples were under the control of Sun Metals personnel until delivered to BV. BV catalogues all received samples and maintains a complete chain of custody of each sample through the analytical process.

For soil samples, sample depth, soil horizon and soil colour and relevant notes were recorded for each sample. Samples were placed in Kraft bags labelled with the grid location, were dried in the Tsayta Lake Lodge core shack, and were put in ~12x11" size cardboard boxes and shipped to BV via courier.

Rock samples were placed in polybags and taken back to camp, where hand specimens were separated from the original sample. Sampler, location, field description, source and source size, sample type, rock type, mineralization and alteration were recorded for each sample. Samples were batched in rice bags and sent via courier to BV for assay.

11.2.7 Databases

Data is collected and stored using a Geospark database. The project manager is responsible for maintenance of the database. Raw datafiles in CSV format provided by the lab are imported directly into the database using a built-in customizable import template. The project manager checks for any QA/QC discrepancies through a reporting function in the database upon import.

11.2.8 Comments

Mr Simpson is of the opinion that the adequacy of sample preparation, security and analytical procedures for the Stardust project are sufficiently reliable to support the mineral resource estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices at the time of collection.

12 DATA VERIFICATION

12.1 Kwanika

12.1.1 Site Visit

A site visit was completed to the project area on September 20, 2022, by Mr. Jason Blais, P. Eng. of Mining Plus. All relevant data and procedures for measuring, capturing, recording, and storing were reviewed, and drill log and assay certificates were compared. Presentations of drill core and geological interpretation were made by senior site geologists. Figure 12-1 shows the interior of the core logging facility and Figure 12-2 shows the typical core storage yard.

Figure 12-1: Kwanika Core Logging Facility

Source: Mining Plus, 2022

Figure 12-2: Kwanika Typical Core Storage

Source: Mining Plus, 2022

Three representative drillholes were examined during the site visit, including K21-207, K21-217, K22-255. Items noted included:

- Drill core condition
- Sample selection
- Core recovery
- Assay certificates
- Logging, sampling, and core handling procedures.

Drill logs and assay certificates were compared as a part of this process. No significant discrepancies were identified.

Following observations and discussion at the main project site and core logging facility, several drillhole collars relating to Kwanika Central and Kwanika South were located, and coordinates verified by handheld GPS.

12.1.2 Digital Record Storage

Since 2021, the drillhole database is stored in MxDeposit. The database is stored on a cloud server. Daily (weekly) backups of the database are completed and stored on the network server. Remote users have full access to the database in real time.

Data are easily distributed as .csv files for use in various software packages.

12.1.3 Database Validation and Verification

Since 2001, drill core is logged directly into MxDeposit using project-specific geological codes to maintain consistency in results and conclusions. Automatic validations and table look-ups are also incorporated into the database to ensure the integrity of the data being loaded. A database manager oversees the data capture process, and well as importing external data such as laboratory assay results, into the database.

Drillhole collar elevations have been validated against a topographic surface generated from a LiDar survey flown in 2016. Drillhole traces were visually checked to validate the downhole surveys.

Independent database audits have been completed several times since 2006 to check the database for errors, as noted in prior Technical Reports. As part of this study, Mining Plus has further verified 5% of the assay database against original assay certificates and found no errors. Comments on Data Verification

Mr. Brian Hartman has reviewed NorthWest Copper's database management practices, on-site procedures and protocols, quality control procedures and analyses, and checks of the assay database against assay certificates. Mining Plus finds the database integrity and QA/QC program to be acceptable for mineral resource estimation and preparation of this PEA.

12.2 Stardust

12.2.1 Site Visit

Ronald Simpson, P. Geo. of GeoSim has visited the Stardust project site on three occasions with the most recent visit being conducted on September 23, 2020. Previous visits were carried out on June 14, 2010, and September 17, 2017. During the sites visits, copper-bearing sulphide mineralization in drill core and outcrop were visually identified. A number of drill sites were checked by GPS and found to be accurate.

12.2.1.1 Drillhole Location

Drillholes are surveyed by an RTK DGPS system. The author checked several drill sites by handheld GPS, and they were found to be accurate. Drill sites have been reclaimed and the drillhole position marked with stakes.

12.2.1.2 Drill Core Logging

A core logging facility is on site. It was found to be clean and well maintained. Inclined benches were used to display core for mark-up and logging. A dedicated digital camera mount attached to a computer was used for core photography (Figure 12-4).

Specific gravity measurements were taken on drill core using the water immersion method.

Figure 12-3: Stardust Core Logging Facility (Sept 2020)

Source: Geosim, 2020

Figure 12-4: Stardust Core Photography Station (Sept 2020)

Source: Geosim, 2020

12.2.1.3 Validation of Sampling and Core Storage Facilities

The core storage facility is located beside the core shack. Core boxes are marked with metal tags and stacked on pallets.

A separate room attached to the core shack is used for sawing and bagging core samples and insertion of certified reference standards and blanks.

12.2.1.4 Independent Sampling

During the site visit on June 14, 2010, several core samples were collected and submitted for analysis. Results of the sample from hole LD200913 were consistent with the initial values obtained from the assay interval of 2.82 g/t Au, 62.1 g/t Ag and 3.13% Cu (Table 12-1). The limonitic material from near the top of hole LD200513 was not previously sampled.

Stardust Independent Sampling Results

Table 12-1: Stardust Independent Sampling Results

Hole	Depth (m)	Au (g/t)	Ag (g/t)	Cu (%)	Description
LD200913	159	4.488	95	4.733	
LD200513	31-32	0.054	3	0.050	Leached limonitic zone
Outcrop		1.302	44	2.311	Roadcut in N CCS Area

The author has also visually identified mineralization in drill core consistent with reported analytical results on more recent site visits and does not consider further independent sampling necessary.

12.2.2 Database Validation

The author independently audited the sample database for location accuracy, downhole survey errors, interval errors and missing sample intervals. The author also reviewed the sample QA/QC results.

12.2.3 Comments

Based on the site visit observations, the QP concludes that drilling, logging, and sampling of drill core during the drilling and exploration programs carried out by NorthWest Copper and previous operators have been conducted in a manner appropriate to the style of mineralization present on the property.

Mr. Ronald Simpson has reviewed NorthWest Copper's database management practices, on-site procedures and protocols, quality control procedures and analyses, and checks of the assay database against assay certificates. Mining Plus finds the database integrity and QA/QC program to be acceptable for mineral resource estimation and preparation of this PEA.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Kwanika and Stardust deposits have been the subject of several metallurgical testwork programs from 2008-2022 that focused on comminution, gravity, and flotation. Three metallurgical testwork programs have been performed on material from the Kwanika deposit, and the Stardust deposit has been the subject of its own metallurgical scoping study as described in Section 13.2. The testwork examined the following:

- head assays
- mineralogy
- comminution tests
- flotation tests
- gravity tests

13.2 Metallurgical Testwork

The earliest test program was performed by SGS Canada Inc. (SGS) in 2008 – 2009 on material from the Kwanika Central deposit. The scope of this testwork included chemical analysis, mineralogical characterization, gravity concentration, and batch and locked cycle flotation testing. The next testwork program was completed by Bureau Veritas Commodities Canada Ltd. (BV Minerals) and by ALS Metallurgy Kamloops (ALS) in 2018-2019. The objectives of this program were to establish optimized copper and gold recoveries for flotation design and determine grindability and head chemical and mineralogical characteristics of the material from the Kwanika Central deposit.

The Stardust scoping study was performed by Base Metallurgical Laboratories Ltd. (Base Met) in 2020-2021. This program focused on the Stardust deposit to generate head assays and a process flowsheet for material from Stardust. A subsequent test program was completed at Base Met in 2022 to evaluate flotation performance of Kwanika Central samples in blends with portions of Stardust samples.

The full body of testwork is summarized in Table 13-1.

Table 13-1: Metallurgical Testwork Summary Table

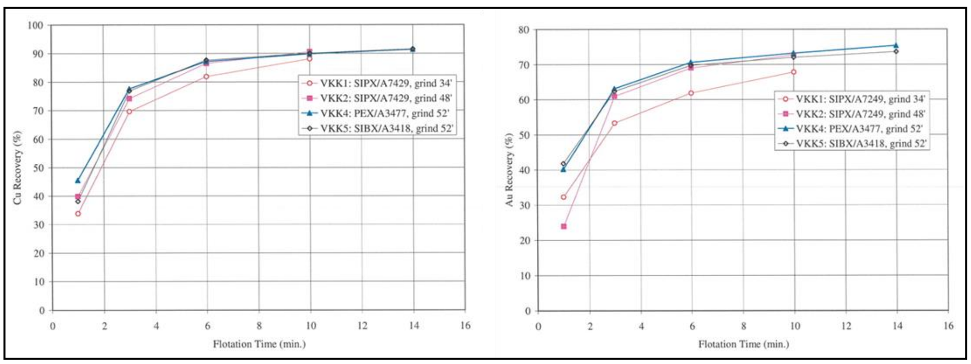
Year	Test Programs	Laboratory
2008-2009	The Recovery of Copper and Gold from Kwanika Deposit	Vancouver Metallurgy, SGS Minerals Services
2009	Follow-up Testwork Summary on Kwanika Deposit	Vancouver Metallurgy, SGS Minerals Services
2018	Comminution Tests on Kwanika Deposit	ALS Metallurgy Kamloops
2018-2019	Prefeasibility Metallurgical Testing to Recover Copper and Gold on Kwanika Deposit	Metallurgical Division, Bureau Veritas Commodities Canada Ltd.
2020-2021	Metallurgical Scoping Study of the Stardust Project	Base Metallurgical Laboratories Ltd.
2022	Metallurgical Assessment of Samples from the Kwanika/Stardust Project	Base Metallurgical Laboratories Ltd.

13.3 2008 – 2009 SGS Program Summary

SGS received around 185 kg of material comprising 52 individual samples. All 52 of these samples were mixed to create 120 kg of master composite for testing. Head assay of the master composite returned values of 0.66% copper, 0.76 g/t gold, and 1.8% sulphur. This composite sample was higher grade than composites produced for subsequent testwork. Additionally, because only a single sample was created, the SGS program does not capture the variability of the Kwanika deposit. Copper and gold were determined to be the only economically recoverable elements. QEMSCAN analysis revealed the copper to be relatively fine-grained, which was interpreted as requiring a primary grind of 75 µm. The ball mill grindability test determined a Bond ball mill work index (BWi) of 16.2 kWh/t.

SGS metallurgical testing utilized rougher and cleaner batch flotation, locked cycle flotation, and gravity testing. Gravity recovery results did not warrant the inclusion of gravity concentrators in the flowsheet. Rougher flotation was performed on P_{80} grind sizes of 133 µm, 87 µm, and 75 µm. The optimal conditions were considered to be a P_{80} of 75 µm, flotation time of 14 minutes, and a natural pH of 7.9. Under these conditions, the rougher concentrate was able to achieve copper recovery of 91.5% and a gold recovery of 75.3%.

Figure 13-1: Flotation Kinetic Curves



Source: SGS January 26, 2009.

Cleaner flotation tests targeted a copper concentrate with a grade of 25-30% after three stages of cleaning. Rougher flotation concentrate was reground to 20 µm, 26 µm, and 32 µm to find the optimum regrind size. At 25 µm, the target grade was achieved with a copper recovery of 82-85%. Locked cycle tests attained a final concentrate grade of 27.7% copper with 88.5% copper recovery. At this concentrate grade, 65.2% of gold was recovered to the final concentrate. The tests showed that the penalty elements included in the final concentrate were very low.

13.4 2009 SGS Follow-up Program Summary

This follow-up testwork program was conducted to assess the potential of recovering gold from the rougher flotation tails. This was achieved by investigating the impact of regrind and scavenger flotation of rougher tails, cyanidation of rougher tails, and determination of gold distributions in different size fractions of rougher tails. Regrind and scavenger flotation of rougher tails was able to recover an additional 10% of the gold present in the sample. This brought the combined gold recovery up to 82.9%. The cyanidation of rougher tails was able to extract an additional 15.6% of the gold after 24 hours of leach time. The gold distribution assays determined that gold in the rougher tailings was distributed evenly in each size fraction. Fine grinding is able to liberate more gold and improve gold recovery, but achieving this additional recovery is not necessarily economical.

13.5 2018-2019 BV Minerals Program Summary

13.5.1 2018-2019 BV Minerals Program Sample Selection

Samples and compositing for this testwork program are summarized below. The samples generated for this test program included three master composite samples representing the mineralization of the open pit, west underground, and east underground. Also included in this test program were four grindability test rejects samples and eleven variability samples from various lithological zones from the Kwanika deposit. The condition of the assay reject samples is unknown. Assay rejects are generally not used for metallurgical testing, if possible, as the sulphide minerals could oxidize somewhat during storage due to the fine crush size. There is some level of uncertainty if any oxidation occurred with these samples or if it affected the flotation performance.

Table 13-2: BV Minerals Sample Summary

Sample Receiving Date	Sample Description	Composite ID
October 16, 2018	Four pails of half core splits "2018 PFS Metallurgy Kwanika SAG sample"	75% Hardness Composite OP High-Grade Composite OP Low Grade Composite Underground Composite
January 10, 2019	Rice bags with 40 individual assay reject bags	OP/HG Master Composite
March 22, 2019	2x15 individual assay reject bags	UG – East Block Master Composite UG – West Block Master Composite
April 16, 2019	Three bags of half core splits	PEA OP Composite UG Tall Composite UG-West-Composite
June 3, 2019	Two pallets with 12 pails of half core splits and nine rice bags with 20 individual bags of half core splits	2019 OP-High-Composite 2019 UG-East-Composite 2019 UG-West-Composite 20 Variability Samples

13.5.2 2018-2019 BV Minerals Program Head Assay

Master composites, grindability samples, and variability samples were all assayed. The results of these assays are presented in Tables 13-3 and 13-4.

Table 13-3: BV Minerals Master Composites Head Assay Results

Analyte	Unit	Composite ID					
		OP/HG Master Comp	East Block Comp	West Block Comp	2019 OP/HG Comp	2019 UG East comp	2019 UG West Comp
Au	g/t	0.75	0.49	0.62	0.76	0.52	0.65
Cu	%	0.69	0.45	0.56	0.73	0.51	0.56
Cu-OX	%	0.044	0.025	0.035	0.033	0.029	0.032
FeO	%	5.07	4.54	5.42	4.91	4.86	5.39
Fe^{3+}	%	1.9	1.8	1.3	1.7	1.6	1.2
Fe^{2+}	%	1.83	1.76	1.24	1.81	1.85	1.49
TOT/S	%	1.83	1.76	1.24	1.81	1.85	1.49
S/S^{2-}	%	0.73	0.76	0.24	0.79	0.78	0.38

Table 13-4: BV Minerals Variability Samples Head Assay Results

	Sample ID	Au g/t	Cu %	Cu-OX %	FeO %	Fe3+ %	Fe2+ %	TOT/S %	S/S^{2-} %
Grindability Samples	75% Hardness	1.10	1.15	0.039	4.22	1.3	3.3	1.94	0.91
	Open Pit High	1.26	1.08	0.041	4.94	1.3	3.8	1.58	0.53
	Low Open Pit	0.18	0.24	0.020	3.59	1.7	2.8	1.46	0.52
	Underground	0.61	0.54	0.027	4.63	1.3	3.6	1.47	0.67
Variability Samples	PEA OP	1.04	1.54	0.025	5.20	1.6	4.1	1.91	1.03
	UG West	1.96	1.54	0.039	3.63	1.0	2.8	1.64	0.81
	UG Tall	5.08	1.50	0.036	8.26	1.3	6.4	1.85	0.89
	K1818495.98 OP Low Grade Var	0.22	0.16	0.017	6.03	2.4	4.7	0.53	0.14
	K18185187.9 OP Low Grade Var	0.41	0.29	0.039	7.55	0.8	5.9	0.68	0.2
	K18187152.2 OP Low Grade Var	0.31	0.36	0.007	2.55	2.5	2.0	3.17	1.38
	K18190164 OP Low Grade Var	0.18	0.39	0.015	3.64	2.8	2.9	0.49	0.16
	K18180533.5 UG Low Grade Var	0.21	0.22	0.004	3.35	1.9	2.6	3.78	0.55
	K18190337 UG Low Grade Var	0.16	0.38	0.007	2.29	1.2	1.8	1.32	0.53
	K18181270.8 Mineralogy Var	0.57	0.83	0.067	7.55	1.2	5.9	1.25	0.39
	K18188529.3 Clay Var	0.38	0.49	0.011	2.72	1.6	2.1	1.79	0.85

13.5.3 2018-2019 BV Minerals Program Comminution Testing

BV Minerals performed Bond ball mill work index (BWi) and Abrasion Index (Ai) testing and contracted ALS Metallurgy to conduct semi-autogenous grinding mill comminution (SMC) and drop weight index (DWi) testing. The SMC and DWi tests were conducted on samples from the 2018 PFS Metallurgy Kwanika SAG samples. The BWi and Ai tests were performed on the 75% Hardness Composite. BWi tests were performed on the grindability samples as well as some of the master composites. BV Minerals' BWi and Ai results are summarized in Table 13-5.

Table 13-5: BV Minerals BWi and Ai Results

Composite ID	Bond Ball Mill Work Index (kWh/t)	Abrasion Index (g)
OP/HG Master Composite	16.9	-
East Block Composite	16.9	-
West Block Composite	17.1	-
75% Hardness	17.8	0.1810
OP High Grade	17.6	-
Underground	17.6	-

For the purpose of analysis, results for the composites created with assay rejects were rejected. The F_{80} of these composites are considered to be too fine for proper BWi testing. The discarded composites are the OP/HG Master Composite, the East Block Composite, and the West Block Composite. Excluding the assay reject composites, the average of the BWi results was 17.7 kWh/t.

The SMC results generated by ALS Metallurgy are summarized in Table 13-6.

Table 13-6: ALS Metallurgy SMC Results

Sample Designation	A	B	Axb	t_a	SCSE (kWh/t)	DWi (kWh/t)	DWi (%)	Mi Parameters (kWh/t)			SG
								Mia	Mih	Mic	
75% Hardness	71.1	0.76	54.0	0.53	8.6	4.9	27.0	15.6	10.8	5.6	2.62
OP High Grade	69.8	0.77	53.7	0.51	8.7	5.1	30.0	15.5	10.8	5.6	2.73
OP Low Grade	68.5	0.78	53.4	0.51	8.7	5.0	29.0	15.5	10.8	5.6	2.69
Underground	61.7	0.93	57.4	0.53	8.5	4.8	27.0	14.6	10.1	5.2	2.78

13.5.4 2018-2019 BV Minerals Program Mineralogical Testing

Five composites representing the open pit, west underground, east underground, 75% hardness material, and high-grade underground were analyzed using Bulk Mineralogy Analysis (BMA) and Particle Mineralogy Analysis (PMA). QEMSCAN was used to identify minerals, liberations, and bulk associations. The results of the mineralogical study are presented in Table 13-7.

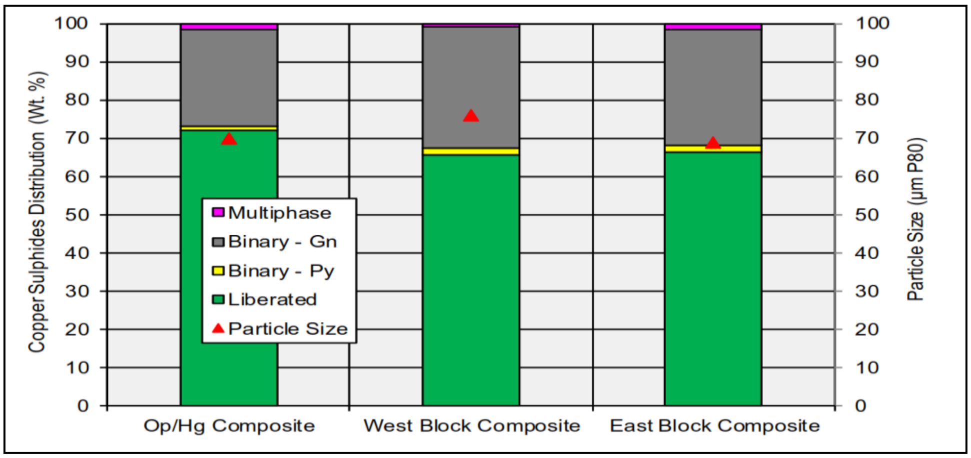
Table 13-7: BV Minerals Mineralogical Composition Results

Sulphide Minerals	OP/HG Comp (%)	West Comp (%)	East Comp (%)	75% Hardness (%)	UG Tall (%)
Chalcopyrite	1.69	0.82	0.76	1.79	2.81
Bornite	0.16	0.27	0.19	0.13	0.60
Chalcocite	0.05	0.08	0.05	0.48	0.06
Covellite	0.03	0.04	0.05	0.08	0.01
Tennantite	0.00	0.03	0.02	0.02	0.00
Molybdenite	0.00	0.00	0.00	0.00	0.00
Galena	0.00	0.00	0.00	0.00	0.00
Sphalerite	0.01	0.01	0.00	0.02	0.00
Pyrite	2.29	2.26	2.66	1.67	1.17
Total	**4.23**	**3.51**	**3.74**	**4.19**	**4.66**
Non-Sulphide Minerals	**OP/HG Comp (%)**	**West Comp (%)**	**East Comp (%)**	**75% Hardness (%)**	**UG Tall (%)**
Iron Oxides	2.3	2.5	1.9	1.1	3.0
Quartz	31.0	27.8	24.2	30.7	36.6
K-Feldspar	21.7	17.4	19.8	22.7	21.1
Plagioclase Feldspar	10.6	12.0	12.0	12.1	5.5
Muscovite/Illite	14.9	18.7	21.1	15.3	17.8
Ankerite/Dolomite	6.3	6.1	8.5	5.9	2.4
Siderite	3.0	5.2	3.5	2.3	6.1
Chlorite	1.6	2.6	1.8	1.5	1.0
Kaolinite (Clay)	1.3	0.7	0.8	0.8	0.6
Rutile/Anatase	0.5	0.6	0.6	0.3	0.3
Non-Sulphide Minerals	**OP/HG Comp (%)**	**West Comp (%)**	**East Comp (%)**	**75% Hardness (%)**	**UG Tall (%)**
Apatite	0.5	0.6	0.7	0.6	0.4
Others	2.6	2.9	2.1	3.1	1.3
Total	**96.3**	**97.0**	**96.9**	**96.4**	**95.8**

The Kwanika deposit showed low sulphide mineralization with only 3.5-4.7% sulphide minerals by weight. The copper in the deposit is fine-grained with most of the copper contained in chalcopyrite, and the remaining copper contained in chalcocite, covellite, and bornite. The non-sulphide mineralogy also supports the SGS findings of a relatively micaceous deposit.

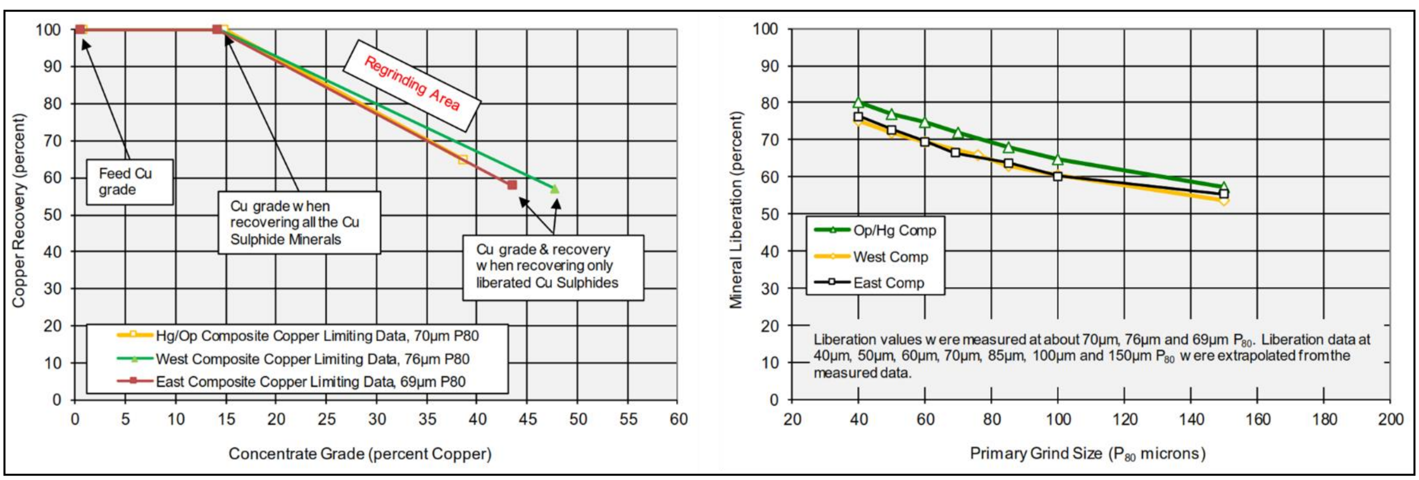
QEMSCAN was also utilized to investigate the gold losses to the rougher and cleaner scavenger tailings. Results from this analysis revealed that liberated gold and gold-chalcopyrite binaries in the tailings were all present in size fractions finer than 10 µm. Unliberated gold was associated with non-sulphide gangue. Based on these findings, it would be very difficult to economically improve gold recoveries in flotation.

Figure 13-2: Copper Sulphide Liberation by Class and Association



Source: BV August 23, 2019.

Figure 13-3: Mineral Liberation Curves

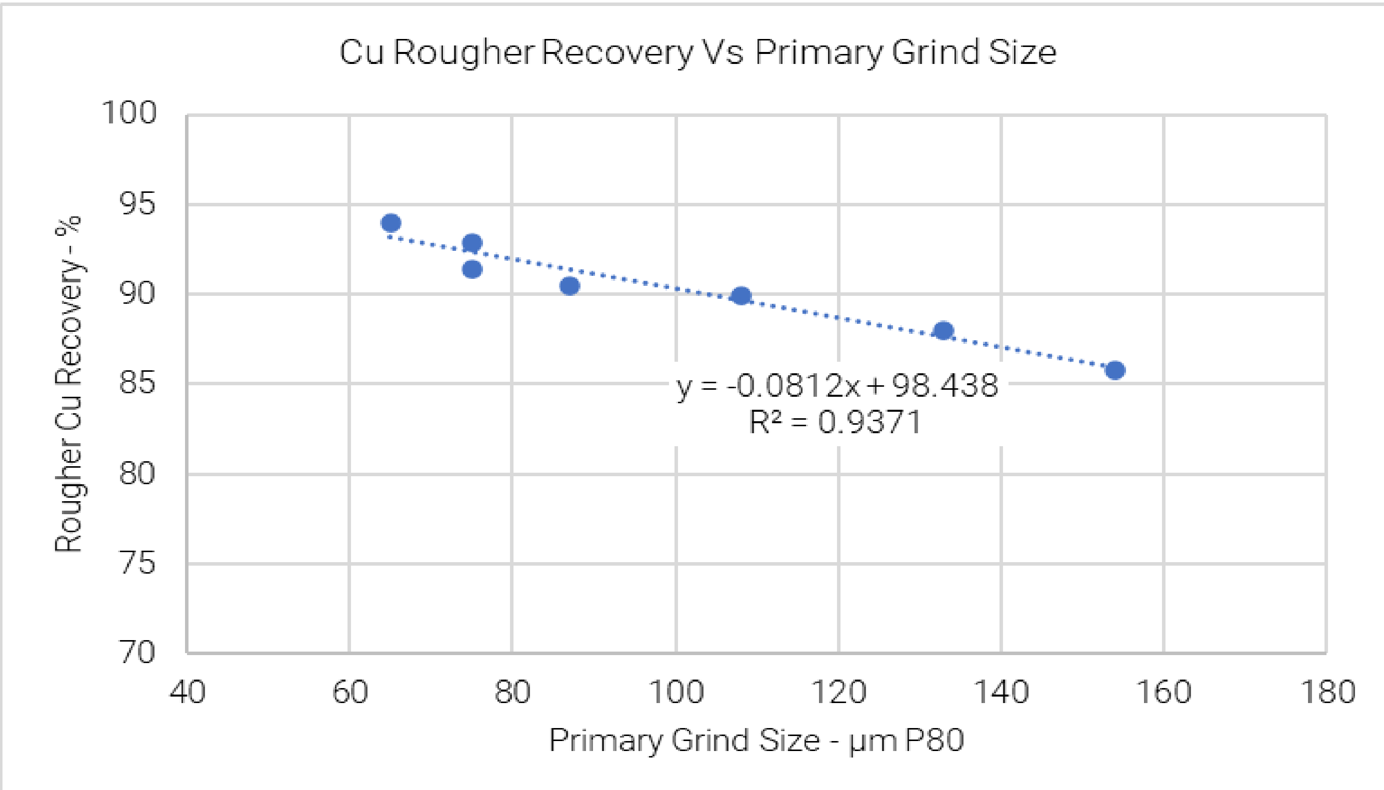


Source: BV August 23, 2019.

13.5.5 2018-2019 BV Minerals Program Flotation Testing

Flotation tests F1 to F17 were performed to optimize conditions including primary grind size, pH, and reagent scheme on the open pit high-grade master composite. This composite is meant to be representative of high-grade material sourced from the Kwanika open pit. The primary grind has been plotted against copper recovery in Figure 13-4. This graph also includes two data points from the 2008-2009 SGS test program for completeness of the analysis.

Figure 13-4: Primary Grind Size vs Copper Recovery



Source: Ausenco, 2022.

For this test program, a primary grind P_{80} of 75 µm was chosen for the flowsheet. After optimization, the flowsheet ultimately selected was that of rougher flotation and scavenger, then rougher concentrate regrinding followed by four stages of cleaner flotation.

13.5.5.1 Flotation Testing Batch Flotation

The batch flotation tests on open pit material are summarized in Table 13-8. These tests used conditions from test F11 with the addition of a rougher scavenger stage.

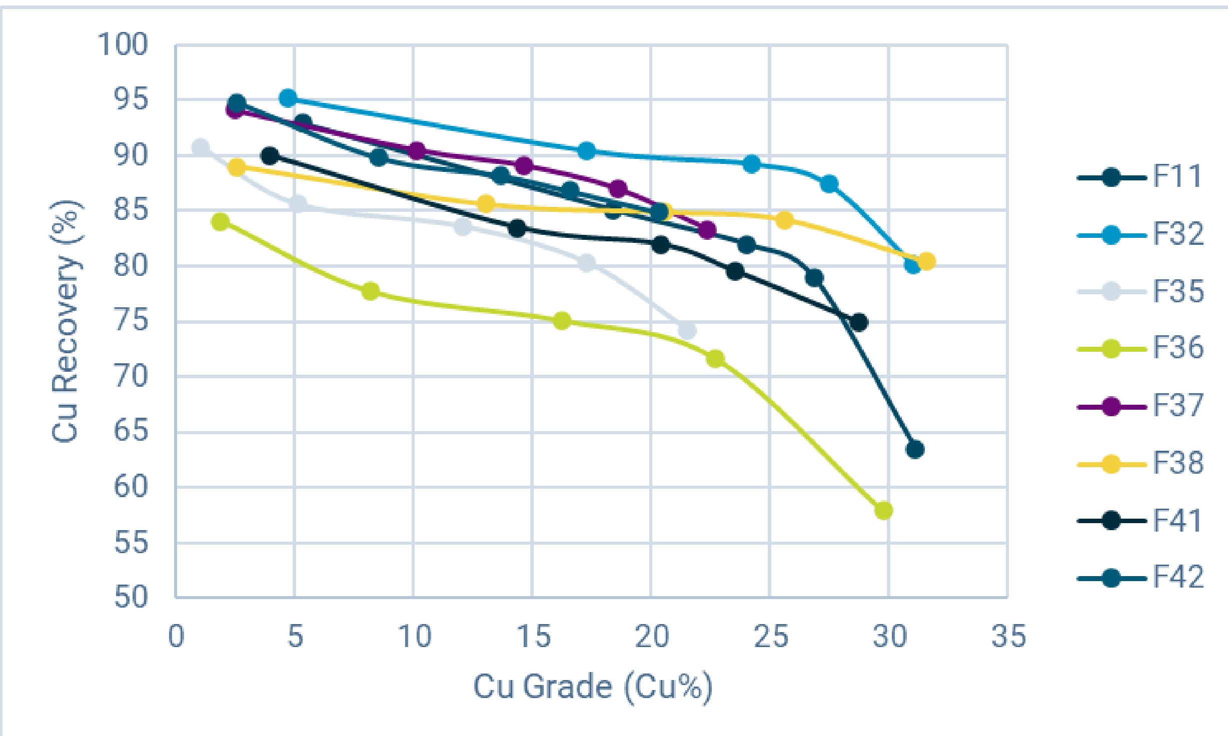
The final cleaner concentrate grade ranged from 20.3% to 31.6% copper with an average overall copper recovery of 88.5% for open pit composites. The average overall gold recovery was 72.5%.

Table 13-8: BV Minerals Open Pit Batch Flotation Results

Sample ID	Test No.	P_{80}		Head Grade (Calc)		4th Cleaner Concentrate		
		Feed (µm)	Regrind (µm)	Cu (%)	Au (g/t)	Cu Grade (%)	Recovery %	
							Cu	Au
OP/HG Master Comp	F11*	75	28	0.71	0.75	31.1	63.5	45.2
2019 OP/HG Master Comp	F32	78	28	0.74	0.79	31.0	80.1	58.5
OP Low Grade Var	F35	78	21	0.15	0.22	21.5	74.2	55.4
OP Low Grade Var	F36	74	15	0.29	0.38	29.8	57.9	38.8
OP Low Grade Var	F37	70	25	0.37	0.27	22.4	83.3	46.4
OP Low Grade Var	F38	66	21	0.38	0.18	31.6	80.4	31.6
Mineralogy Var	F41	71	25	0.83	0.57	28.7	75.0	66.2
Clay Var	F42	67	20	0.49	0.38	20.3	84.9	63.1

*F11 did not include a rougher scavenger flotation stage.

Figure 13-5: Open Pit Cu Grade vs Cu Recovery Curves



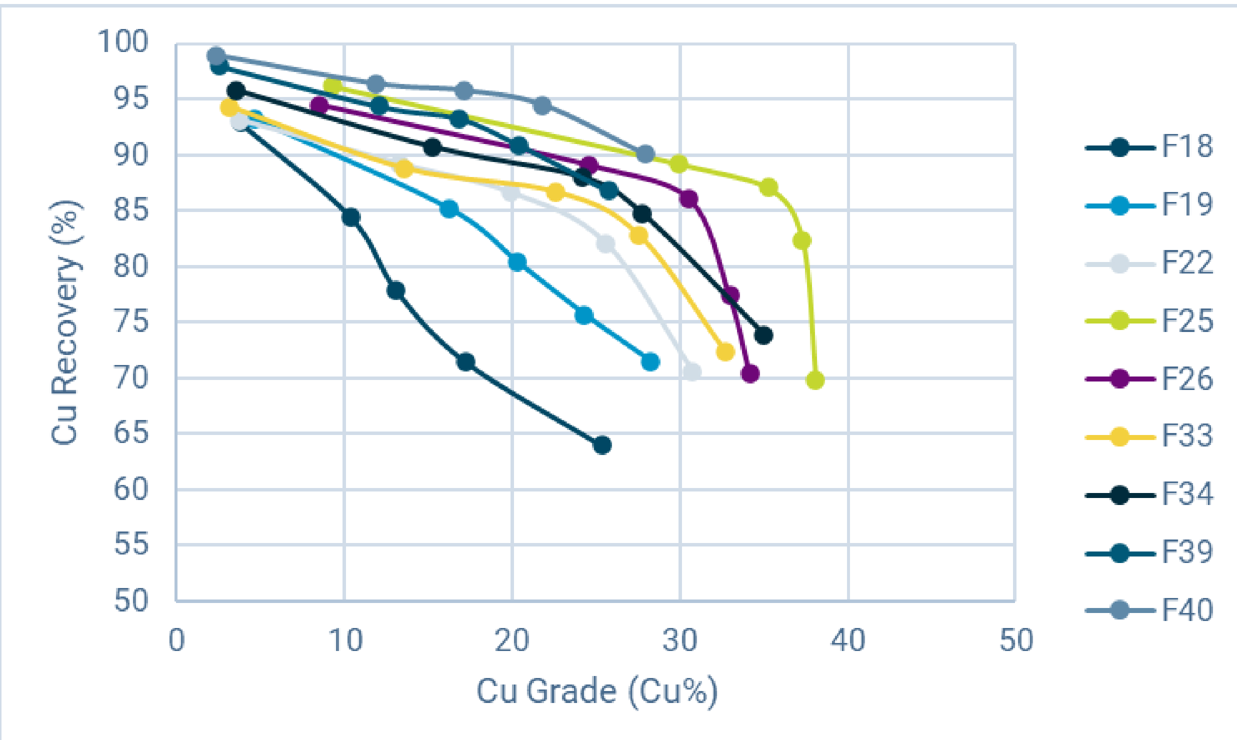
Source: Ausenco, 2022

The batch flotation tests on underground material are summarized in Table 13-9. These tests used conditions from test F11 with the addition of a rougher scavenger stage.

Table 13-9: BV Minerals Underground Batch Flotation Results

Sample ID	Test No.	P_{80}		Head Grade (Calc)		4th Cleaner Concentrate		
		Feed (µm)	Regrind (µm)	Cu (%)	Au (g/t)	Cu Grade (%)	Recovery %	
							Cu	Au
East Block Comp	F18	69	24	0.45	0.49	25.4	64.0	43.8
West Block Comp	F19	74	26	0.56	0.67	28.3	71.5	55.6
Underground	F22	72	25	0.54	0.67	30.7	70.6	52.1
UG Tall	F25	76	28	1.53	5.00	38.1	69.9	65.1
UG West	F26	70	25	1.53	1.73	34.2	70.5	64.6
2019 UG East Comp	F33	72	23	0.51	0.54	32.7	72.3	51.2
2019 UG West Comp	F34	78	24	0.55	0.72	35.0	73.9	49.6
UG Low Grade Var	F39	67	23	0.23	0.24	25.7	86.8	49.3
UG Low Grade Var	F40	67	21	0.32	0.16	28.0	90.1	45.4

Figure 13-6: Underground Cu Grade vs Cu Recovery Curves



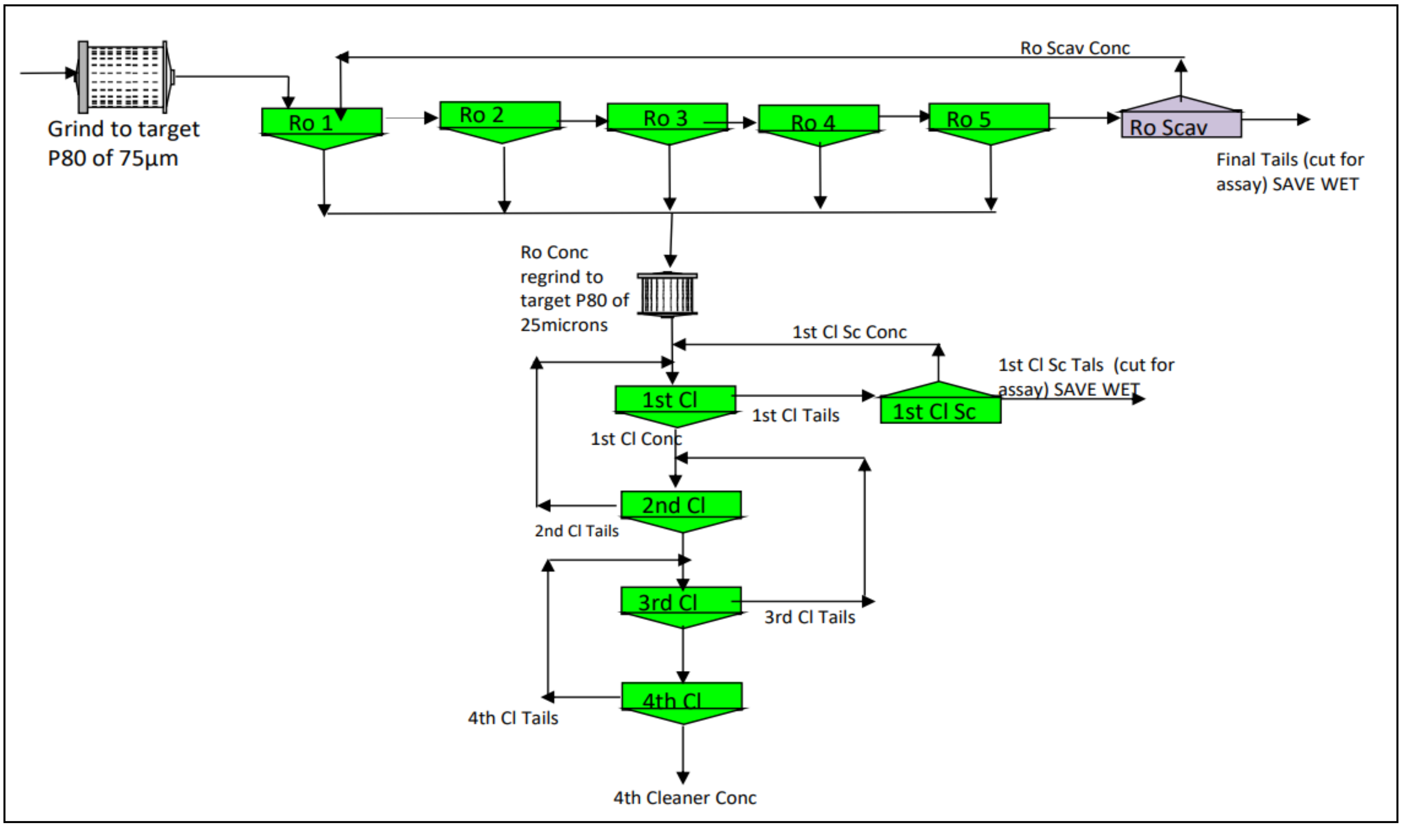
Source: Ausenco, 2022.

The final cleaner concentrate grade ranged from 25.4% to 38.1% copper with an average overall copper recovery of 90.0% for underground composites. The average overall gold recovery was 74.7%. The improved concentrate grades compared to the open pit material are due to the heightened content of secondary copper minerals present in the underground material.

13.5.5.2 Flotation Testing Locked Cycle Flotation

Locked cycle tests were performed on three of the master composite samples, OP/HG Master, UG East, and UG West. These three composite samples are meant to represent the high-grade feed from the Kwainka open pit, eastern part of the underground block cave, and the western part of the underground block cave respectively. Tests LC1-LC4 used the flowsheet shown in Figure 13-7 and are summarized in Table 13-10.

Figure 13-7: BV Minerals LC1 to LC4 Flowsheet



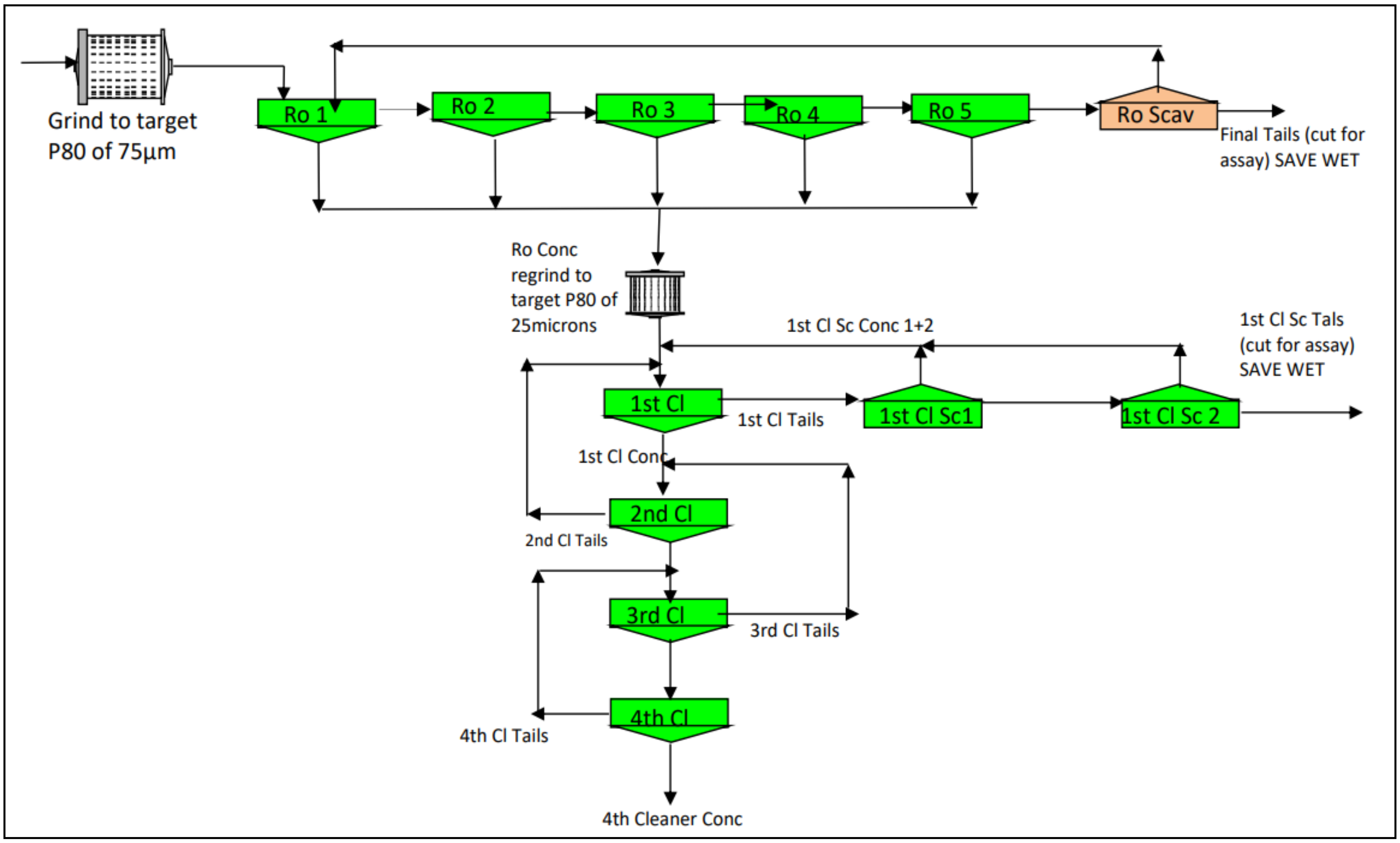
Source: BV August 23, 2019.

Table 13-10: BV Minerals LC1 to LC4 Results

Sample ID	Test No.	P_{80}		Head Grade (Calc)		4th Cleaner Concentrate		
		Feed (µm)	Regrind (µm)	Cu (%)	Au (g/t)	Cu Grade (%)	Recovery %	
							Cu	Au
OP/HG Master Comp	LC1	71	25	0.70	0.81	25.1	87.6	72.9
OP/HG Master Comp	LC2	70	25	0.75	0.81	25.1	88.9	74.0
East Block Comp	LC3	68	25	0.42	0.43	28.4	81.1	59.6
West Block Comp	LC4	70	25	0.51	0.58	32.3	82.5	69.5

The locked cycle tests were able to achieve a saleable copper concentrate grade and reasonable copper and gold recoveries. The average copper recovery attained in these tests was 85.0% and the average gold recovery was 69.0%. Additional locked cycle tests were performed on new samples obtained from half-sawn drill core. In an effort to increase recoveries, tests LC5 to LC7 included an additional stage of cleaner scavenger flotation. The flowsheet for these tests and their results is shown in Figure 13-8 and summarized in Table 13-11.

Figure 13-8: LC5 to LC7 Flowsheet



Source: BV August 23, 2019.

Table 13-11: BV Minerals LC5 to LC7 Results

Sample ID	Test No.	P_{80}		Head Grade (Calc)		4th Cleaner Concentrate		
		Feed (µm)	Regrind (µm)	Cu (%)	Au (g/t)	Cu Grade (%)	Recovery %	
							Cu	Au
2019 OP/HG Master Comp	LC5	77	25	0.74	0.82	22.0	93.9	78.4
2019 UG West Comp	LC6	76	25	0.54	0.66	25.8	89.7	72.5
2019 UG East Comp	LC7	75	25	0.54	0.50	29.0	89.7	72.5

The locked cycle tests with the additional cleaner scavenger stage achieved greater copper and gold recoveries than the previous locked cycle tests. Across these tests, the average copper recovery was 91.8% and the average gold recovery was 73.9%.

The BV Minerals locked cycle may not have reached steady state, as the copper concentrate grades were still changing in the final cycles. Sulphur contents were not measured in the tests, so it is unclear if the circulating loads of pyrite stabilized in the cleaner circuits. Due to this uncertainty, the test results are difficult to fully analyze.

13.5.6 2018-2019 BV Minerals Program Concentrate Testing

Cleaner concentrates from the locked cycle tests were assayed for potentially deleterious elements. A summary of these assay results is presented in Table 13-12. These impurities are not anticipated to incur smelter penalties as the levels of impurities are below or very near the smelting penalty thresholds applied by most smelters according to the metallurgical testwork performed to date.

Table 13-12: BV Minerals Concentrate Assay Results

Analyte	Unit	LC1 Cl Con (OP/HG Comp)	LC5 Cl Con (2019 OP/HG Comp)	LC6 Cl Conc (2019 UG West Comp)	LC7 Cl Conc (2019 UG East Comp)	Typical Penalty Levels
Cl	%	<0.08	<0.08	<0.08	<0.08	0.03
F	ppm	342	90	48	49	300
Hg	ppm	0.24	0.13	0.25	0.09	5
As	ppm	120	135	1164	942	2000
Se	ppm	280	293	342	360	300

13.5.7 2018-2019 BV Minerals Program Tailings Testing

BV Minerals sent the combined tailings from LC1 to LC4 to Pocock Industrial for solid liquid separation testing. The objective of this testwork was to generate data for the design of thickening equipment. A high-rate thickener was recommended with the operating parameters summarized in Table 13-13.

Table 13-13: Pocock Industrial Thickener Operating Parameters

Material	Feed Solids (%)	Flocculant			Net Feed Loading (m^3/m^2hr)	Predicted Overflow TSS (mg/l)	Predicted Underflow Density (%)
		Type	Dose (g/t)	Conc (g/l)			
Flotation Tailings	15.27	SNF AN901SH	29-33	0.1-0.2	3.21	150-250	61%

Pocock Industrial also performed viscosity tests on the underflow generated by the thickening tests. The data generated by these tests are presented in Table 13-14.

Table 13-14: Pocock Industrial Apparent Viscosity Results

Solids Conc (%)	Coefficient of Rigidity (Pa)	Yield Value (Pa)	Apparent Viscosity, (Pa•sec) @ the following Shear Rates:								
			5 sec^{-1}	25 sec^{-1}	50 sec^{-1}	100 sec^{-1}	200 sec^{-1}	400 sec^{-1}	600 sec^{-1}	800 sec^{-1}	1000 sec^{-1}
65.5	0.086	86.2	7.792	2.816	1.816	1.172	0.756	0.487	0.377	0.314	0.273
63.6	0.056	54.0	5.267	1.820	1.151	0.728	0.461	0.292	0.223	0.184	0.159
59.8	0.031	23.9	2.931	0.889	0.532	0.318	0.191	0.114	0.084	0.068	0.058
54.3	0.018	9.7	1.439	0.413	0.241	0.141	0.082	0.048	0.035	0.028	0.024

13.6 2020-2021 Base Met Stardust Scoping Summary

Base Metallurgical Laboratories was contracted to identify possible flowsheet options for processing material contained in the Stardust deposit. This test program was conducted separately of any results or flowsheet considerations from the earlier SGS and BV Minerals Kwanika test programs.

13.6.1 2020-2021 Base Met Head Assay

Base Met assembled three composites based on head grades. The head assays of these three LG, MG, and NHG Composites are summarized in Table 13-15.

Table 13-15: Base Met Composites Head Assay Results

Comp	Element								
	Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	S (%)	As (ppm)	Co (ppm)	Sb (ppm)	Bi (ppm)
NHG	4.56	86	3.94	16.1	7.65	99	66	15	2
LG	1.3	21	1.11	14.6	2.32	42	23	7	12
MG	2.36	39	2.13	154	6.52	74	97	19	15

The MG and NHG Composites are both higher grade in copper and gold than the Stardust LOM average grades described in the mine plan. Material from the Stardust deposit is anticipated to have a higher grade than material from the Kwanika deposit.

13.6.2 2020-2021 Base Met Gravity Testing

Gravity concentration by Knelson concentrator and Mozley table was added into later tests as a step before rougher flotation. The tests utilizing gravity concentration targeted a primary grind size P_{80} of 150 µm. Gravity gold recovery was 24% for the MG and NHG composites and 42% for the LG Comp. Gravity concentration was included in the flowsheet because its addition allowed for significant coarsening of the primary grind size without losing out on overall gold recovery.

13.6.3 2020-2021 Base Met Flotation Testing

13.6.3.1 Flotation Testing Primary Grind

Primary grind analysis was conducted on the NHG Comp across the primary grind size P_{80} of 75, 100, and 150 µm. Primary grind size was found to have very little effect on rougher copper recovery, but rougher gold recovery was significantly improved at a P_{80} of 75 µm. The 75 µm test achieved a copper recovery of 93.8% and a gold recovery of 88.8% to the rougher. Based on this testing, the initial cleaner tests were performed at a primary grind size of 75 µm.

13.6.3.2 Flotation Testing Cleaner Regrind

Starting from a primary grind size of 75 µm, cleaner tests investigated flotation performance with no regrind, 45 µm regrind, and 35 µm regrind. Both copper and gold recoveries were greatly improved by finer regrind sizes. The test with 35 µm regrind achieved a final concentrate copper grade of 30.4% and recovered 90.0% of the copper and 85.4% of the gold.

13.6.4 2020-2021 Base Met Combined Gravity and Flotation Testing

The addition of gravity concentration prior to flotation allows for the coarsening of the primary grind. Two comparison tests were performed on material from the NHG composite. This composite is meant to be representative of highest-grade feed from the Stardust deposit. The comparison tests were completed at a relatively coarse grind P_{80} of 150 µm and a regrind of 40 µm with and without initial gravity concentration. The two tests had very similar final concentrate copper recoveries and grades, but the test without gravity recovered only 58.7% of the gold in the final concentrate. The test with gravity concentration achieved a gold recovery of 83.5%. However, the inclusion of gravity concentration is not considered economically viable when considering the much larger Kwanika deposit. Material from Kwanika is lower grade than Stardust and is not amenable to gravity concentration. This Base Met test program does prove that the Stardust material can be processed by a similar flowsheet as the one established in earlier BV Minerals testwork.

13.7 2022 Base Met Assessment of Kwanika/Stardust Samples

Base Metallurgical Laboratories was contracted to evaluate the metallurgical performance on blend composites assembled from Kwanika and Stardust samples. Half drill core samples of Kwanika material were provided and samples from the previous Stardust test program were used to assemble blends. Five composites were assembled from the Kwanika drill core samples and tested individually. Ten Blend composites were assembled which contained one of the Kwanika composites and either LG or MG Stardust sample, combined at a consistent ratio of 90% Kwanika and 10% Stardust. The Kwanika material represents material types from the open pit and the eastern portion of the underground block cave. The Stardust material is blended high-grade mineralization from the deposit.

13.7.1 2018-2019 BV Minerals Program Comminution Testing

Bond ball mill work index (BWi) were conducted on each of the Kwanika composites. The tests were conducted using a 106 µm closing screen and results are summarized in the Table 13-16.

Table 13-16: Base Met BWi Results

Composite ID	Bond Ball Mill Work Index kWh/t
Met 1	19.9
Met 2	21.8
Met 3	22.4
Met 4	18.7
Met 5	18.1

13.7.2 2022 Base Met Head Assay

The head assays of these five composites are summarized in Table 13-17.

Table 13-17: Base Met Composites Head Assay Results

Comp	Au (g/t)	Ag (g/t)	Cu (%)	S (%)
Met 1	0.46	1.3	0.38	0.88
Met 2	0.25	1.2	0.36	1.90
Met 3	0.19	1.1	0.56	2.40
Met 4	2.77	1.6	0.52	1.16
Met 5	0.55	1.7	0.71	1.13

13.7.3 2022 Base Met Mineral Composition

BMA via QEMSCAN were conducted on head samples of each Kwanika composite. The composites contained modest sulphide mineral contents, ranging from 1.4 to 3.1%. Pyrite accounted for 33 to 61% of the sulphide mineral content and 36 to 77% of the total sulphur in the samples. Chalcopyrite was generally the dominant copper sulphide mineral; however, significant levels of secondary copper sulphide minerals were present. Details of the copper deportment by mineral are presented in Table 13-18.

Table 13-18: Base Met Composites – Copper Mineral Distribution

Mineral	Copper Distribution - %				
	Met-1	Met-2	Met-3	Met-4	Met-5
Chalcopyrite	59.9	87.6	89.9	26.8	72.2
Bornite	37.0	12.4	10.1	43.1	23.3
Chalcocite	2.87	0.00	0.00	27.9	1.60
Covellite	0.00	0.00	0.00	2.19	2.88
Other Sulphides	0.24	0.00	0.00	0.00	0.06
Total	**100**	**100**	**100**	**100**	**100**

13.7.4 2020-2021 Base Met Flotation Testing – Kwanika Composites

13.7.4.1 Gravity Plus Rougher Flotation Testing

Kinetic rougher flotation tests were conducted on the composites at a primary grind sizing of 75 µm P_{80}, with gravity concentration preceding flotation. Modest gravity gold recoveries were measured, so this separation process was not continued in the subsequent cleaner tests.

13.7.4.2 Cleaner Flotation Testing

Open circuit cleaner tests were conducted to investigate flotation performance with the developed rougher conditions and a 20 µm regrind target. Two tests were conducted on each composite, which evaluated progressively selective cleaner circuit conditions. Copper recoveries to final concentrates ranged from 82 to 92 percent at concentrate grades of 26 to 35% copper. As the selectivity against pyrite increased, gold recovery to final concentrate decreased in these tests, ranging from 44 to 65%. Results are summarized in Table 13-19.

Table 13-19: Base Met Composites – Cleaner Test Results - Kwanika

Composite	Cleaner Conditions	Mass %	Concentrate Grade				Distribution - Percent			
			Cu (%)	Au (g/t)	Ag (g/t)	S (%)	Cu	Au	Ag	S
Met 1	Baseline	1.5	20.2	19	53	37.2	82.5	63.1	77.1	63.7
	Selective	1.2	26.1	21	61	28.4	82.2	61.8	72.0	39.9
Met 2	Baseline	1.4	23.1	15	67	33.4	90.4	65.3	73.7	22.6
	Selective	1.1	30.1	14	68	32.7	88.5	44.1	57.6	18.1
Met 3	Baseline	1.9	26.7	5	43	31.9	90.4	49.4	74.9	23.5
	Selective	2.0	27.2	4	46	32.8	91.6	60.0	64.3	24.3
Met 4	Baseline	2.1	17.5	95	51	38.6	83.4	78.9	87.1	64.0
	Selective	1.0	35.4	169	85	31.1	78.2	65.3	56.0	27.5
Met 5	Baseline	3.0	20.7	16	44	37.0	91.0	72.0	85.6	77.2
	Selective	2.1	30.3	19	65	31.5	90.7	60.2	62.8	50.9

13.7.5 2022 Base Met Combined Gravity and Flotation Testing – Blend Composites

A series of gravity plus rougher flotation tests were conducted on the Blend composites. Copper recoveries ranged from 91 to 96%, at mass recoveries of 7 to 16 percent. Total gold recoveries ranged from 81 to 90%, and between 8 and 22% was recovered in the gravity circuit.

Open circuit cleaner tests were conducted on the composites, using similar conditions as in the Kwanika composites. Gravity concentration was only applied on Blends 4 and 9 which contained higher gold feed grades from the Met 4 material. Similar metallurgical performance was measured, and copper recoveries ranged from 81 to 92% to final concentrate grades of 26 to 35% copper. Gold recoveries to final concentrates ranged from 52 to 71%, with 14 to 16% of the gold reporting to gravity concentrates where applied.

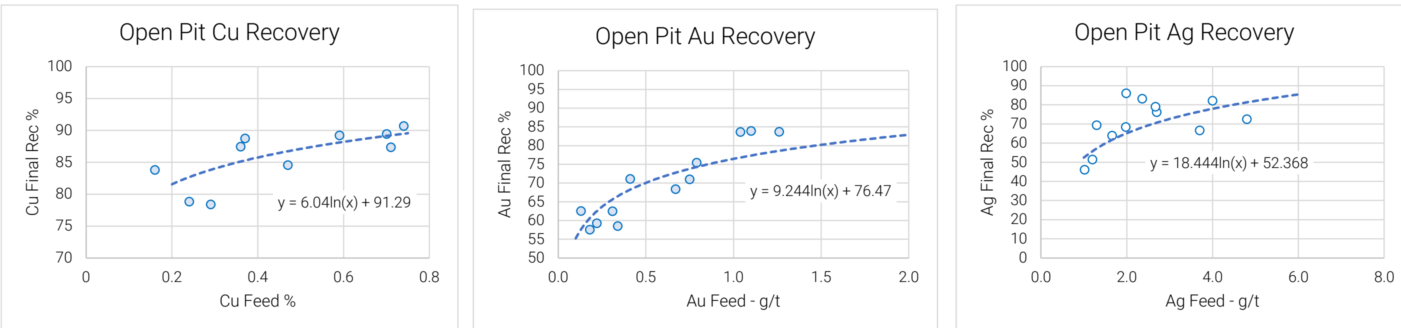
13.8 Recovery Estimate

To create a recovery estimate for the Kwanika deposit, tests from all test programs were considered. The recovery equations are scaled for 100 µm, although most of the testwork was performed at a primary grind size of 75 µm. Results for the samples that were atypical in grade or mineral assemblage were discarded for the purpose of the analysis. Out of the remaining samples, several were of higher grade than the mine plan, including six of the ten open pit samples and two of the seven underground samples. The open circuit cleaner tests from BV Minerals were inconsistent in terms of concentrate quality and cleaner circuit losses and were thus difficult to use for estimating final grade-recovery points. The locked cycle tests provided a clearer assessment of final grade-recovery points, but none of these tests were conducted on open pit material possessing mine plan feed grades. The BV Minerals tests also included the recycle of a rougher scavenger flotation concentrate which was recovered using high PAX doses. This flowsheet could have contributed to cycle instability experienced by some samples. For Stardust, average recoveries were determined from open circuit cleaner tests on the MG and LG samples which graded 1.1-2.2% copper.

It is proposed that metal recovery equations can be reasonably estimated by using measured rougher recoveries obtained on variability open circuit cleaner tests and applying cleaner circuit recoveries measured in locked cycle tests. Estimations of the circuit recoveries at 100 µm from feed grade or performance criteria are shown in Figures 13-9 and 13-10.

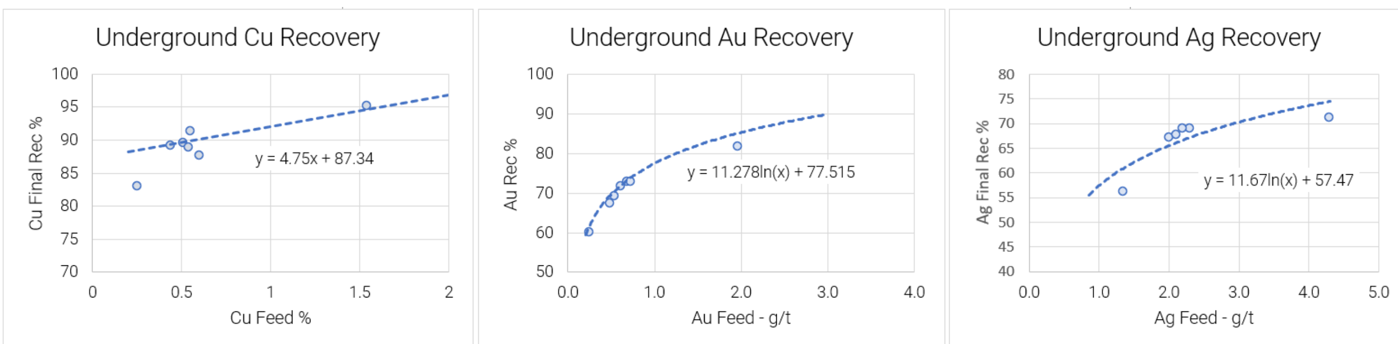
The open pit and underground recovery equations are presented In Table 13-20.

Figure 13-9: Open Pit Recovery Curves



Source: Ausenco, 2022.

Figure 13-10: Underground Recovery Curves



Source: Ausenco, 2022.

Table 13-20: Open Pit and Underground Recovery Equations

Metal	Open Pit	Underground
Copper	Rec = 6.04 * LN(Cu %) + 91.3	Rec = 4.75 * (Cu %) + 87.34
Gold	Rec = 9.24 * LN(Au g/t) + 76.5	Rec = 11.28 * LN(Au g/t) +77.5
Silver	Rec = 18.4 * LN(Ag g/t) + 52.37	Rec = 11.67 * LN(Ag g/t) + 57.5

13.9 Concentrate Quality

Final concentrates from 4 locked cycle tests and 10 open circuit cleaner tests, representing in total 9 unique Kwanika composites, were assayed for minor element contents. No deleterious elements were measured at potential penalty levels. The highest arsenic levels were measured in final concentrates from two underground composites, returning values of 1070 and 1164 ppm As.

13.10 Comments on Mineral Processing and Metallurgical Testing

Both Kwanika and Stardust deposits are amenable to conventional sulphide flotation. The greater portion of the combined resource is contained within the Kwanika deposit, which hosts relatively fine-grained copper minerals that require a relatively fine primary grind size and regrind of rougher flotation concentrate. Cleaner flotation can produce a saleable concentrate with acceptable cleaner circuit recoveries. The levels of impurities in the concentrate are below the smelting penalty thresholds applied by most smelters according to the metallurgical testwork performed to date. Future testwork should focus on completing more tests at target mine plan feed grades and carrying the testwork through to stable locked cycle tests. Additional testing should be completed to investigate coarser primary grind and regrind discharge sizes.

14 MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimates presented herein represent the copper, gold, and silver mineral resource evaluation prepared for the Kwanika and Stardust projects. The effective date of these Mineral Resource Estimates is Jan 4, 2023.

The current updated Kwanika Central and Kwanika South block models were completed by RockRidge Consulting Resource Geologists of Vancouver, British Columbia The estimates were further reviewed and signed-off by Mr. Brian Hartman, P. Geo. of Ridge Geoscience LLC and subcontractor to Mining Plus, a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. The current Mineral Resource estimate has been updated using additional drilling in 2020-2021 on both the Kwanika Central and South Zones. The estimate incorporates geological and structural constrains developed through lithological and structural modelling and familiarity with the deposit. Mining Plus completed a site visit to the project on September 20, 2022.

The Kwanika Central Mineral Resource is reported using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus G&A costs. The underground mineral resources are reported using an economic cut-off of US$ 16.41, which covers the additional underground mining and G&A costs of US$8.20/tonne. Additionally, the Mineral Resource is constrained by an open pit mining shell and underground block caving shape to satisfy reasonable prospects for eventual economic extraction. These constraining shapes were generated by Mining Plus. The Kwanika Central Zone Mineral Resource is shown in Table 14-1. Mineral Resources that are not Mineral Reserves do no have demonstrated economic viability.

Table 14-1: Mineral Resource Statement, Kwanika Central Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit	8.21	Measured	30.7	0.31	0.31	1.05	210.8	310.5	1,041.7
		Indicated	35.9	0.22	0.19	0.80	174.9	222.0	923.9
		M&I	66.6	0.26	0.25	0.92	385.7	532.5	1,965.6
		Inferred	4.1	0.15	0.15	0.58	13.8	20.1	77.3
	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Underground	16.41	Measured	25.6	0.50	0.61	1.62	284.4	501.3	1,332.6
		Indicated	11.3	0.51	0.65	1.56	126.2	236.7	565.1
		M&I	36.8	0.51	0.62	1.60	410.6	738.0	1,897.8
		Inferred	-	-	-	-	-	-	-

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
3. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
4. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.

5. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Underground Mineral Resources are reported at an economic cut-off of US$16.41 and constrained by a conceptual block cave shape. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne, and underground mining and G&A costs are assumed to be US$8.20/tonne.
6. Actual SG measurements were interpolated into the block model, with an average SG of 2.74.
7. The quantity and grade of reported Inferred Mineral Resources in this report are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
8. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The Kwanika South Mineral Resource is reported using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus G&A costs. Additionally, the Mineral Resource is constrained by an open pit mining shell to satisfy reasonable prospects for eventual economic extraction. There are no underground mineral resources at Kwanika South. Mineral Resources that are not Mineral Reserves do no have demonstrated economic viability. Table 14-2 shows the Kwanika South Zone Mineral Resource.

Table 14-2: Mineral Resource Statement, Kwanika South Zone

Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
						Cu (Mlbs)	Au (koz)	Ag (koz)
8.21	Inferred	25.4	0.28	0.06	1.68	155.0	52.4	1,373.9

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
3. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
4. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
5. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 62% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne.
6. Actual SG measurements were interpolated into the block model, with an average SG of 2.68.
7. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
8. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

The Stardust Mineral Resource Estimate was completed by Ronald G Simpson, P. Geo. of Geosim Services Inc. and previously reported in "Stardust Project – Updated Mineral Resource Estimate NI 43-101 Technical Report" dated May 17, 2021. No changes have been made to the Stardust mineral resource estimate. Mr. Simpson is the Qualified Person for information that relates to the Stardust Mineral Resource.

The Stardust underground mineral resource estimate is based on a cut-off of US$88/tonne and 2 m minimum mining width and is shown in Table 14-3. Mineral Resources that are not Mineral Reserves do no have demonstrated economic viability.

Table 14-3: Mineral Resource Statement – Stardust CCS Zone

Underground	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
	88.00	Indicated	1.6	1.49	1.63	30.1	52.2	83.1	1,536.4
		Inferred	4.1	1.00	1.38	22.8	90.0	181.1	3,004.3

Notes:

14. The Mineral Resources have been compiled by Mr. B Ronald G. Simpson of GeoSim Services Inc. Mr. Simpson has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
15. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
16. Mineral Resources are not mineral reserves and do not have demonstrated economic viability.
17. Reasonable prospects for economic extraction were determined by applying a minimum mining width of 2.0 m. and excluding isolated blocks and clusters of blocks that would likely not be mineable.
18. The base case cut-off of US$88/t was determined based on metal prices of $1,650/oz gold. $21.50/oz silver and $3.50/lb copper, underground mining cost of US$64/t, transportation cost of US$6/t, processing cost of US$8.25/t, and G&A cost of US$9.75/t. Recovery formulas were based on recent metallurgical test results. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag.
19. Block tonnes were estimated using a density of 3.4 g/cm^3 for mineralized material.
20. Six separate mineral domains models were used to constrain the estimate. Minimum width used for the wireframe models was 1.5 m.
21. For grade estimation, 2.0-metre composites were created within the zone boundaries using the best-fit method.
22. Capping values on composites were used to limit the impact of outliers. For Zone 102, gold was capped at 15 g/t, silver at 140 g/t and copper at 7.5%. For all other zones, gold was capped at 6 g/t, silver at 140 g/t and copper at 5%.
23. Grades were estimated using the inverse distance cubed method. Dynamic anisotropy was applied using trend surfaces from the vein models. A minimum of 3 and maximum of 12 composites were required for block grade estimation.
24. Blocks were classified based on drill spacing. Blocks falling within a drill spacing of 30 m within Zones 2, 3, and 6 were initially assigned to the Indicated category. All other estimated blocks within a maximum search distance of 100 m were assigned to the Inferred category. Blocks were reclassified to eliminate isolated Indicated resources within inferred resources.
25. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
26. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

14.1 Kwanika Central Zone

14.1.1 Resource Database

The database used for the Kwanika Central resource estimate comprises collar, survey, assay, lithology, alteration, density, and structural information for exploration drilling conducted between 2006 and September 2021. Drilling on the Central Zone totalled 76,156 m in 166 holes. A total 29,431 core samples were submitted to the lab for analysis.

Drill spacing in generally just under 50 m in the densely drilled portions of the project.

14.1.2 Geological Models

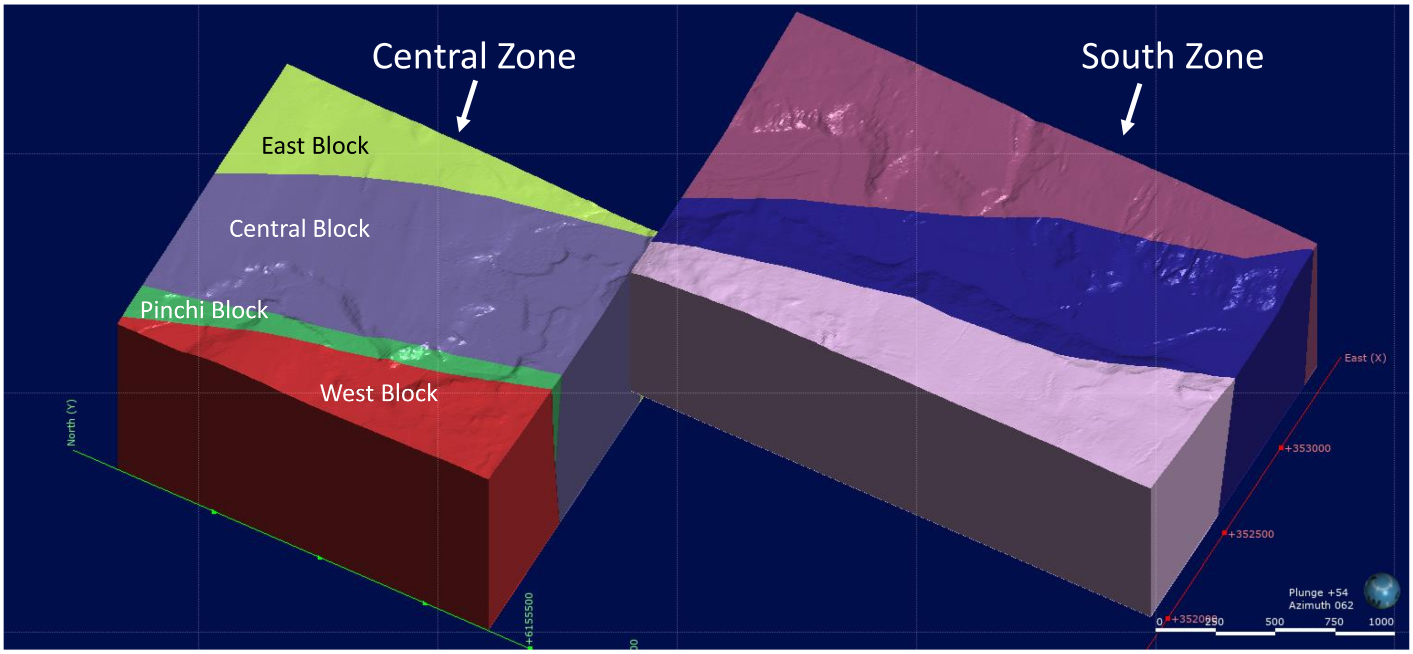
14.1.2.1 Topographic Surfaces

A bare earth Lidar survey was flown over the property in September of 2016. A surface was created from the Lidar data points for both the Central and South zones. These surfaces were triangulated using all points for each zone. The collar data was reconciled onto these high-resolution surfaces.

14.1.2.2 Lithology and Fault Solids Modelling

Structural domains were modelled using surface maps, geology sections and logged lithologies in drillholes. The structural model was inspected to identify the magnitude of displacements, and only faults with significant displacements were selected to be the bounding planes for fault blocks. This resulted in the creation of four blocks for the Central Zone as shown in Figure 14-1.

Figure 14-1: 3D Perspective View Showing Fault Blocks and Relative Positions of Central and South Zones, Looking Northeast



Source: Mining Plus, 2022

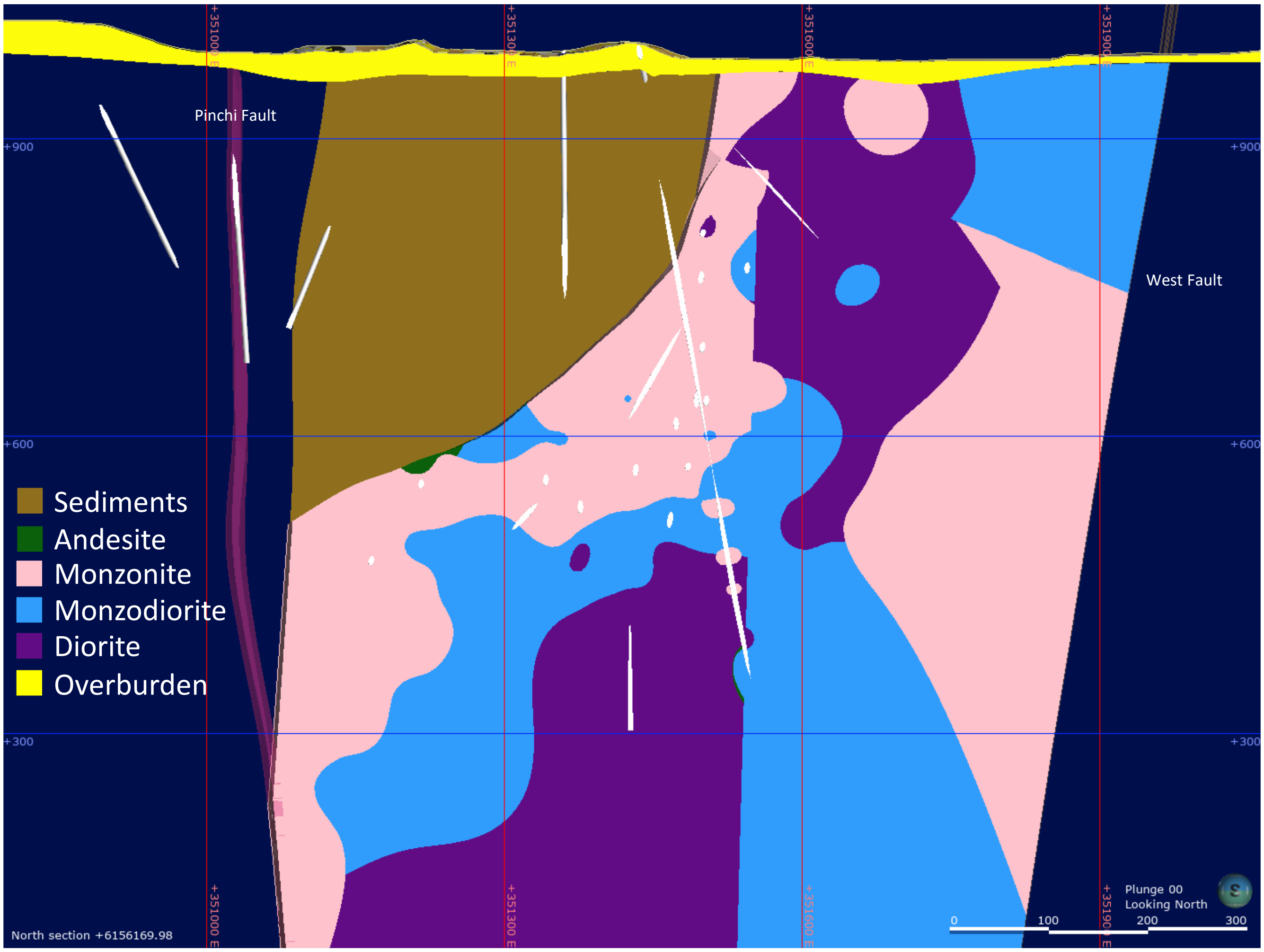
An updated geologic interpretation was completed by NorthWest Copper and reviewed for use in the resource model. The geologic interpretation was then used to create mineralization shapes for copper and separately for Au-Ag mineralization.

Based on the updated drilling and on refinement of previous interpretations, the solids and surfaces created for the Central Zone modelling include:

- The regional Pinchi Fault bounding mineralization to the west
- The Central Fault which controls mineralization orientation within the deposit
- Lithologies including diorite, monzonite, monzodiorite, the Takla Group andesites, the sedimentary basin (with sub-groups within the sediments), the Cache Creek Group, and barren dykes
- Alteration zones
- The bottom of the Overburden

Figure 14-2 shows the main lithology units in sectional view, looking north.

Figure 14-2: Representative 2D Section Showing Main Lithology Units Within the Central Fault Block, Looking North



Source: Mining Plus, 2022

The lithologic model aided in the creation of the domains used in resource modelling. The dykes, however, were not modelled explicitly because it was determined that the true thickness of the post-mineral dykes is less than the minimum thickness separable during mining of around 2 m. Therefore, the dykes are included within the modelled mineralization domains as internal dilution. Furthermore, the oxides have not been explicitly modelled because it has been determined that they are not volumetrically significant and will have minimal effect on the overall metallurgical recoveries.

14.1.2.3 Estimation Domains

Six domains were used in the resource estimate and were created using the fault blocks and lithologies. All domains were modelled inside the central fault block, which is defined by the Pinchi Fault System to the west (used as the footwall boundary) and the West Fault (forming the hangingwall or eastern boundary).

<u>Oxide</u>

Interval selection is based on the presence of oxide copper minerals. The domain was clipped to the surrounding HG and Chalcocite domains, the overburden, and the unconformity.

Chalcocite

Interval selection was based on the presence of Chalcocite. The domain was clipped to the overburden and unconformity.

Outer LG

The Outer LG zone was modelled inside the central fault block boundary and defines the volume containing mostly monzonite and monzodiorite lithologies. Both a 0.1% Cu grade shell and a 0.1 g/t Au grade shell were created. The merged volume of these two shells defines the domain.

Outer MG

Consists of the modelled monzonite lithology and merged with a ≥5% vein indicator. The domain volume was then clipped to a distance buffer of 100 m from drillholes and intersection with the Low-Grade domain.

Central HG-North

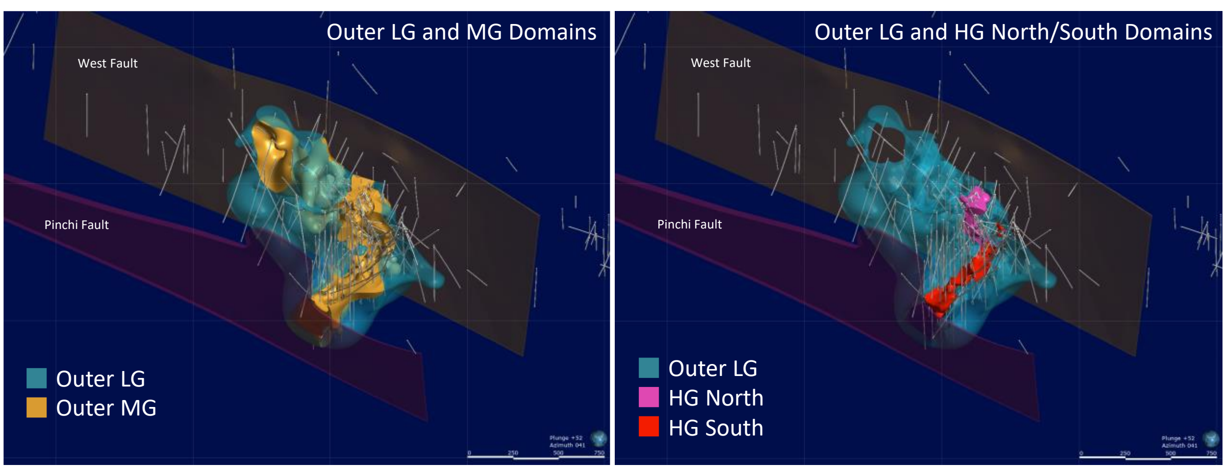
A single HG domain was initially created using the Outer MG as a boundary and creating a 0.8% Cu equivalent grade shell. A clear break in mineralization was observed through the initial volume. Mineralization trends displayed different orientations on opposite sides of this break. A plane was created along this feature and the domain was split into a northern and southern portion.

Central HG-South

This domain is the southern portion of the HG domain created with a 0.8% CuEq grade shell.

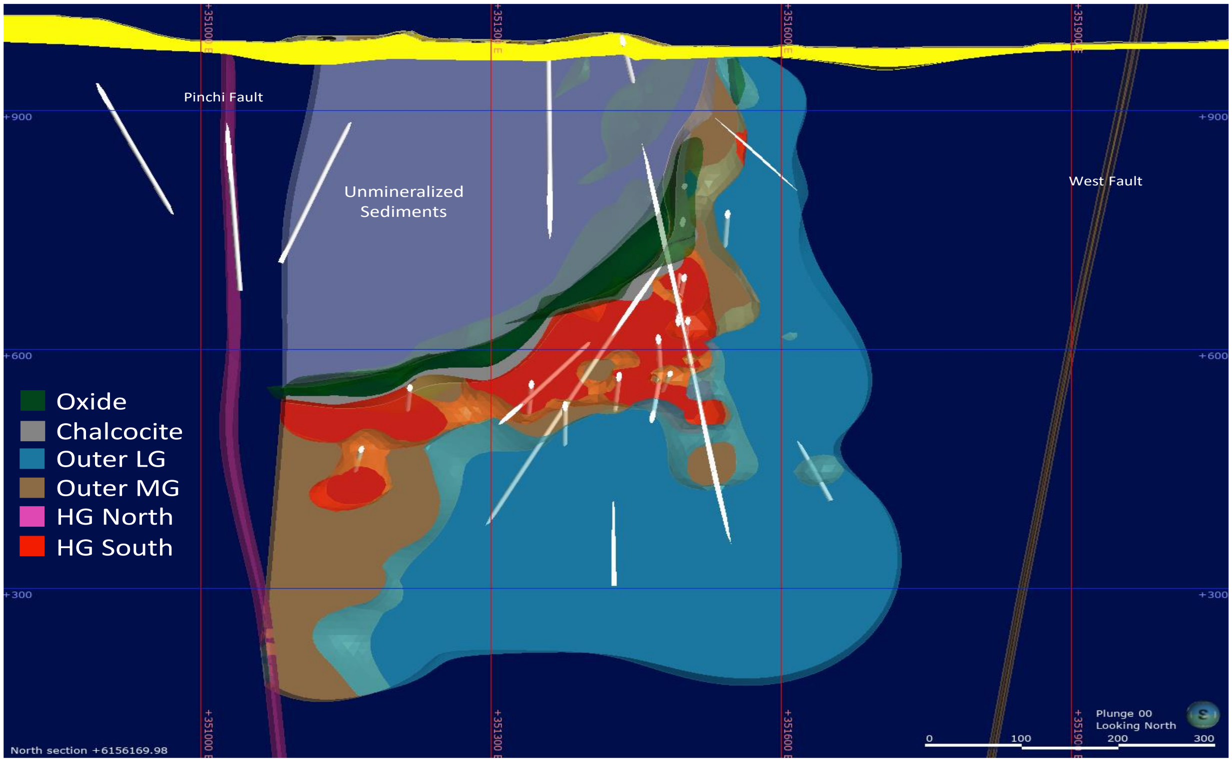
Figure 14-3 and Figure 14-4 show a 3D perspective view 2D section view of the estimation domains, respectively.

Figure 14-3: 3D Perspective View Showing Estimation Domains Within the Central Fault Block, Looking Northeast



Source: Mining Plus, 2022

Figure 14-4: Representative 2D Section Showing Estimation Domains Within the Central Fault Block, Looking North



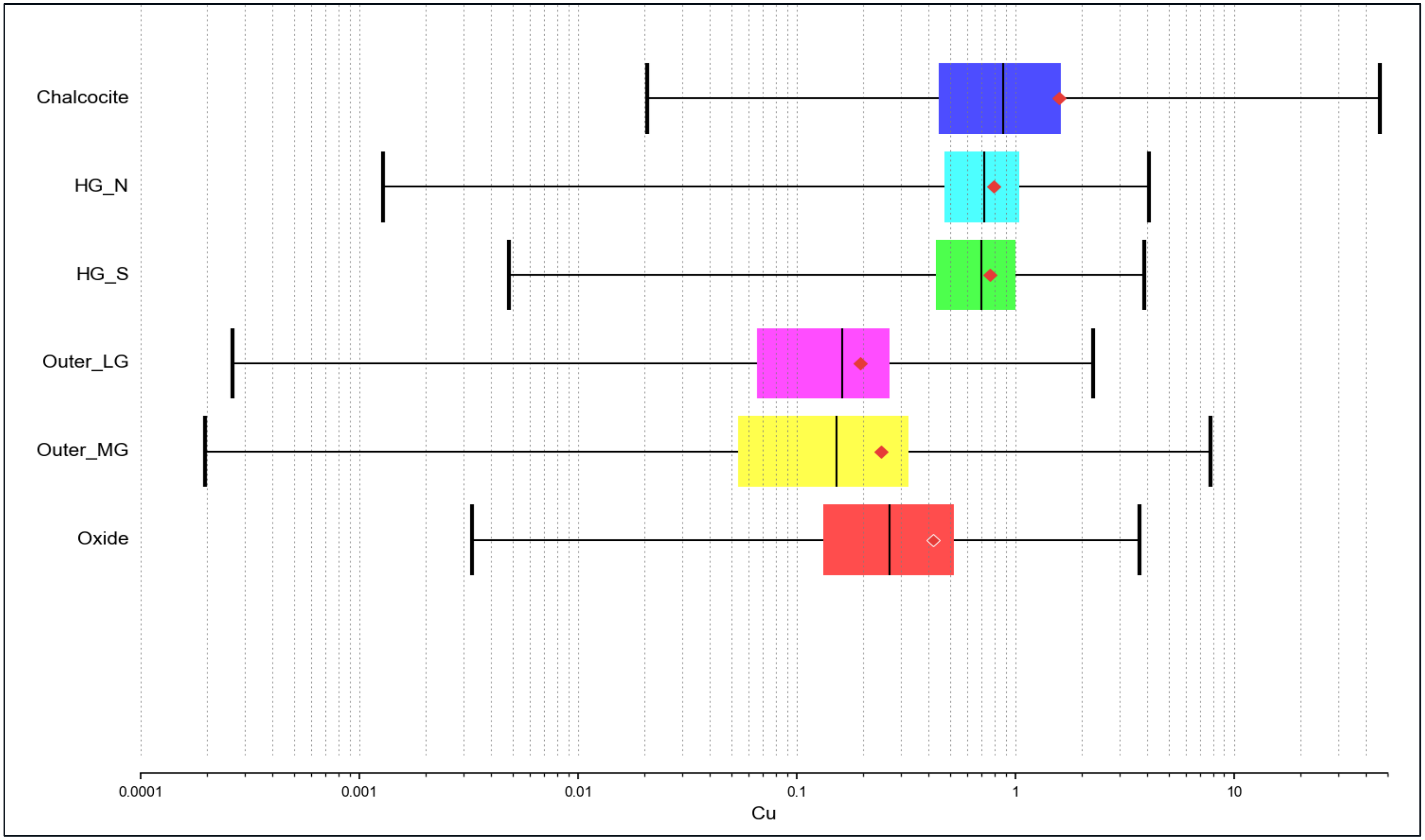
Source: Mining Plus, 2022

14.1.3 Assays, Composites, and Capping

The assay data were examined to determine a suitable composite interval. The chosen interval should standardize the assay intervals to give an equal weight to each record, but still reflect the variability in the original data as far as possible. The Kwanika drill core was predominantly sampled at an interval of 2 m or less. Assay records were assigned a domain code and then composited to approximate 2 m intervals. The composite interval was varied around an average of the selected 2-m interval while keeping as close as possible to a full 2 m. This was done where required to avoid excessively short interval composites from forming at domain boundaries or at the ends of holes. All grade distributions are positively skewed and exhibit quite low standard deviation to mean ratios (Coefficient of Variation) in the high-grade domains and only moderately high ratios in the lower grade domains. A copper assay boxplot is shown below in Figure 14-5.

The presence of high-grade outlier values was investigated as these anomalous values could adversely influence the estimate by contributing excessively to the total metal content of the deposit. In all estimation domains and for all elements, the location of the high-grade outliers was not concentrated in an area, but rather scattered throughout each domain. Appropriate capping limits were selected by studying coefficient of variation plots, probability plots and decile analyses plots. Values above the capping limits were reduced to the capping limit, except for copper and silver in the chalcocite zone, where a reduced search ellipse was used for values over the high-grade threshold. Statistics summarizing composites and capping for each domain is provided in Table 14-4.

Figure 14-5: Boxplot Showing Cu% by Central Zone Estimation Domains



Source: Mining Plus, 2022

Table 14-4: Capping and Composite Statistics by Central Zone Estimation Domain

Copper %	Central HG N	Central HG S	Chalcocite	Oxide	Outer MG	Outer LG
Total Composites	1,313	1,194	465	512	9,193	9,193
Min Before Capping	0.00	0.00	0.02	0.00	0.00	0.00
Max Before Capping	4.06	3.87	46.61	3.70	7.81	2.25
Mean Before	0.79	0.76	1.58	0.42	0.24	0.20
Std Dev Before	0.49	0.46	3.84	0.47	0.31	0.18
CV Before	0.62	0.61	2.43	1.12	1.30	0.92
Capping Value	2.95	NONE	11.68*	2.24	4.05	NONE
No of Capped Comps	4	0	7	6	5	0
Mean After	0.79	0.76	1.34	0.41	0.23	0.20
Std Dev After	0.48	0.46	1.72	0.44	0.29	0.18
CV After	0.60	0.61	1.28	1.06	1.24	0.92
Capped %	0.3%	0.0%	1.5%	1.2%	0.1%	0.0%
Metal % Capped	0.3%	0.0%	15.1%	1.5%	0.4%	0.0%
Gold (g/t)	**Central HG N**	**Central HG S**	**Chalcocite**	**Oxide**	**Outer MG**	**Outer LG**
Total Composites	1,313	1,194	465	512	9,193	9,193
Min Before Capping	0.01	0.02	0.01	0.01	0.00	0.00
Max Before Capping	6.52	6.62	5.54	3.20	6.10	6.05
Mean Before	1.10	1.19	0.75	0.31	0.27	0.15
Std Dev Before	0.95	1.01	0.94	0.45	0.38	0.18
CV Before	0.87	0.85	1.24	1.44	1.43	1.24
Capping Value	NONE	NONE	NONE	2.15	2.60	2.04
No of Capped Comps	0	0	0	6	37	4
Mean After	1.10	1.19	0.75	0.30	0.27	0.14
Std Dev After	0.95	1.01	0.94	0.42	0.36	0.17
CV After	0.87	0.85	1.24	1.38	1.34	1.16
Capped %	0.0%	0.0%	0.0%	1.2%	0.4%	0.0%
Metal % Capped	0.0%	0.0%	0.0%	1.7%	1.2%	0.4%
Silver (g/t)	**Central HG N**	**Central HG S**	**Chalcocite**	**Oxide**	**Outer MG**	**Outer LG**
Total Composites	1,313	1,194	465	512	9,193	9,193
Min Before Capping	0.1	0.1	0.1	0.1	0.1	0.1
Max Before Capping	13.7	10.7	114.3	12.8	44.9	66.9
Mean Before	2.6	2.4	3.7	1.5	0.8	0.8
Std Dev Before	1.6	1.4	8.8	1.6	1.0	1.3
CV Before	0.6	0.6	2.3	1.1	1.2	1.7
Capping Value	NONE	NONE	32.4*	6.1	8.6	19.0
No of Capped Comps	0	0	5	7	8	6
Mean After	2.6	2.4	3.3	1.4	0.8	0.7
Std Dev After	1.6	1.4	4.5	1.4	0.9	0.9
CV After	0.6	0.6	1.4	1.0	1.1	1.2
Capped %	0.0%	0.0%	1.1%	1.4%	0.1%	0.1%
Metal % Capped	0.0%	0.0%	12.8%	3.5%	0.8%	1.9%

* Indicates that a reduced search ellipse was used rather than a simple cap

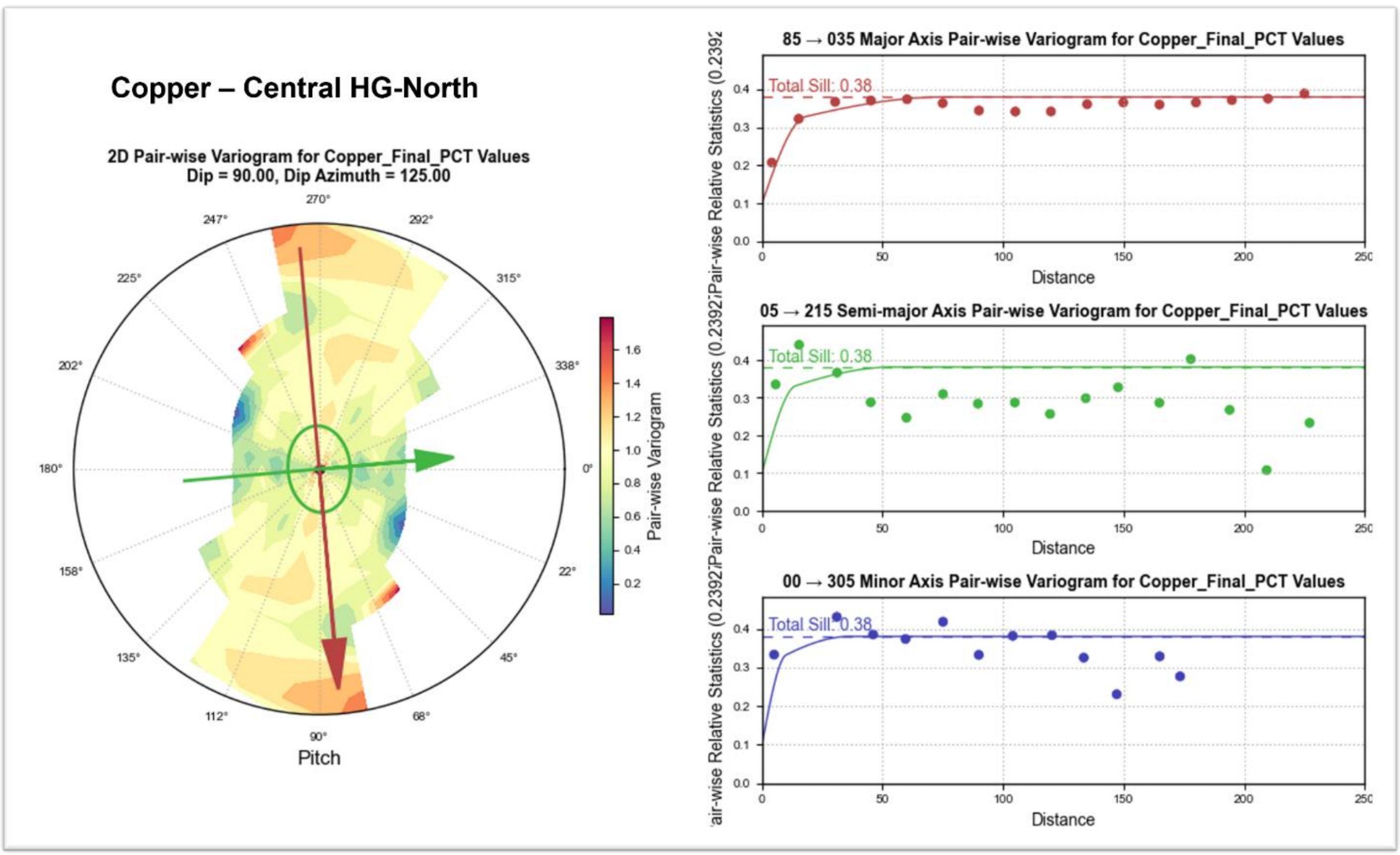
14.1.4 Variography

Experimental pairwise relative semi-variograms were calculated and modelled for each metal in each mineralized domain. Spherical two structure models were fitted to experimental semi-variograms in all cases. An example of experimental semi-variograms with fitted model for Cu is shown in Figure 14-6.

All domains had sufficient samples to create adequate experimental semi-variograms for each metal. Strong anisotropy was observed for the most part and directional variogram models were used. The nugget values (i.e., the sample variability at close distance) were established from downhole variograms.

Nugget values vary from 20-50% of the total sill value for all elements in all domains. Major axes range for the short first structures of all the variograms were around 20-50 m and the ranges for the second structure were 70-110 m on average. All variogram model parameters are listed per element in Table 14-5.

Figure 14-6: Model Variogram for Copper in the Central Zone HG-North Estimation Domain



Source: Mining Plus, 2022

Table 14-5: Central Zone Variogram Parameters by Metal and Estimation Domain

Metal	Estimation Domain	Direction						Structure 1 (Spherical)					Structure 2 (Spherical)				
		Dip	Dip Azi	Pitch	Variance	Nugget	Norm Nugget	Sill	Norm sill	Major	Semi	Minor	Sill	Norm sill	Major	Semi	Minor
Copper	Central HG-North	90	125	85	0.24	0.02	0.10	0.05	0.20	18	14	10	0.02	0.08	75	54	36
	Central HG-South	80	345	40	0.22	0.03	0.15	0.03	0.15	26	24	16	0.02	0.09	82	64	38
	Chalcocite	70	280	135	14.73	4.46	0.30	1.91	0.13	27	15	10	1.86	0.13	68	45	25
	Outer LG	40	260	35	0.03	0.01	0.17	0.01	0.16	48	22	20	0.01	0.24	108	92	76
	Outer MG	0	10	90	0.09	0.02	0.26	0.02	0.23	32	24	22	0.02	0.20	110	84	68
	Oxide	40	270	70	0.22	0.04	0.19	0.02	0.10	40	18	8	0.06	0.28	86	72	20
Gold	Central HG-North	90	125	85	0.91	0.09	0.10	0.19	0.21	18	14	10	0.17	0.19	75	54	36
	Central HG-South	80	345	40	1.02	0.17	0.16	0.19	0.18	26	24	16	0.12	0.12	82	64	38
	Chalcocite	70	280	135	0.88	0.17	0.19	0.10	0.11	27	15	10	0.46	0.53	68	45	25
	Outer LG	40	260	35	0.03	0.01	0.17	0.01	0.17	48	22	20	0.01	0.21	108	92	76
	Outer MG	0	10	90	0.15	0.04	0.26	0.03	0.20	32	24	22	0.03	0.20	110	84	68
	Oxide	40	270	70	0.20	0.03	0.14	0.03	0.14	40	18	8	0.07	0.37	86	72	20
Silver	Central HG-North	90	125	85	2.61	0.39	0.15	0.34	0.13	18	14	9	0.26	0.10	75	54	36
	Central HG-South	80	345	40	1.85	0.23	0.12	0.25	0.13	26	24	16	0.15	0.08	82	64	38
	Chalcocite	70	280	135	77.50	19.59	0.25	13.95	0.18	27	15	10	7.46	0.10	68	45	25
	Outer LG	40	260	35	1.69	0.24	0.14	0.27	0.16	48	22	20	0.25	0.15	108	92	76
	Outer MG	0	10	90	1.06	0.23	0.22	0.18	0.17	32	24	22	0.15	0.14	110	84	68
	Oxide	40	270	70	2.56	0.56	0.22	0.20	0.08	40	18	8	0.56	0.22	86	72	20

14.1.5 Block Model

The block model was constructed to fill the domain volumes with 10 m x 10 m x 10 m blocks in the X, Y and Z directions to best represent the data density, deposit shapes, and to minimize blocks unsupported by data.

Accurate representation of the domain volume was achieved by allowing sub-blocks to be created at domain boundaries. Each parent cell could be split into a minimum sub-block size of 2.5 x 2.5 x 0.5 m. Each sub-block was assigned the estimate derived for the parent block.

The block model size and extents are shown in Table 14-6.

Table 14-6: Central Zone Block Model Dimensions

Direction	Minimum	Maximum	Block Size (m)	# Blocks
Easting	350600	352100	10	150
Northing	6155700	6157460	10	176
Elevation	-40	1100	10	114

14.1.6 Interpolation Methods

Anisotropic search orientations along mineralization trends were used to select data informing block estimates. The Chalcocite and Oxide domains were estimated with variable search orientations to better follow the domain shapes. Search radii were based on the variogram ranges. Ordinary kriging was used to estimate all grades into the block model in three estimation passes whereby each successive pass utilized a less restrictive sample search strategy to estimate any remaining un-estimated blocks. Search radii for the first estimation pass equals half of the variogram range. The second pass increases the search to the range and the third pass further expands to twice the variogram range.

The orientation of the sample search ellipse is aligned with the variogram models for copper and the search ranges for the successive passes are factors of the modelled variogram range. The search ellipse orientations in all cases display the strongest trend NNW-SSE with a steep dip towards the west and a northward plunge. All domains were estimated using ordinary kriging. Search orientations and sample selection criteria for each domain is shown below in Table 14-7.

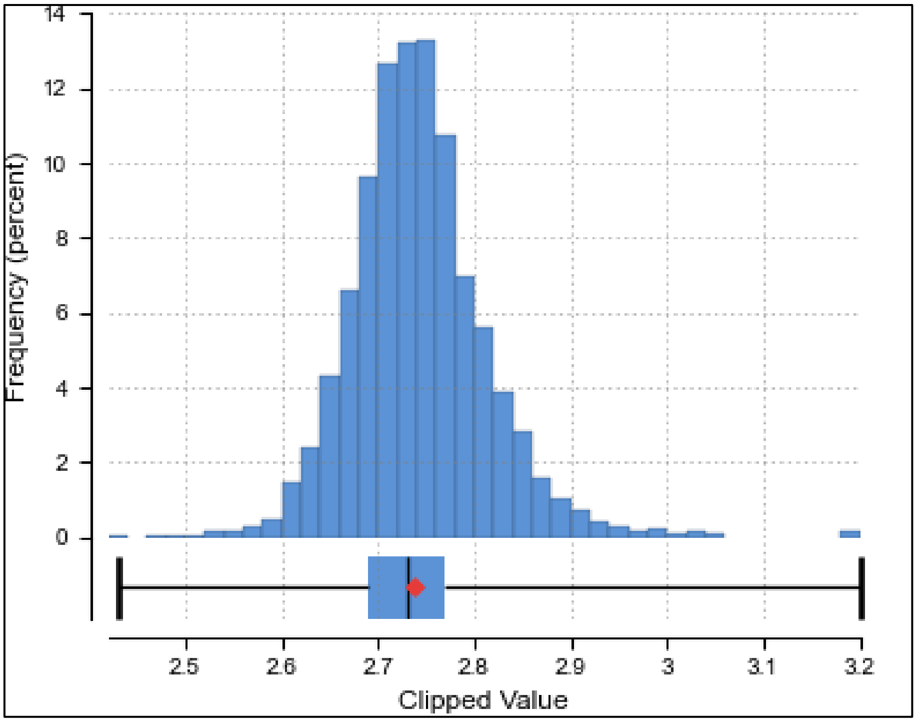
Table 14-7: Kwanika Central Zone Estimation Parameters

Domain	Metal	Ellipsoid Ranges (m)			Ellipsoid Directions			Number of Samples			Capped Value	Outlier Restriction	
		Max	Int	Min	Dip	Dip Azi	Pitch	Min	Max	Max/Hole		Threshold	Distance (%)
Central HG-North	Cu (%)	37.5	27	18	90	125	85	8	20	5	2.95	-	-
		75	54	36	90	125	85	6	20	4	2.95	-	-
		150	108	72	90	125	85	4	20	3	2.95	-	-
	Au (g/t)	37.5	27	18	90	125	85	8	20	5	-	-	-
		75	54	36	90	125	85	6	20	4	-	-	-
		150	180	72	90	125	85	4	20	3	-	-	-
	Ag (g/t)	37.5	27	18	90	125	85	8	20	5	-	-	-
		75	54	36	90	125	85	6	20	4	-	-	-
		150	108	72	90	125	85	4	20	3	-	-	-
Central HG-South	Cu (%)	41	32	19	80	345	40	8	20	5	-	-	-
		82	64	38	80	345	40	6	20	4	-	-	-
		164	128	76	80	345	40	4	20	3	-	-	-
	Au (g/t)	41	32	19	80	345	40	8	20	5	-	-	-
		82	64	38	80	345	40	6	20	4	-	-	-
		164	128	76	80	345	40	4	20	3	-	-	-
	Ag (g/t)	41	32	19	80	345	40	8	20	5	-	-	-
		82	64	38	80	345	40	6	20	4	-	-	-
		164	128	76	80	345	40	4	20	3	-	-	-
Chalcocite	Cu (%)	34	22.5	12.5	Variable Orientation			8	20	5	-	11.68	50
		68	45	25				6	20	4	-	11.68	25
		136	90	50				4	20	3	-	11.68	12.5
	Au (g/t)	34	22.5	12.5	Variable Orientation			8	20	5	-	-	-
		68	45	25				6	20	4	-	-	-
		136	90	50				4	20	3	-	-	-
	Ag (g/t)	34	22.5	12.5	Variable Orientation			8	20	5	-	32.36	50
		68	45	25				6	20	4	-	32.36	25
		136	90	50				4	20	3	-	32.36	12.5
Outer LG	Cu (%)	54	46	38	40	260	35	8	20	5	-	-	-
		108	92	76	40	260	35	6	20	4	-	-	-
		216	184	152	40	260	35	4	20	3	-	-	-
	Au (g/t)	54	46	38	40	260	35	8	20	5	2.04	-	-
		108	92	76	40	260	35	6	20	4	2.04	-	-
		216	184	152	40	260	35	4	20	3	2.04	-	-
	Ag (g/t)	54	46	38	40	260	35	8	20	5	19.00	-	-
		108	92	76	40	260	35	6	20	4	19.00	-	-
		216	184	152	40	260	35	4	20	3	19.00	-	-
Outer MG	Cu (%)	55	42	34	0	10	90	8	20	5	4.05	-	-
		110	84	68	0	10	90	6	20	4	4.05	-	-
		220	168	136	0	10	90	4	20	3	4.05	-	-
	Au (g/t)	55	42	34	0	10	90	8	20	5	2.60	-	-
		110	84	68	0	10	90	6	20	4	2.60	-	-
		220	168	136	0	10	90	4	20	3	2.60	-	-
	Ag (g/t)	55	42	34	0	10	90	8	20	5	8.64	-	-
		110	84	68	0	10	90	6	20	4	8.64	-	-
		220	168	136	0	10	90	4	20	3	8.64	-	-
Oxide	Cu (%)	43	72	10	Variable Orientation			8	20	5	2.24	-	-
		86	72	20				6	20	4	2.24	-	-
		172	144	40				4	20	3	2.24	-	-
	Au (g/t)	43	36	10	Variable Orientation			8	20	5	2.15	-	-
		86	72	20				6	20	4	2.15	-	-
		172	144	10				4	20	3	2.15	-	-
	Ag (g/t)	43	36	10	Variable Orientation			8	20	5	6.06	-	-
		86	72	20				6	20	4	6.06	-	-
		172	144	40				4	20	3	6.06	-	-

14.1.7 Density Assignment

A total of 4,151 SG measurements were provided for the Central Zone. Values range from 2.43 to 4.91, with an average value of 2.74. The values were used to estimate density into the block model using Simple Kriging with a mean value of 2.74. A histogram of SG values for the Central Zone is shown in Figure 14-7.

Figure 14-7: Histogram of SG Values for the Central Zone



Source: Mining Plus, 2022

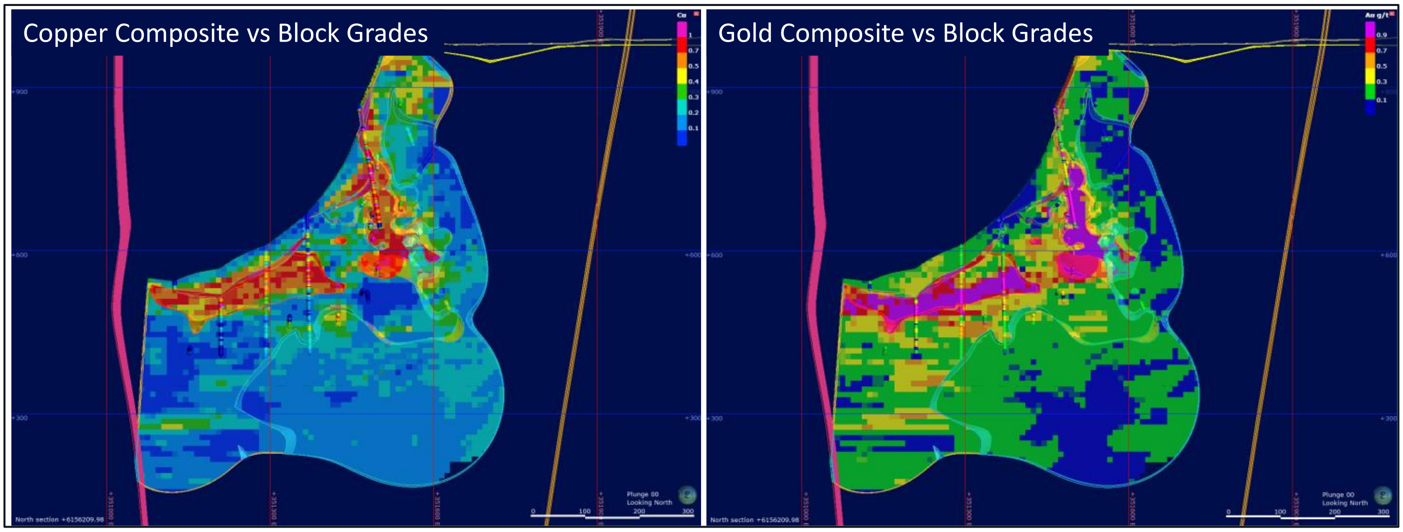
14.1.8 Block Model Validation

The Kwanika Central block model was validated using the following methods:

- visual comparison of colour-coded block grades to drillhole composite grades in sectional view
- spot check on individual blocks to confirm that composite selection and kriging weights were applied according to the search strategy
- global comparison of a Nearest Neighbour (NN) and Inverse Distance model with the OK model
- swath plot analysis comparing NN and OK grades for easting, northing, and elevation.

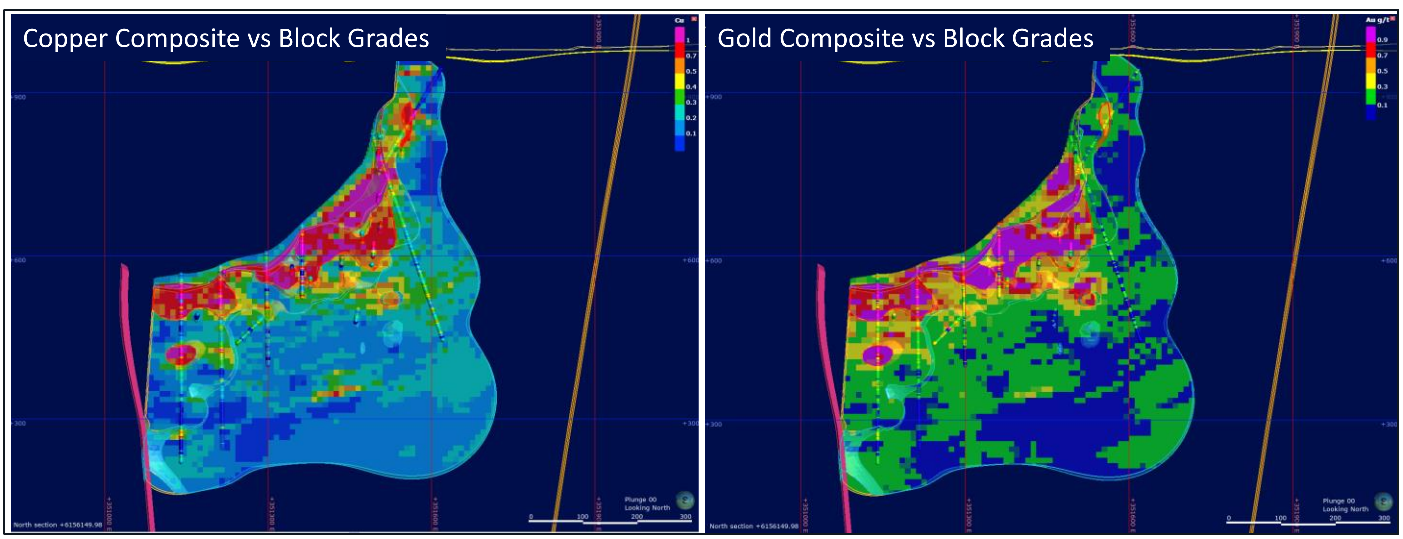
The visual comparison of block model grades with composite grades for copper and gold show a good correlation between values and no large discrepancies are apparent. Figure 14-8 and Figure 14-9 show colour-coded block model copper and gold grades with the drillhole composite grades in two representative sections.

Figure 14-8: Visual Comparison of Composite and Block Copper and Gold Grades Along Northing 6156210, Looking North



Source: Mining Plus, 2022

Figure 14-9: Visual Comparison of Composite and Block Copper and Gold Grades Along Northing 6156150, Looking North



Source: Mining Plus, 2022

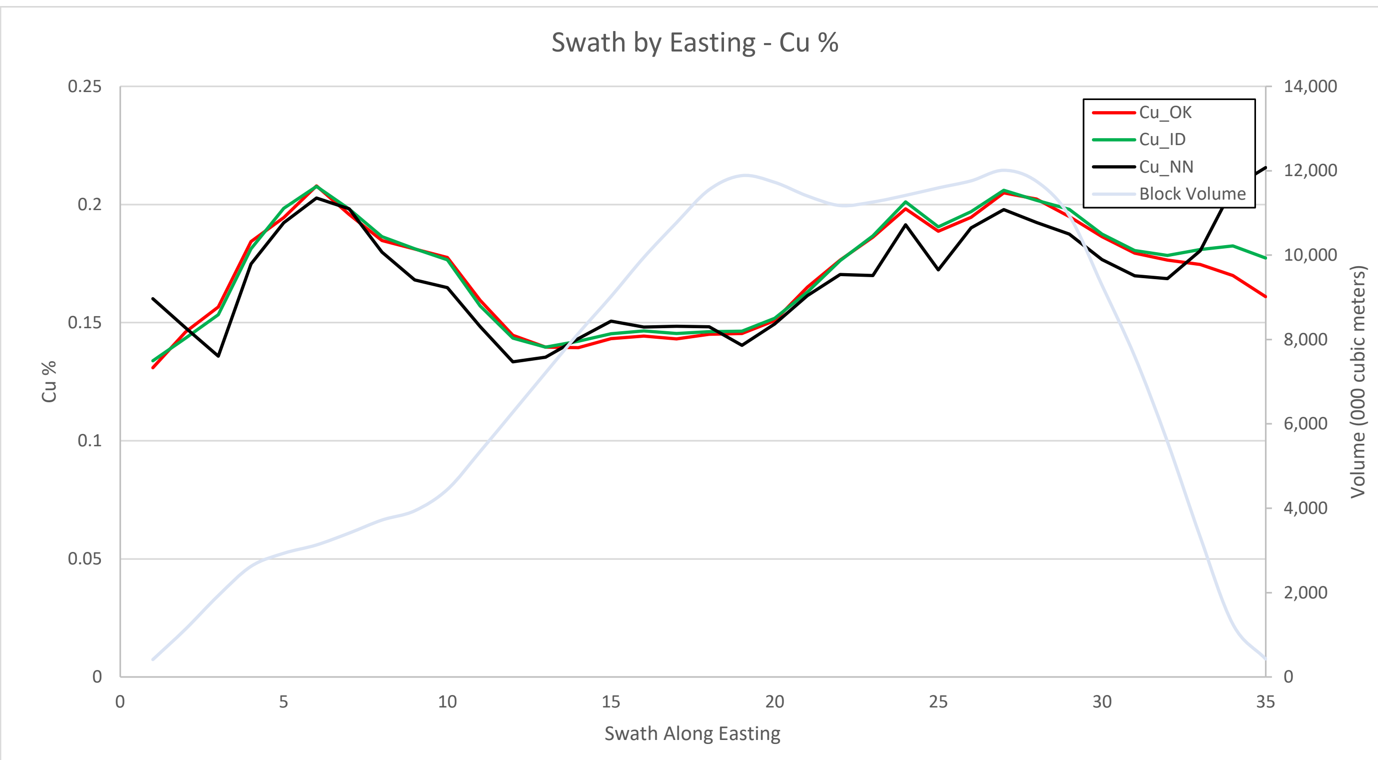
A NN model was completed for copper and gold to serve as a check against the resource model. The NN interpolation method simply assigns a block the same grade as its closest composite. These models are intended to represent a theoretical unbiased estimate of the average grade when no cut-off grade is imposed and is a good basis for checking performance of different estimation methods. The NN model utilized the same search criteria as the OK model except uses a single composite to estimate a block. A comparison of NN, ID, and OK grades was made for all blocks within the six estimation domains, classified as Measured, Indicated, or Inferred, and at a zero percent copper cut-off and is summarized below in Table 14-8. Copper and gold grades compare well and are well within acceptable tolerances.

Table 14-8: Comparison of NN, IDW, and OK Model Grades at a 0% Copper Cut-off

Method	Cu%	Au g/t
NN	0.170	0.169
ID	0.173	0.176
OK	0.172	0.175

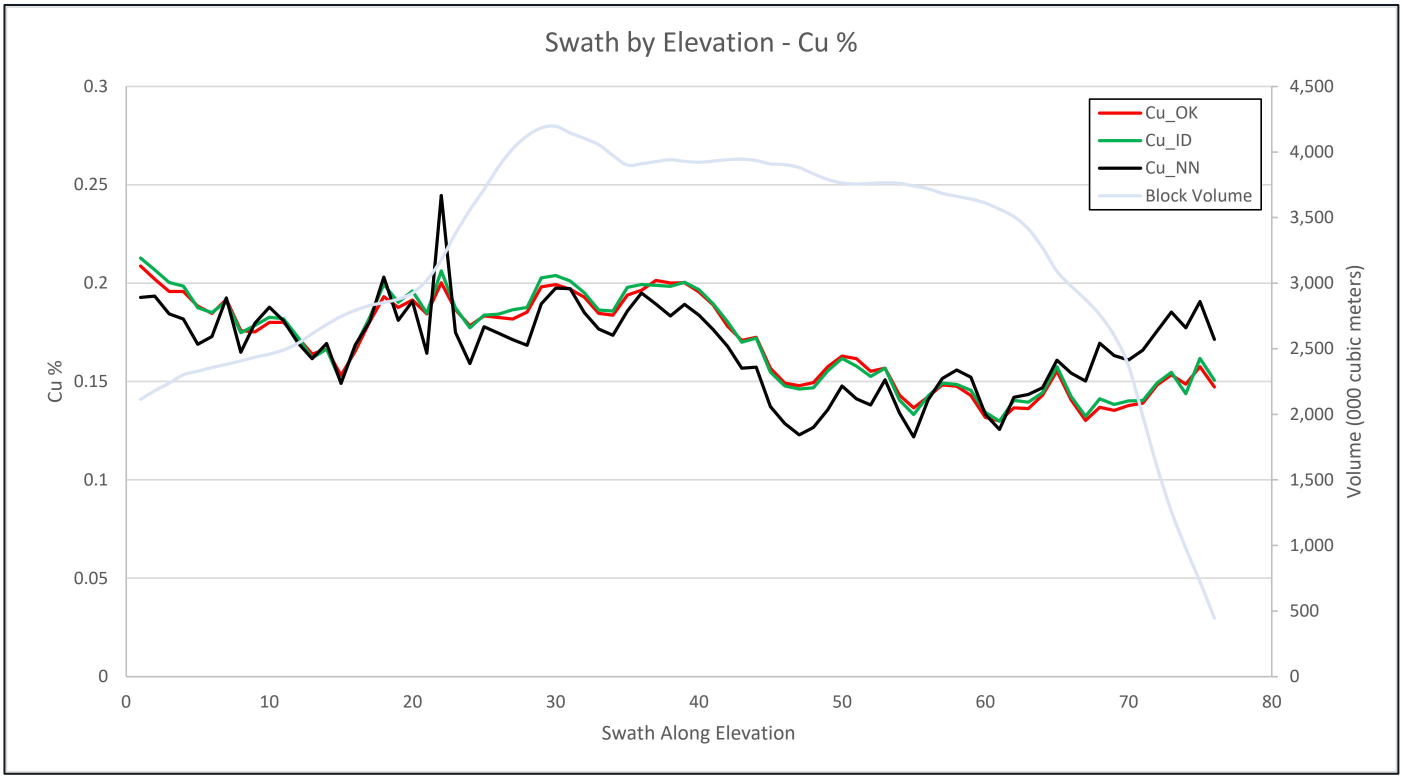
Several swath plots were generated and demonstrate good comparison between nearest neighbour, inverse distance, and kriged copper and gold grades, indicating that the block model is a reasonable representation of the informing data. Swath plots of copper grades by easting and elevation are shown in Figure 14-10 and Figure 14-11, respectively. The trends shown by the composite data (represented by the NN model) are honoured by the block model. The NN estimate compares well with the OK estimate. The comparisons show the effect of the interpolation, which results in smoothing of the block grades, compared to the nearest neighbour grades.

Figure 14-10: Swath of Copper Grades by Easting



Source: Mining Plus, 2022

Figure 14-11: Swath of Copper Grades by Elevation



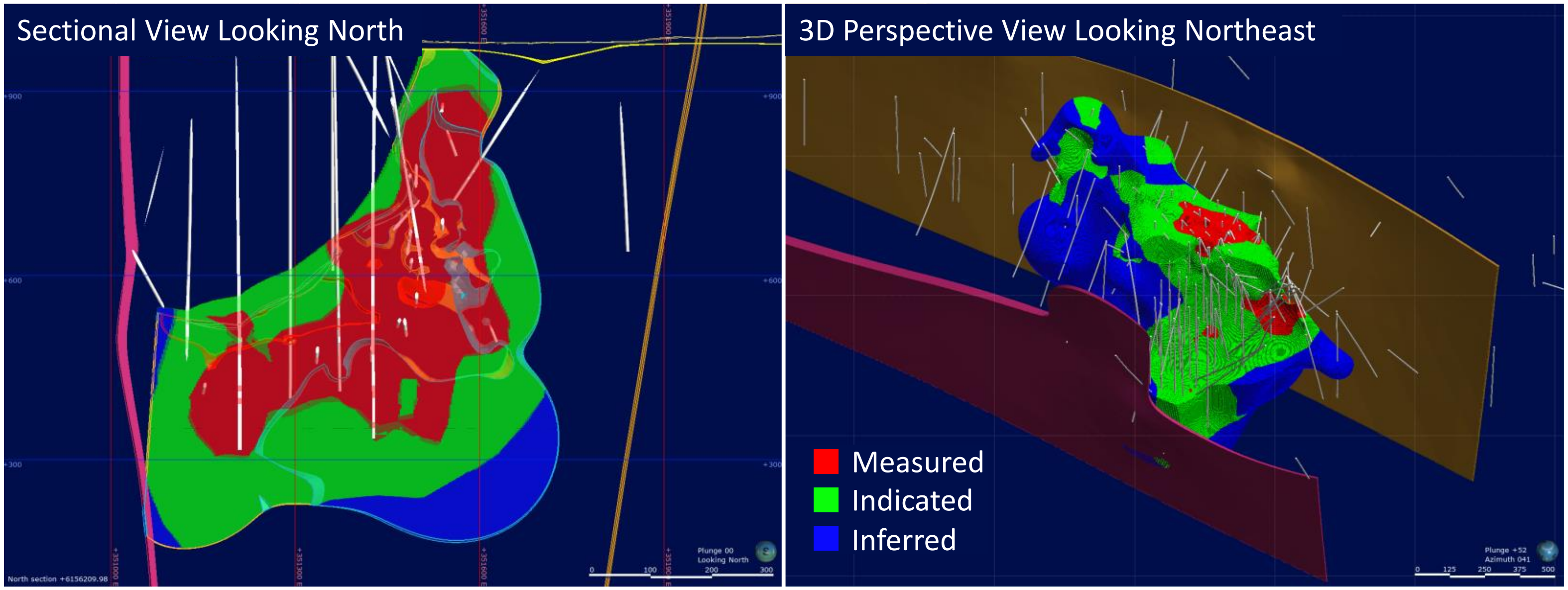
Source: Mining Plus, 2022

14.1.9 Classification of Mineral Resources

Mineral Resources are subdivided, in order of increasing geological confidence, into Inferred, Indicated, and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource, but a lower level of confidence than a Measured Mineral Resource.

Blocks were assigned a preliminary classification based on the variography, drillhole spacing and number of samples in each pass as well as by domain. Search distances for the first pass are half the variogram range and this was used as the initial classification for assigning blocks to the Measured resource category. Blocks estimated in the second pass employed a search distance of the full variogram range and were allocated to the Indicated resource category. Blocks estimated in the third pass that allowed a relaxed search up to three times the range were assigned to the Inferred resource category. The preliminary classification boundaries were then adjusted to create continuity of blocks within the corresponding resource category classification. Resource classification is shown in both sectional and 3D perspective view in Figure 14-12.

Figure 14-12: Central Zone Mineral Resource Classification Shown in Both Sectional and 3D Perspective View



Source: Mining Plus, 2022

14.1.10 Reasonable Prospects for Eventual Economic Extraction

To determine that the mineral resource demonstrates “reasonable prospects for eventual economic extraction”, Mining Plus generated a pit shell using the parameters shown in Table 14-9. The maximum pit shell at revenue factor 100% was used for resource reporting. For blocks that lie outside of the pit shell, a block caving mining method is assumed, and a shape was generated using additional mining and G&A costs of US$8.20/tonne. A simplified NSR/tonne was calculated using the metal prices and recoveries shown in Table 14-10. All blocks within the pit shell and above the internal cut-off of US$8.21/tonne (processing plus G&A costs) are included in the open pit Mineral Resource tabulation. All blocks within the block cave shape and above a cut-off of US$16.41 (US$8.21 processing plus G&A, plus US$8.20 underground costs) are included in the underground Mineral Resource tabulation. Any block lying outside of the pit shell, or the underground block cave shape are not included in the Mineral Resource.

Figure 14-13 illustrates the pit shell and block cave shapes used to constrain the Kwanika Central Mineral Resource.

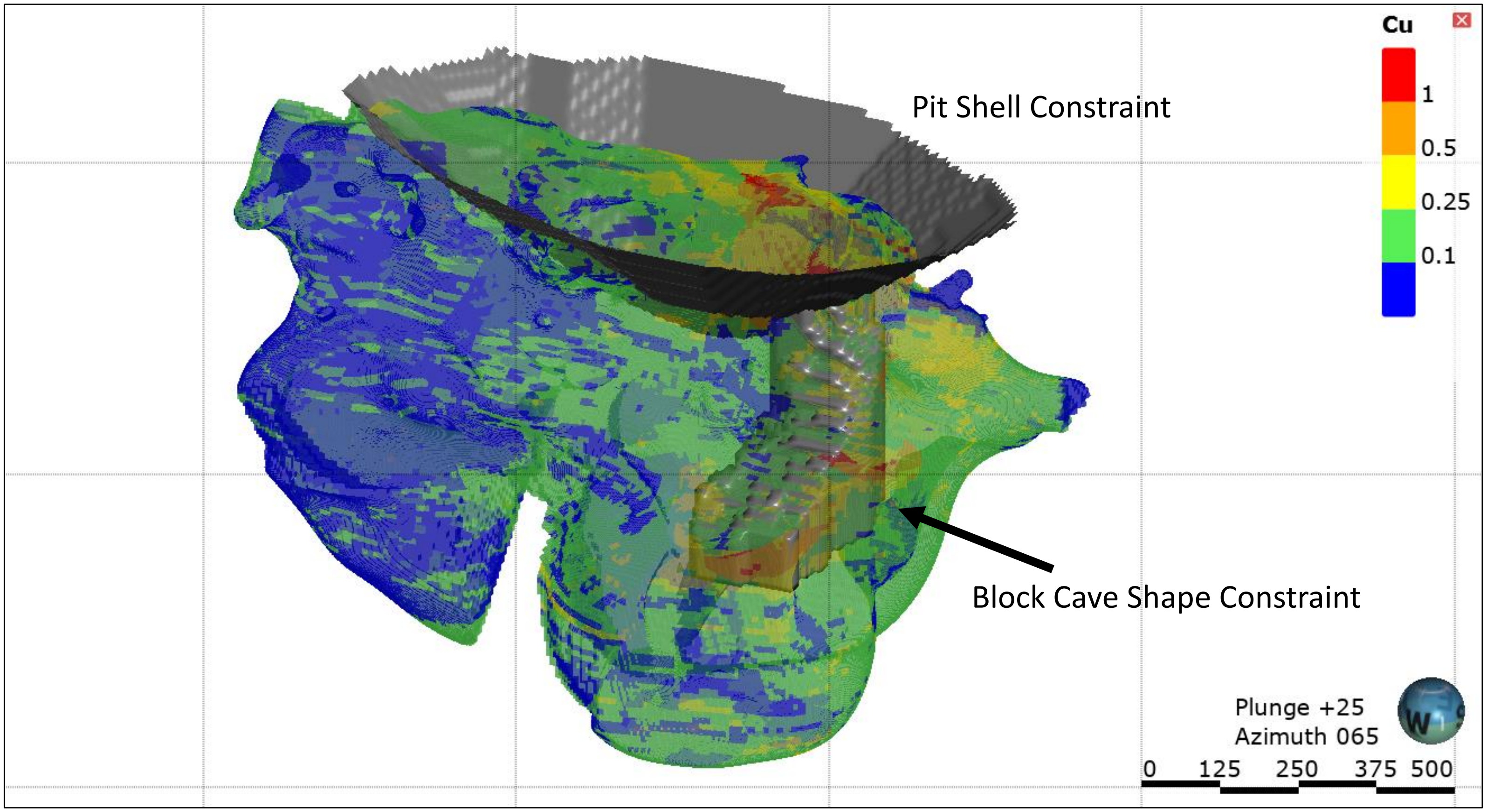
Table 14-9: Parameters Used to Generate Conceptual Pit Shell Constraint

Overall Slope Angles	Degree	
Mining dilution	%	2
Mining loss	%	5
Cu Processing recovery	%	IF([Cu]<0.1, 0.5, IF([Cu]<1, 0.93833*[Cu]^0.0655, 0.95))
Au Processing recovery	%	MIN(0.10*[Au]+0.66, 0.85)
Ag Processing recovery	%	MIN(IF([Ag]<0.5, 0.1, 0.32493 + 0.25676 * LN([Ag])), 0.62)
Costs		
Mining	US$/t	2.39
Additional mining cost	US%/t/bench	0.06
UG mining	US$/t	24.71
Milling + G&A	US$/t	8.21
Sales		
Gold price	US$/oz	1650
Copper price	US$/lb	3.50
Silver price	US$/oz	21.50
Timing		
Process rate	ktpa	9124
Discount rate	%	7
Minimum mining Width	m	30

Table 14-10: Parameters Used to Calculate NSR/tonne

Copper Price	US$/lb	3.5
Gold Price	US$/oz	1650
Silver Price	US$/oz	21.5
Open Pit Copper Recovery	%	If(Cu<0.1, 0.5, If(Cu<1.0,0.93833*Cu^0.0655,0.95))
Open Pit Gold Recovery	%	Min(0.1*Au+0.66,0.85)
Open Pit Silver Recovery	%	Min(If(Ag<0.5,0.1,0.32493+0.25676*LN(Ag)),0.62)
Underground Copper Recovery	%	If(Cu<0.1, 0.5, If(Cu<1.0,0.94712*Cu^0.0649,0.95))
Underground Gold Recovery	%	Min(0.1*Au+0.66,0.85)
Underground Silver Recovery	%	Min(If(Ag<0.5,0.3,0.51169+0.26032*LN(Ag)),0.72)

Figure 14-13: Resource Pit Shell and Block Caving Shapes Used to Constrain the Kwanika



Source: Mining Plus, 2022

14.1.11 Mineral Resource Statement

The Kwanika Central Mineral Resource is reported with an effective date of January 4, 2023 and using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus G&A costs. The underground mineral resources are reported using an economic cut-off of US$ 16.41, which covers the additional underground mining and G&A costs of US$8.20/tonne. Additionally, the Mineral Resource is constrained by an open pit mining shell and underground block caving shape to satisfy reasonable prospects for eventual economic extraction. Table 14-11 shows the Kwanika Central Zone Mineral Resource. Table 14-12 shows the open pit and underground Measured plus Indicated tabulations at various cut-offs. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-11: Mineral Resource Statement - Kwanika Central Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit	8.21	Measured	30.7	0.31	0.31	1.05	210.8	310.5	1,041.7
		Indicated	35.9	0.22	0.19	0.80	174.9	222.0	923.9
		M&I	66.6	0.26	0.25	0.92	385.7	532.5	1,965.6
		Inferred	4.1	0.15	0.15	0.58	13.8	20.1	77.3
	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Underground	16.41	Measured	25.6	0.50	0.61	1.62	284.4	501.3	1,332.6
		Indicated	11.3	0.51	0.65	1.56	126.2	236.7	565.1
		M&I	36.8	0.51	0.62	1.60	410.6	738.0	1,897.8
		Inferred	-	-	-	-	-	-	-

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
3. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
4. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Underground Mineral Resources are reported at an economic cut-off of US$16.41 and constrained by a conceptual block cave shape. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne, and underground mining and G&A costs are assumed to be US$8.20/tonne.
5. Actual SG measurements were interpolated into the block model, with an average SG of 2.74.
6. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
7. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

Table 14-12: Kwanika Central Open Pit and Underground Measured plus Indicated Tabulation at Various Cut-offs

	Economic Cut-off US$	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)
Open Pit - Measured plus Indicated	6.00	69.7	0.25	0.24	0.90
	7.00	67.9	0.26	0.25	0.91
	8.00	66.8	0.26	0.25	0.92
	8.21	**66.6**	**0.26**	**0.25**	**0.92**
	9.00	65.6	0.27	0.25	0.92
	10.00	63.9	0.27	0.26	0.93
	11.00	61.8	0.27	0.26	0.94
	12.00	59.4	0.28	0.27	0.95
	Economic Cut-off US$	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**
Underground - Measured plus Indicated	12.00	38.9	0.49	0.60	1.55
	14.00	38.0	0.49	0.61	1.57
	16.00	37.1	0.50	0.62	1.60
	16.41	**36.8**	**0.51**	**0.62**	**1.60**
	18.00	35.9	0.51	0.63	1.63
	20.00	34.7	0.53	0.65	1.66
	22.00	33.3	0.54	0.67	1.69

14.1.12 Factors That May Affect the Mineral Resource Estimate

The mineral resource estimate is based on limited information and sampling gathered through appropriate techniques diamond drill core holes. The estimate was prepared using industry standard techniques and has been validated for bias and acceptable grade-tonnage characteristics.

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price falling below the assumed price
- Assumptions that all required permits will be forthcoming
- Actual metallurgical recoveries being lower than assumed recoveries in the resource estimate
- Significant increase in mining and process cost than the current assumptions
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the province of British Columbia in terms of environmental, permitting, taxation, socio-economic, marketing, and political. Ridge Geosciences is not aware of any legal or title issues that would materially affect the Mineral Resource estimate.

14.2 Kwanika South Zone

14.2.1 Resource Database

The database used for the Kwanika South resource estimate comprises collar, survey, assay, lithology, alteration, density, and structural information for exploration drilling conducted between 2006 and September 2021. Drilling on the South Zone totalled 19,099 m in 62 holes. A total 8,490 core samples were submitted to the lab for analysis.

14.2.2 Geological Models

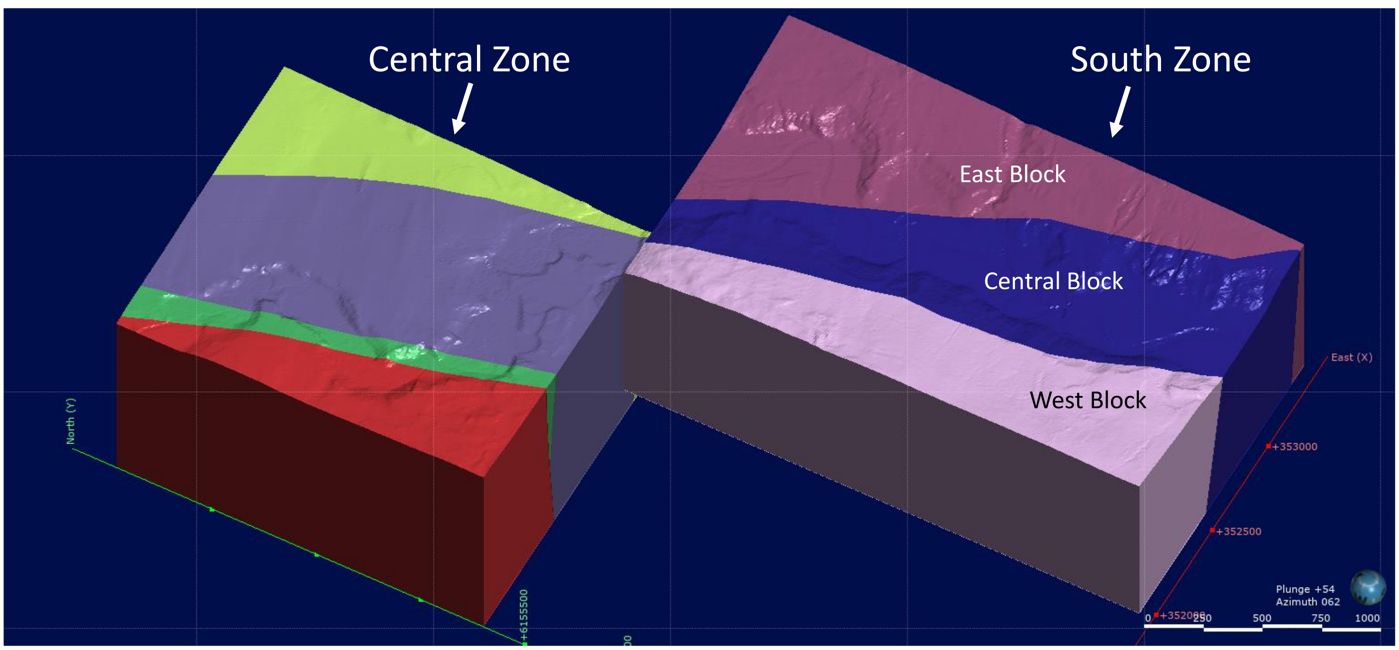
14.2.2.1 Topographic Surfaces

A bare earth Lidar survey was flown over the property in September of 2016. A surface was created from the Lidar data points for both the Central and South zones. These surfaces were triangulated using all points for each zone. The collar data was reconciled onto these high-resolution surfaces.

14.2.2.2 Lithology and Fault Solids Modelling

Structural domains were modelled using surface maps, geology sections and logged lithologies in drillholes. The structural model was inspected to identify the magnitude of displacements, and only faults with significant displacements were selected to be the bounding planes for fault blocks. This resulted in the creation of three blocks for the South Zone as shown in Figure 14-14.

Figure 14-14: Perspective View Showing Fault Blocks and Relative Positions of Central and South Zones, Looking Northeast



Source: Mining Plus, 2022

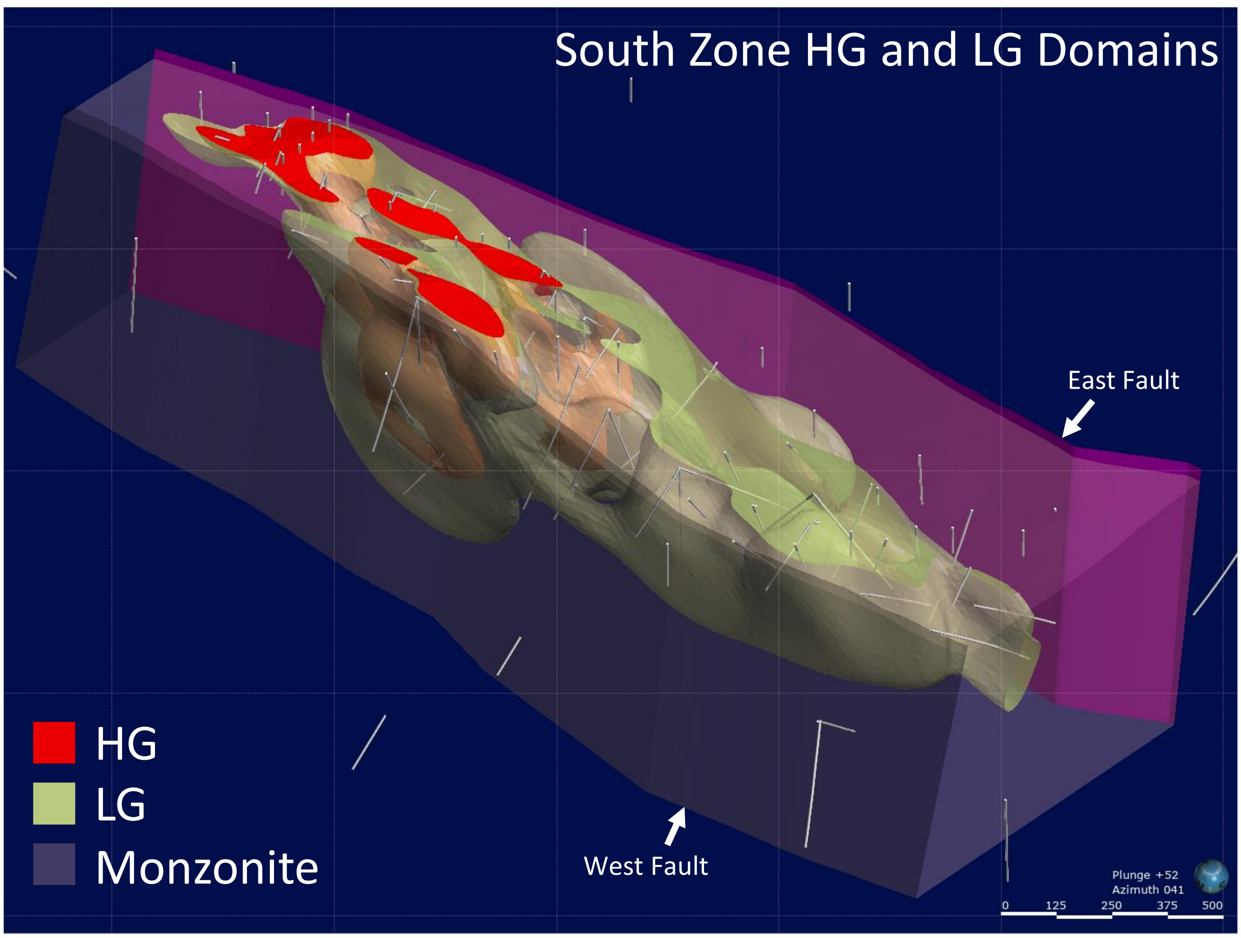
The South Zone contains a mix of dominantly monzonite and monzodiorite lithologies which are around bound by the West Fault. The grade appears to be structurally controlled and not bound by lithology or alteration. There is no clear correlation between grade and alteration, fracturing, or veining. The control of mineralization is not well understood in the South Zone.

The faults and the overburden were modelled for the South Zone. The West Fault was modelled as dipping steeply to the west based on logged fault interceptions. The East Fault was modelled from geophysical interpretation and drilling from outside the modelled area.

14.2.2.3 Estimation Domains

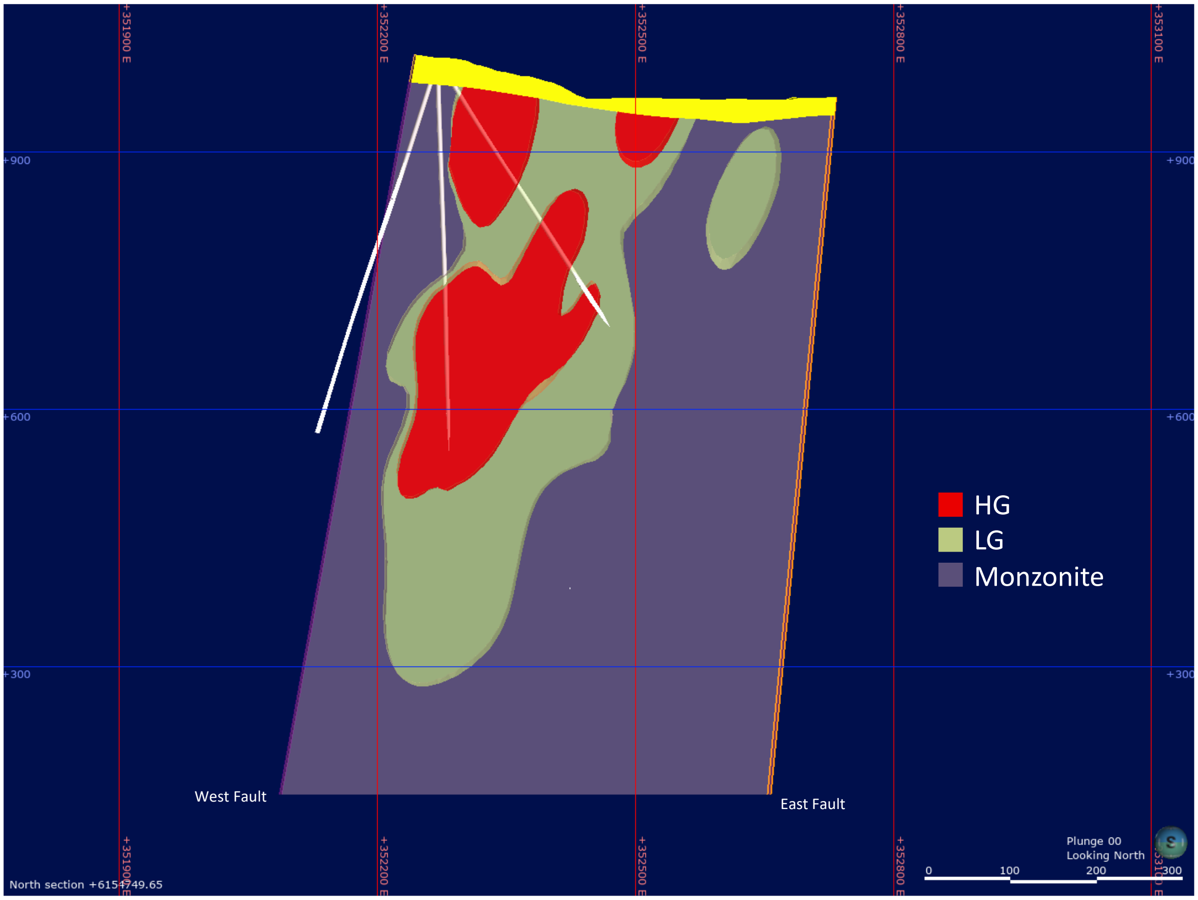
Two grade shells were created using different copper cut-offs. The HG domain uses a 0.2% CuEq cut-off and the low-grade (LG) domain uses a 0.1% CuEq cut-off. The solids were clipped against each other so the higher-grade solids would not extend beyond the lower grade shell. The grade shells were also clipped to the overburden and the West Fault. Figure 14-15 and Figure 14-16 show a 3D perspective view 2D section view of the estimation domains, respectively.

Figure 14-15: 3D Perspective View Showing Estimation Domains at Kwanika South, Looking Northeast



Source: Mining Plus, 2022

Figure 14-16: Representative 2D Section Showing Estimation Domains at Kwanika South, Looking North



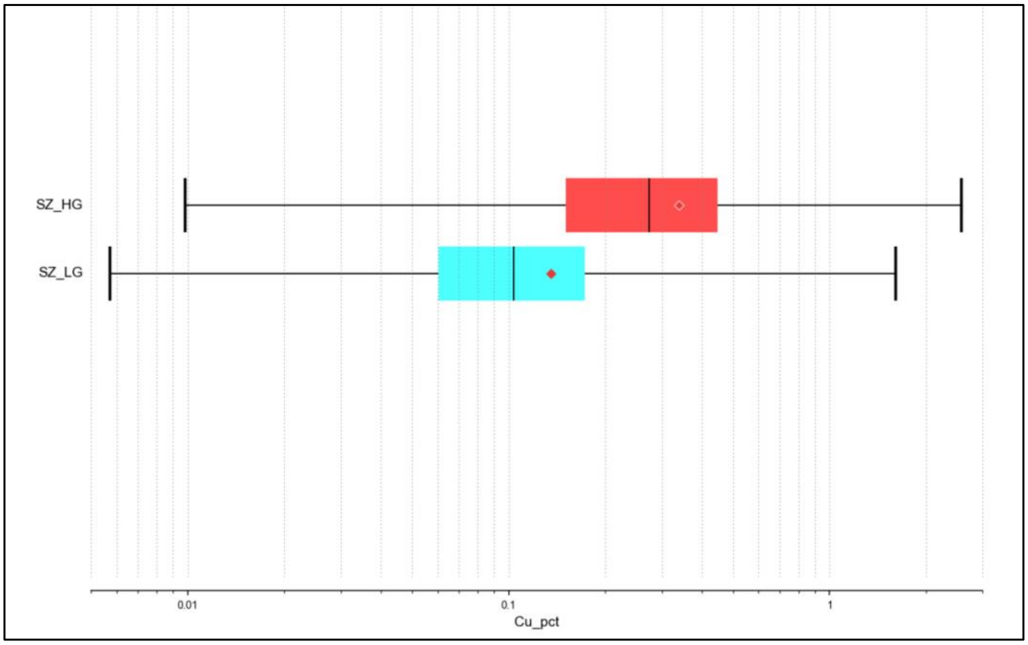
Source: Mining Plus, 2022

14.2.3 Assays, Compositing, and Capping

The assay data was examined to determine a suitable composite interval. The chosen interval should standardize the assay intervals to give an equal weight to each record, but still reflect the variability in the original data as far as possible. The Kwanika drill core was predominantly sampled at an interval of 2 m or less. Assay records were assigned a domain code and then composited to approximate 2 m intervals. The composite interval was varied around an average of the selected 2 m interval while keeping as close as possible to a full 2 m, where required, to avoid excessively short interval composites from forming at domain boundaries or at the ends of holes. All grade distributions are positively skewed and exhibit quite low standard deviation to mean ratios (Coefficient of Variation) in the high-grade domains and only moderately high ratios in the lower grade domains. A boxplot for copper values by domain in shown below in Figure 14-17.

The presence of high-grade outlier values was investigated as these anomalous values could adversely influence the estimate by contributing excessively to the total metal content of the deposit. In all estimation domains and for all elements, the location of the high-grade outliers was not concentrated in an area, but rather scattered throughout each domain. Appropriate capping limits were selected by studying coefficient of variation plots, probability plots and decile analyses plots. Values above the capping limits were reduced to the capping limit. Statistics summarizing composites and capping for each domain is provided in Table 14-13.

Figure 14-17: Boxplot Showing Cu% by South Zone Estimation Domain



Source: Mining Plus, 2022

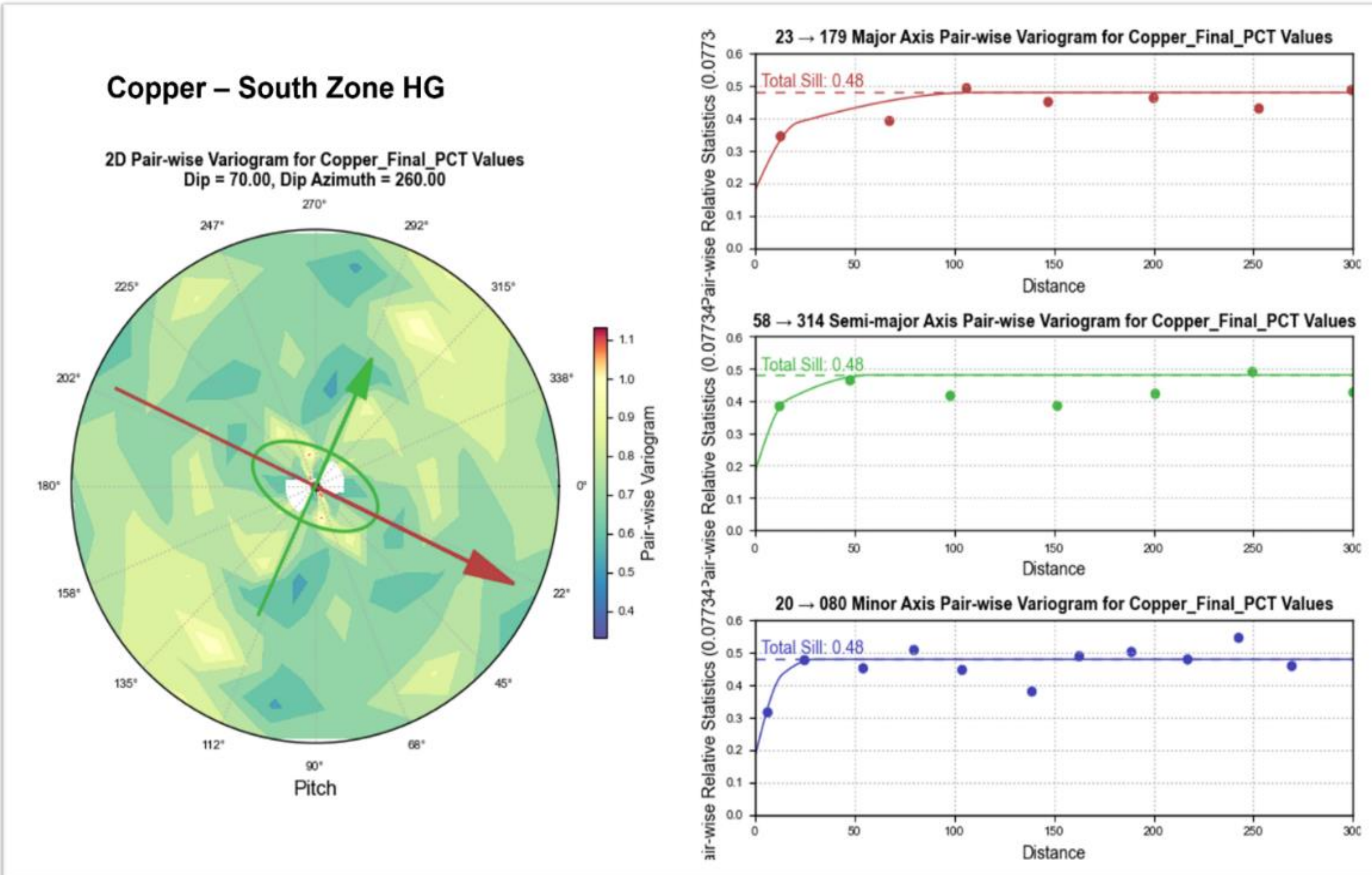
Table 14-13: Capping and Composite Statistics by South Zone Estimation Domain

	High-Grade Zone				Low-Grade Zone			
	Cu%	Au g/t	Ag g/t	Mo ppm	Cu%	Au g/t	Ag g/t	Mo ppm
Total Composites	1,337	1,333	1,333	1,360	2,814	2,354	2,354	2,853
Min Before Capping	0.01	0.01	0.2	0	0.01	0.00	0.1	0
Max Before Capping	2.58	6.62	15.4	2,160	1.61	3.48	18.8	5,374
Mean Before	0.34	0.09	2.0	122	0.14	0.06	1.0	48
Std Dev Before	0.28	0.26	1.7	195	0.12	0.12	1.1	138
CV Before	0.82	2.93	0.8	2	0.89	2.14	1.0	3
Capping Value	NONE	1.39	NONE	834	NONE	0.56	11.7	497
No of Capped Comps	0	3	0	20	0	18	2	19
Mean After	0.34	0.08	2.0	117	0.14	0.05	1.0	44
Std Dev After	0.28	0.11	1.7	166	0.12	0.08	1.0	69
CV After	0.82	1.40	0.8	1	0.89	1.43	1.0	2
Capped %	0.0%	0.2%	0.0%	1.5%	0.0%	0.8%	0.1%	0.7%
Metal % Capped	0.0%	9.1%	0.0%	4.1%	0.0%	6.6%	0.3%	8.8%

14.2.4 Variography

Experimental pairwise relative semi-variograms were calculated and modelled for each metal in each mineralized domain. Spherical two structure models were fitted to experimental semi-variograms in all cases. An example of experimental semi-variograms with fitted model for Cu is shown in Figure 14-18.

Figure 14-18: Model Variogram for Copper in the South Zone HG Estimation Domain



Source: Mining Plus, 2022

All domains had sufficient samples to create adequate experimental semi-variograms for each metal. Strong anisotropy was observed for the most part and directional variogram models were used. The nugget values (i.e., the sample variability at close distance) were established from downhole variograms.

Nugget values vary from 24-38% of the total sill value for all elements in all domains. Major axes range for the short first structures of all the variograms are around 20-70 m and the ranges for the second structure are 110-120 m on average. All variogram model parameters are listed per element in Table 14-14.

Table 14-14: South Zone Variogram Parameters by Metal and Estimation Domain

Domain	Metal	Direction						Structure 1 (Spherical)					Structure 2 (Spherical)				
		Dip	Dip Azi	Pitch	Variance	Nugget	Norm Nugget	Sill	Norm sill	Major	Semi	Minor	Sill	Norm sill	Major	Semi	Minor
HG	Copper	70	260	25	0.1	0.01	0.18	0.01	0.17	22	16	14	0.01	0.13	110	58	30
	Gold	70	260	25	0.1	0.01	0.18	0.01	0.20	22	16	14	0.01	0.14	110	58	30
	Silver	70	260	25	2.8	0.50	0.18	0.46	0.16	22	16	14	0.37	0.13	110	58	30
	Molybdenum	70	260	25	38117	8245	0.22	8146	0.21	22	16	14	12579	0.33	110	58	30
LG	Copper	70	260	20	0.0	0.00	0.10	0.00	0.13	68	14	10	0.00	0.10	122	86	68
	Gold	70	260	20	0.0	0.00	0.10	0.00	0.14	68	14	10	0.00	0.06	122	86	68
	Silver	70	260	20	1.1	0.11	0.10	0.21	0.18	68	14	10	0.15	0.13	122	86	68
	Molybdenum	70	260	20	986.0	167.62	0.17	128	0.13	42	14	10	286	0.29	122	86	68

14.2.5 Block Model

The block model was constructed to fill the domain volumes with 20 m x 20 m x 10 m blocks to best represent the data density, deposit shapes, and to minimize blocks unsupported by data.

Accurate representation of the domain volume was achieved by allowing sub-blocks to be created at domain boundaries. Each parent cell could be split into a minimum sub-block size of 2.5 x 2.5 x 0.5 m. Each sub-block was assigned the estimate derived for the parent block.

The block model size and extents are shown in Table 14-15.

Table 14-15: Kwanika South Zone Block Model Dimensions

Direction	Minimum	Maximum	Block Size (m)	# Blocks
Easting	351850	353330	20	74
Northing	6153230	6155710	20	124
Elevation	150	1100	10	95

14.2.6 Interpolation Methods

Anisotropic search orientations along mineralization trends were used to select data informing block estimates. Search radii were based on the variogram ranges. Ordinary kriging was used to estimate all blocks into the model in three estimation passes whereby each successive pass utilized a less restrictive sample search strategy to estimate any remaining un-estimated blocks. Search radii for the first estimation pass equals half of the variogram range. The second pass increases the search to the range and the third pass further expands to twice the variogram range.

The orientation of the sample search ellipse is aligned with the variogram models for copper and the search ranges for the successive passes are factors of the modelled variogram range. The search ellipse orientations in all cases display the strongest trend NNW-SSE with a steep dip towards the west and a northward plunge. All domains were estimated using ordinary kriging. Search orientations and sample selection criteria for each domain is shown below in Table 14-16.

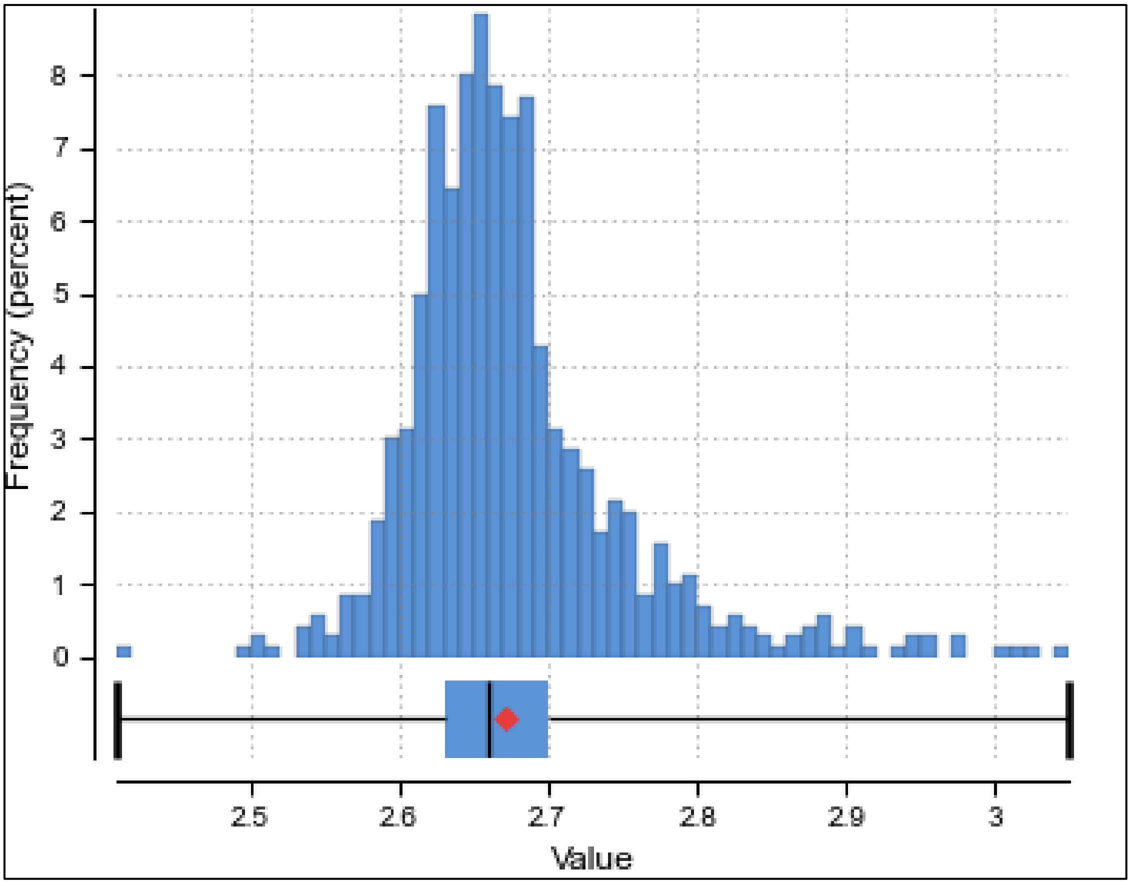
Table 14-16: Kwanika South Zone Estimation Parameters

Domain	Metal	Ellipsoid Ranges (m)			Ellipsoid Directions			Number of Samples			Capped Value
		Max	Int	Min	Dip	Dip Azi	Pitch	Min	Max	Max/Hole	
HG	Cu (%)	55	39	15	70	260	25	8	20	5	-
		110	58	30	70	260	25	6	20	4	-
		330	174	90	70	260	25	4	20	3	-
	Au (g/t)	55	39	15	70	260	25	8	20	5	1.39
		110	58	30	70	260	25	6	20	4	1.39
		330	174	90	70	260	25	4	20	3	1.39
	Ag (g/t)	55	39	15	70	260	25	8	20	5	-
		110	58	30	70	260	25	6	20	4	-
		330	174	90	70	260	25	4	20	3	-
	Mo (ppm)	55	39	15	70	260	25	8	20	5	833.6
		110	58	30	70	260	25	6	20	4	833.6
		330	174	90	70	260	25	4	20	3	833.6
LG	Cu (%)	61	43	34	70	260	20	8	20	5	-
		122	86	68	70	260	20	6	20	4	-
		366	258	204	70	260	20	4	20	3	-
	Au (g/t)	61	43	34	70	260	20	8	20	5	0.56
		122	86	68	70	260	20	6	20	4	0.56
		366	258	204	70	260	20	4	20	3	0.56
	Ag (g/t)	61	43	34	70	260	20	8	20	5	11.7
		122	86	68	70	260	20	6	20	4	11.7
		366	258	204	70	260	20	4	20	3	11.7
	Mo (ppm)	61	43	34	70	260	20	8	20	5	496.6
		122	86	68	70	260	20	6	20	4	496.6
		366	258	204	70	260	20	4	20	3	496.6

14.2.7 Density Assignment

A total of 505 SG measurements were provided for the Central Zone. Values ranged from 2.51 to 3.05, with an average of 2.68. The values were uses to estimate density into the block model using Simple Kriging with a mean value of 2.68. A histogram of SG values for the South Zone is shown in Figure 14-19.

Figure 14-19: Histogram of SG Values for the Kwanika South Zone



Source: Mining Plus, 2022

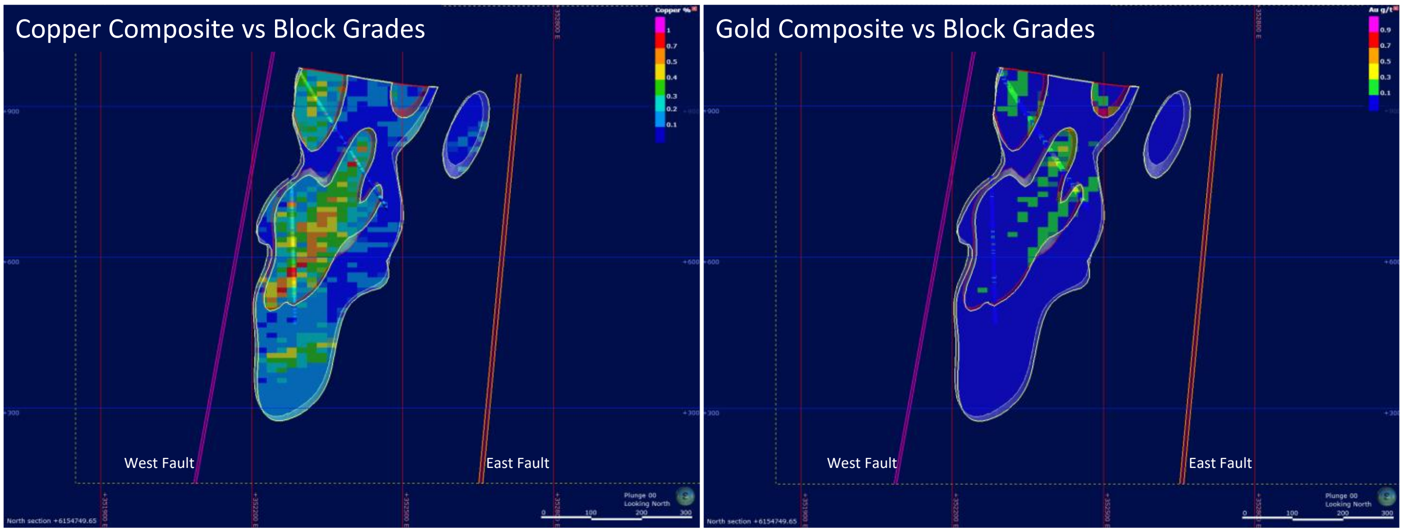
14.2.8 Block Model Validation

The Kwanika South block model was validated using the following methods:

- Visual comparison of colour-coded block grades to drillhole composite grades in sectional view,
- Spot check on individual blocks to confirm that composite selection and kriging weights were applied according to the search strategy,
- Global comparison of a NN model with the OK model,
- Swath plot analysis comparing NN and OK grades for easting, northing, and elevation

The visual comparison of block model grades with composite grades for copper and gold show a good correlation between values and no large discrepancies are apparent. Figure 14-20 shows colour-coded block model copper and gold grades with the drillhole composite grades in a representative section looking north.

Figure 14-20: Visual Comparison of Composite and Block Copper and Gold Grades Along Northing 6154750, Looking North



Source: Mining Plus, 2022

A NN model was completed for copper and gold to serve as a check against the resource model. The NN interpolation method simply assigns a block the same grade as its closest composite. These models are intended to represent a theoretical unbiased estimate of the average grade when no cut-off grade is imposed and is a good basis for checking performance of different estimation methods. The NN model utilized the same search criteria as the OK model except uses a single composite to estimate a block. A comparison of NN, ID, and OK grades was made for all blocks within the LG and HG domains, classified as Measured, Indicated, or Inferred, and at a zero percent copper cut-off and is summarized below in Table 14-17. The OK copper and gold grades are around 6-7% lower than NN grades. This is primarily influenced by a few isolated high-grade holes that are contributing to larger high-grade tonnage in the NN estimate versus the averaged lower OK grades in the same area. When comparing only those blocks estimated within the first two passes, grades compare more favourably, as shown in Table 14-18.

Table 14-17: Comparison of South Zone NN, IDW, and OK Model Grades in All Resource Classes at a 0% Copper Cut-off

Method	Cu%	Au g/t
NN	0.163	0.053
ID	0.160	0.051
OK	0.153	0.049

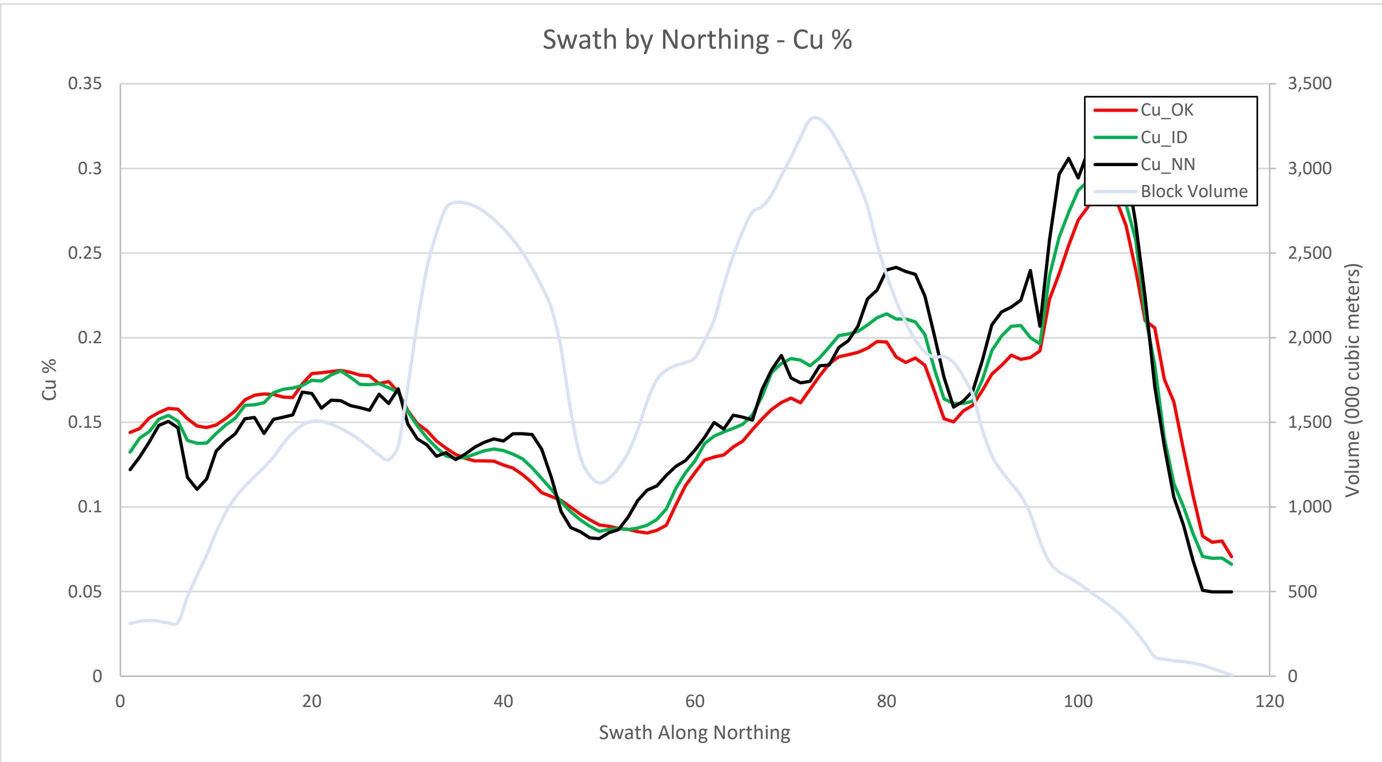
Table 14-18: Comparison of South Zone NN, IDW, and OK Model Grades in Measured and Indicated at a 0% Copper Cut-off

Method	Cu%	Au g/t
NN	0.161	0.056
ID	0.158	0.054
OK	0.160	0.054

Several swath plots were generated and demonstrate good comparison between nearest neighbour, inverse distance, and kriged copper and gold grades, indicating that the block model is a reasonable representation of the informing data. Swath plots of copper grades by Northing are shown in Figure 14-21. The trends shown by the composite data (represented by the NN model) are honoured by the block model. The NN estimate compares well with the OK estimate.

The comparisons show the effect of the interpolation, which results in smoothing of the block grades, compared to the nearest neighbour grades.

Figure 14-21: Swath of Copper Grades by Northing



Source: Mining Plus, 2022

14.2.9 Classification of Mineral Resources

Mineral Resources are subdivided, in order of increasing geological confidence, into Inferred, Indicated, and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource, but a lower level of confidence than a Measured Mineral Resource.

Given the wider drill spacing and overall limited understanding of controls on mineralization, all blocks within the estimation domains at South Zone are classified as Inferred.

14.2.10 Reasonable Prospects for Eventual Economic Extraction

To determine that the mineral resource demonstrates "reasonable prospects for eventual economic extraction", Mining Plus generated a pit shell using the parameters shown in Table 14-19. A simplified NSR/tonne was calculated using the metal prices and recoveries shown in Table 14-20. All blocks within the pit shell and above the internal cut-off of US$8.21/tonne (processing plus G&A costs) are included in the open pit Mineral Resource tabulation. While mineralization does occur outside of the pit shell, it does not exist in sufficient quantity or grade to satisfy reasonable prospects for eventual economic extraction and is therefore not included in the mineral resource statement. There are no underground mineral resources at Kwanika South.

Table 14-19 illustrates the pit shells used to constrain the Kwanika South Mineral Resource.

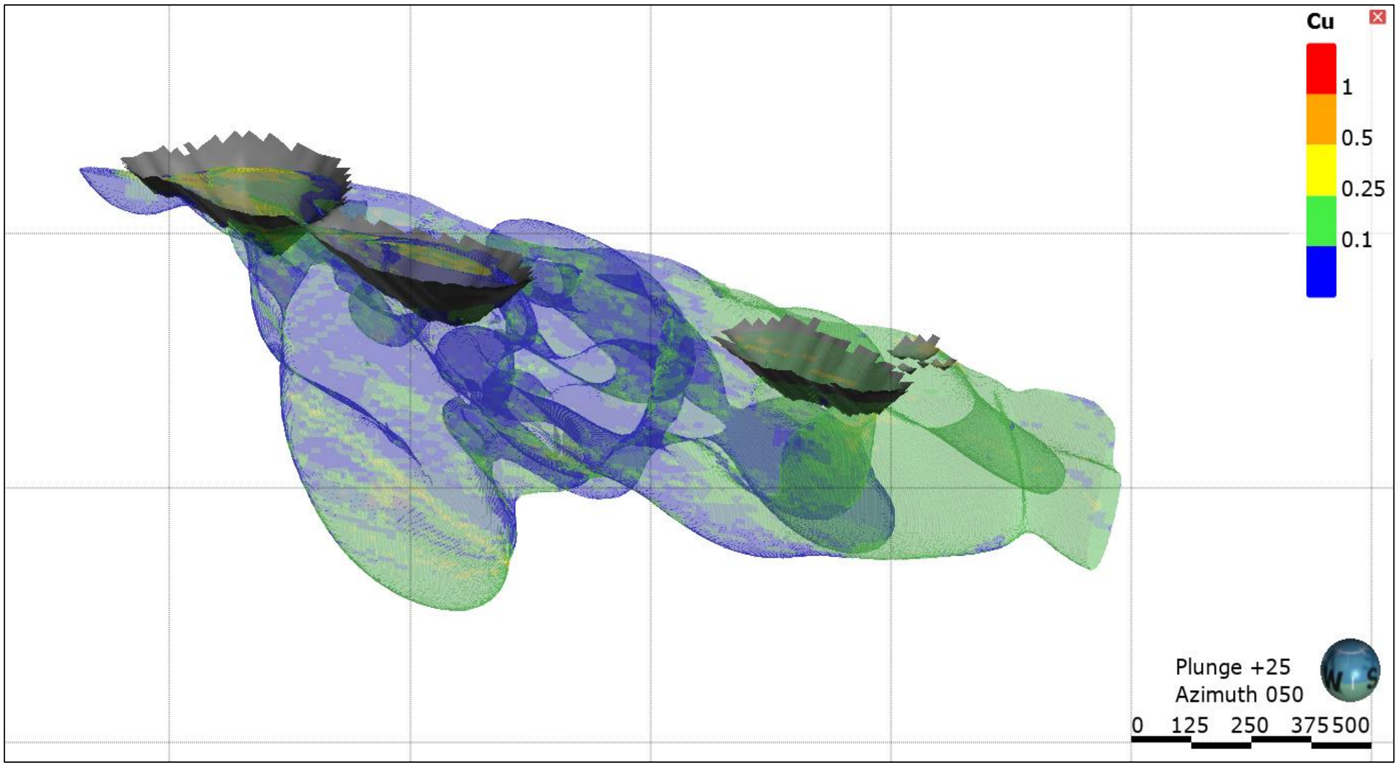
Table 14-19: Parameters Used to Generate Conceptual Pit Shell Constraint

Overall slope angles	degree	
Mining dilution	%	2
Mining loss	%	5
Cu Processing recovery	%	IF([Cu]<0.1, 0.5, IF([Cu]<1, 0.93833*[Cu]^0.0655, 0.95))
Au Processing recovery	%	MIN(0.10*[Au]+0.66, 0.85)
Ag Processing recovery	%	MIN(IF([Ag]<0.5, 0.1, 0.32493 + 0.25676 * LN([Ag])), 0.62)
Costs		
Mining	US$/t	2.39
Additional mining cost	US%/t/bench	0.06
UG mining	US$/t	24.71
Milling + G&A	US$/t	8.21
Sales		
Gold price	US$/oz	1650
Copper price	US$/lb	3.50
Silver price	US$/oz	21.50
Timing		
Process rate	ktpa	9124
Discount rate	%	7
Minimum mining Width	m	30

Table 14-20: Parameters Used to Calculate NSR/tonne

Description	Unit	Value
Copper Price	US$/lb	3.5
Gold Price	US$/oz	1650
Silver Price	US$/oz	21.5
Open Pit Copper Recovery	%	If(Cu<0.1, 0.5, If(Cu<1.0,0.93833*Cu∧0.0655,0.95))
Open Pit Gold Recovery	%	Min(0.1*Au+0.66,0.85)
Open Pit Silver Recovery	%	Min(If(Ag<0.5,0.1,0.32493+0.25676*LN(Ag)),0.62)

Figure 14-22: Pit Shell Used to Constrain the Kwanika South Mineral Resource, Looking NE



Source: Mining Plus, 2022

14.2.11 Mineral Resource Statement

The Kwanika South Mineral Resource is reported with an effective date of January 4, 2023 and using an economic cut-off of US$8.21 for open pit resources, which equates to processing plus G&A costs. Additionally, the Mineral Resource is constrained by an open pit mining shell to satisfy reasonable prospects for eventual economic extraction. Table 14-21 shows the Kwanika South Zone Mineral Resource. Table 14-22 shows the tabulation at various cut-offs. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-21: Mineral Resource Statement – Kwanika South Zone

	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
Open Pit	**8.21**	Inferred	25.4	0.28	0.06	1.68	155.0	52.4	1,373.9

Notes:

1. The Mineral Resources have been compiled by Mr. Brian Hartman of Ridge Geoscience LLC, and subcontractor to Mining Plus. Mr. Hartman is a Registered Member of the Society for Mining, Metallurgy & Exploration, and a Practicing Member with Professional Geoscientists Ontario. Mr. Hartman has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.

2. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. The totals contained in the above table have been rounded to reflect the relative uncertainty of the estimate. Rounding may cause some computational discrepancies.
3. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
4. Open Pit Mineral Resources are reported on an in-situ basis at an economic cut-off of US$8.21 and constrained by an economic pit shell. Cut-offs are based on assumed prices of US$3.50/lb for copper, US$21.50/oz for silver, and US$1650/oz for gold. Assumed metallurgical recoveries are based on a set of recovery formulas derived from recent metallurgical testwork. Maximum recoveries were limited to 95% for Cu, 85% for Au and 62% for Ag. Milling plus G&A costs were assumed to be US$8.21/tonne.
5. Actual SG measurements were interpolated into the block model, with an average SG of 2.68.
6. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
7. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

Table 14-22: Kwanika South Tabulation at Various Cut-offs

	Economic Cut-off US$	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)
Open Pit & Inferred	6.00	26.0	0.27	0.06	1.66
	7.00	25.8	0.27	0.06	1.67
	8.00	25.5	0.28	0.06	1.68
	8.21	**25.4**	**0.28**	**0.06**	**1.68**
	9.00	25.0	0.28	0.06	1.69
	10.00	24.4	0.28	0.07	1.71
	11.00	23.8	0.29	0.07	1.73
	12.00	22.9	0.29	0.07	1.76

14.2.12 Factors That May Affect the Mineral Resource Estimate

The mineral resource estimate is based on limited information and sampling gathered through appropriate techniques diamond drill core holes. The estimate was prepared using industry standard techniques and has been validated for bias and acceptable grade-tonnage characteristics.

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price falling below the assumed price
- Assumptions that all required permits will be forthcoming
- Actual metallurgical recoveries being lower than assumed recoveries in the resource estimate
- Significant increase in mining and process cost than the current assumptions
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the province of British Columbia in terms of environmental, permitting, taxation, socio-economic, marketing, and political. Ridge Geosciences is not aware of any legal or title issues that would materially affect the Mineral Resource estimate.

14.3 Stardust

14.3.1 Key Assumptions and Basis of Estimate

The database for the CCS Zone contains 206 drillholes representing 74,253 m of drilling (excluding 3 holes completed in 2021 which have not yet been incorporated into the model). Fifty-eight of these holes (38,329 m) have been completed between 2018 and 2020 by Sun Metals. Grade estimation is based on 186 drillholes and 3,124 composites of nominal 2.0 m lengths.

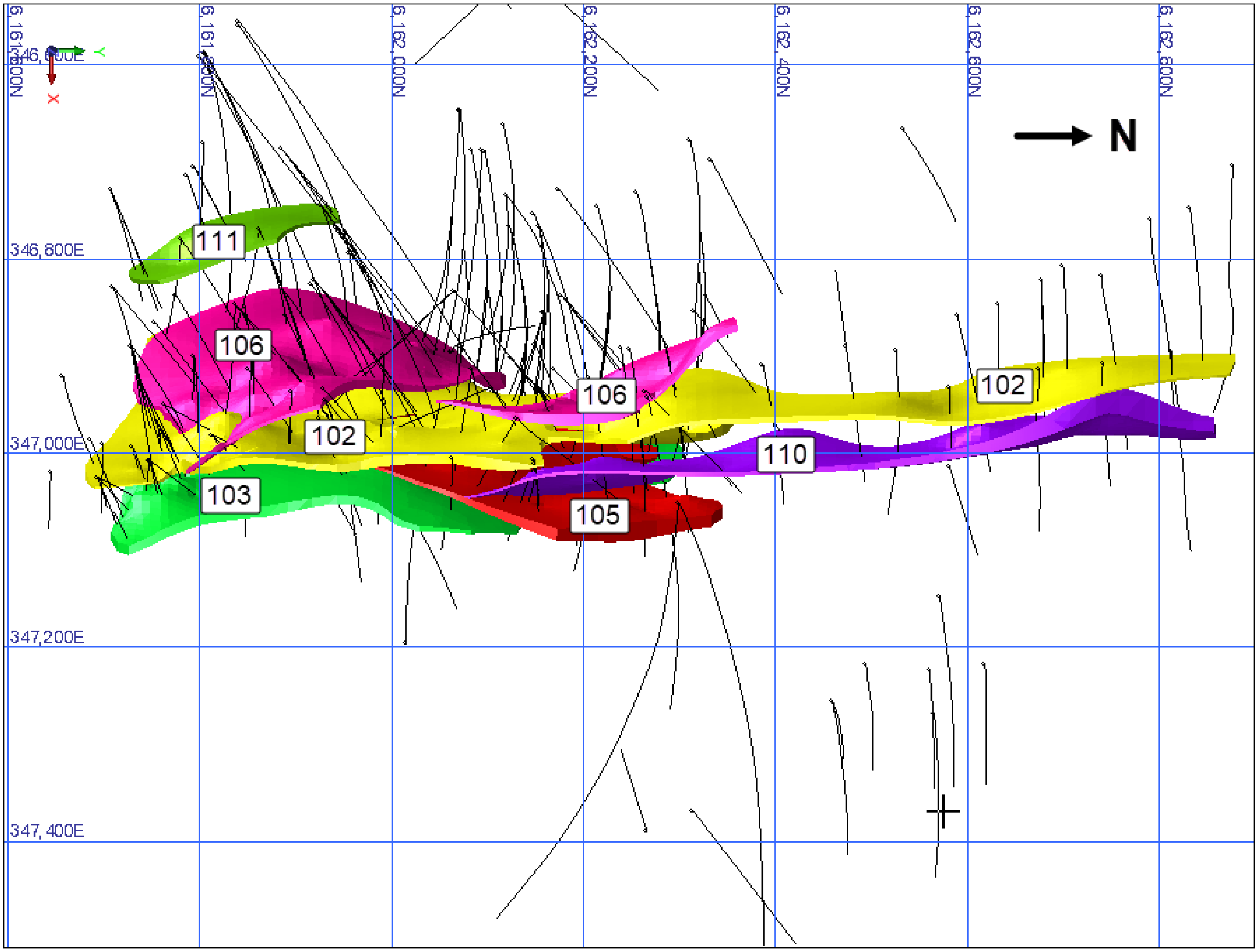
Mineral resources were estimated for gold, silver, and copper. Significant grades of Zinc have been encountered but the distribution is highly irregular and would not likely justify the additional cost of extraction.

14.3.2 Geological Models

Geological wireframe models of the mineralized skarn zones were initially generated in Leapfrog Geo software. The wireframes were then clipped using Surpac Vision software to define the extents of the individual zones that displayed reasonable prospects for economic extraction.

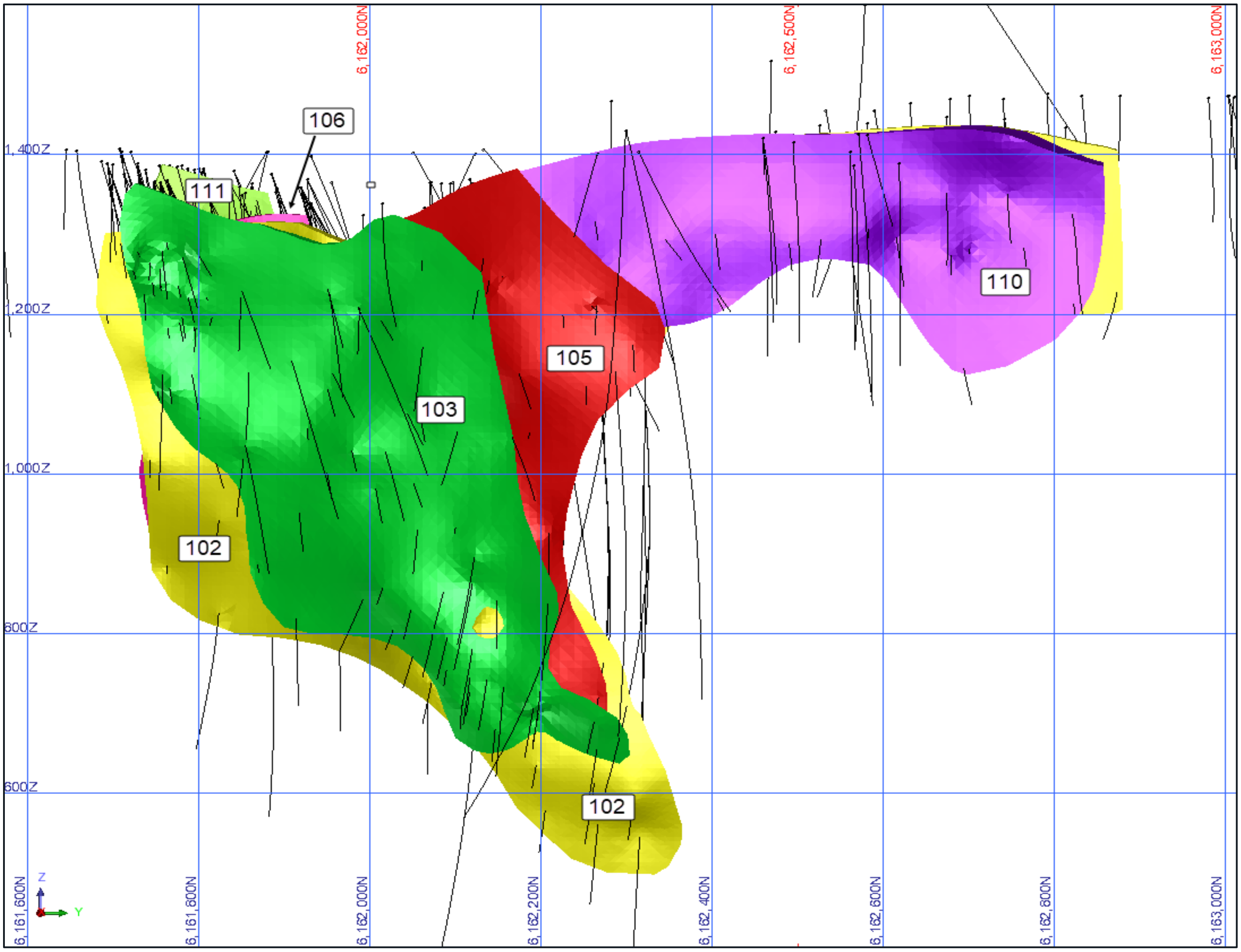
Out of the initial 11 zones and 5 sub-zones (splays), 6 were selected that contained sufficient sampling information and grades to qualify as potential mineral resources. These zones were assigned their initial integer codes with a ''10'' prefix, so they are not consecutive. These mineral zones collectively extend around 1,200 m along strike and have been intersected from surface down to 900 m in depth and are shown in Figure 14-23 and Figure 14-24. *Note: In the press release dated May 17, 2021, Zone 102 is referred to as "Zone 2". The codes have been modified to avoid confusion with historic mineral zone names on the property.*

Figure 14-23: Canyon Creek Mineral Zone Wireframes - Plan View



Source: Geosim, 2022.

Figure 14-24: Canyon Creek Mineral Zone Wireframes - Looking West



Source: Geosim, 2022.

14.3.3 Exploratory Data Analysis

For this modelling exercise it was decided to use the 'best-fit' method of compositing. This procedure produces samples of variable length, but of equal length within a contiguous drillhole zone, ensuring the composite length is as close as possible to the nominated composite length. In this case, the nominated length was set at 2 m with a tolerance of 50% meaning that composite widths for a given zone intercept could range from 1 to 3 metres. This also has the advantage of avoiding partial composites at the beginning and end of the zone intercepts.

The composite intervals were extracted by determining the drillhole intercepts within the wireframe models of each zone. If part of the interval was not sampled, then the values were assumed to be '0' and the composite grade was diluted. Statistics of the composites within the zone models are presented in Table 14-23.

Table 14-23: Composite Statistics - Canyon Creek Zone

	Au (g/t)	Ag (g/t)	Cu (%)
n	3124	3124	3124
Min	0.000	0.0	0.000
Max	20.681	542.8	10.733
Mean	0.596	10.9	0.499
Median	0.149	2.6	0.136
Std Dev	1.320	25.0	0.983
COV	2.213	2.3	1.969

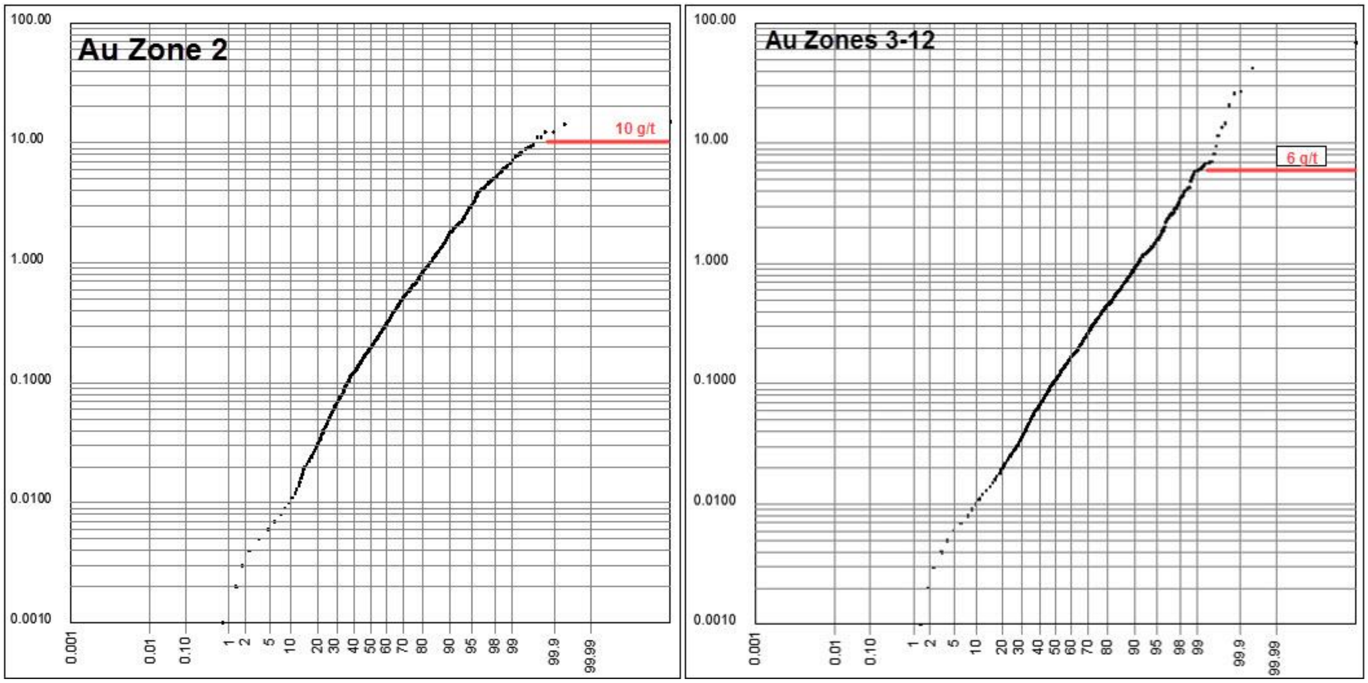
14.3.4 Grade Capping and Outlier Restriction

Grade distribution in the composited sample data was examined to determine if grade capping or special treatment of high outliers was warranted. A decile analyses was performed on the composites within the zone constraints and log probability plots examined (Figure 14-25 to Figure 14-27).

Since Zone 102 contained just over half of the total composites and more consistent high grades, it was analyzed separately. The other 5 zones were analyzed collectively.

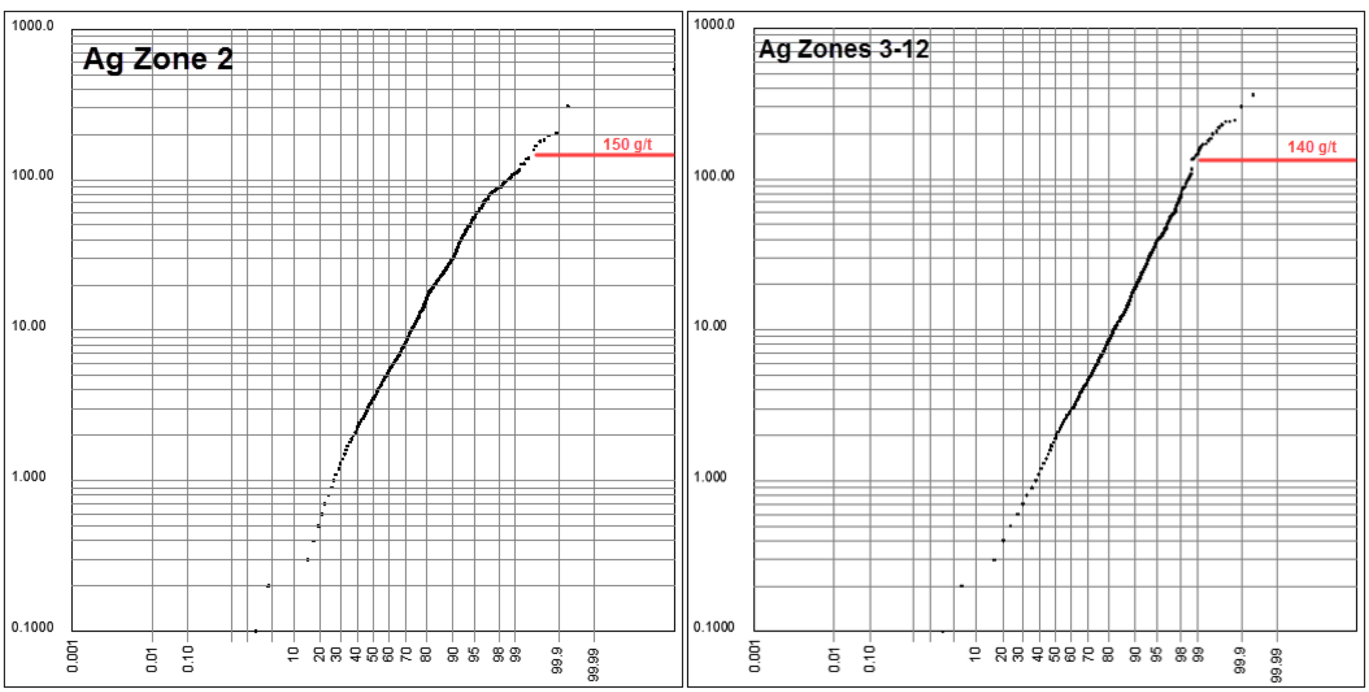
Cap grades are summarized in Table 14-24 and statistics of capped composites in Table 14-25.

Figure 14-25: CPP Charts and Capping Thresholds - Gold



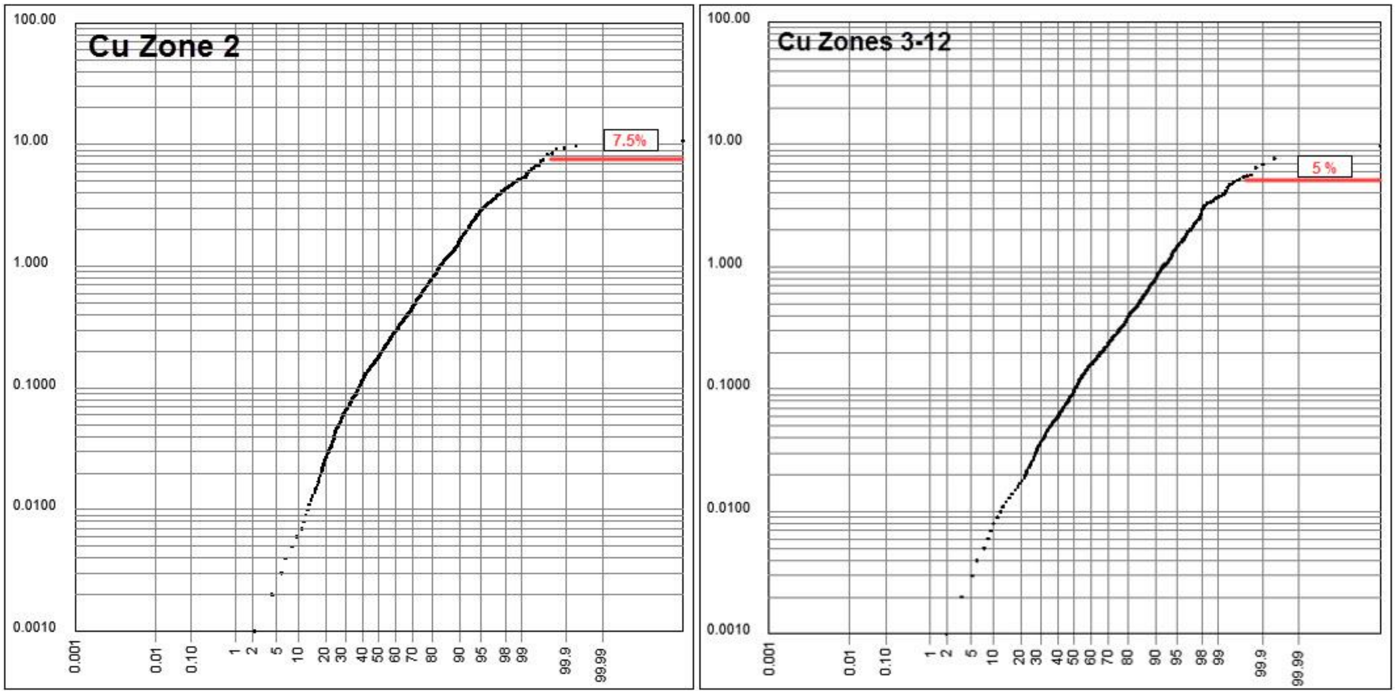
Source: Geosim, 2022.

Figure 14-26: CPP Charts and Capping Thresholds - Silver



Source: Geosim, 2022.

Figure 14-27: CPP Charts and Capping Thresholds - Copper



Source: Geosim, 2022.

Table 14-24: Grade Caps

	Zone 102	Zones 103-111
Cap Au g/t	10	6
Cap Ag g/t	150	140
Cap Cu %	7.5	5

Table 14-25: Capped Composite Statistics

	Au g/t	Ag g/t	Cu %
n	3124	3124	3124
Min	0.000	0.0	0.000
Max	13.641	150.0	7.500
Mean	0.586	10.7	0.496
Median	0.149	2.6	0.136
Std Dev	1.222	22.3	0.954
COV	2.087	2.1	1.925

14.3.5 Density Assignment

The drilling database includes 9,325 specific gravity measurements from drill core collected between 1997 and 2020. The vast bulk of this data (98%) was collected between 2018 and 2020 from the 421 zone which is part of Zone 102. In order to evaluate a reasonable bulk density for tonnage calculations, a block model estimate was run using 1631 SG data points within Zone 102 with a search range of 50 m. The method used was IDW to the second power (ID2). A minimum of 3 and maximum of 12 samples were used to estimate a block. The mean and median SG values of blocks within 5 m of a sample and above a potentially economic cut-off grade were then calculated. The median value of 3.42 was selected.

14.3.6 Variography

Semi-variograms were constructed using composite data from Zone 102 which had the most drill intercepts. Due to the thin nature of the other mineral zones, there were insufficient sample pairs available to model reasonable variograms. Directional semi-variograms for Cu, Au and Ag constructed in the plane of the number 2 zone showed nested spherical structures with ranges of 31 m for Cu, 36 for Au and 41.5 m for Ag (Table 14-26). There was no clear anisotropy evident in the plane of the mineralized structure.

Table 14-26: Variogram Models

Element	Nugget	Sill	Range	Sill	Range
	co	c1	a1	c2	a2
Au	0.18	0.43	9.9	0.39	36
Ag	0.26	0.42	8.5	0.32	41.5
Cu	0.22	0.34	9.9	0.44	31

14.3.7 Interpolation Methods

A block model with block dimensions of 0.5 x 3.0 x 2.5 m was created using Geovia-Surpac© software. The narrow width in the x direction was chosen in order to investigate model economics using a column processing function to impose minimum mining widths. Model extents are shown in Table 14-27.

Table 14-27: Block Model Extents

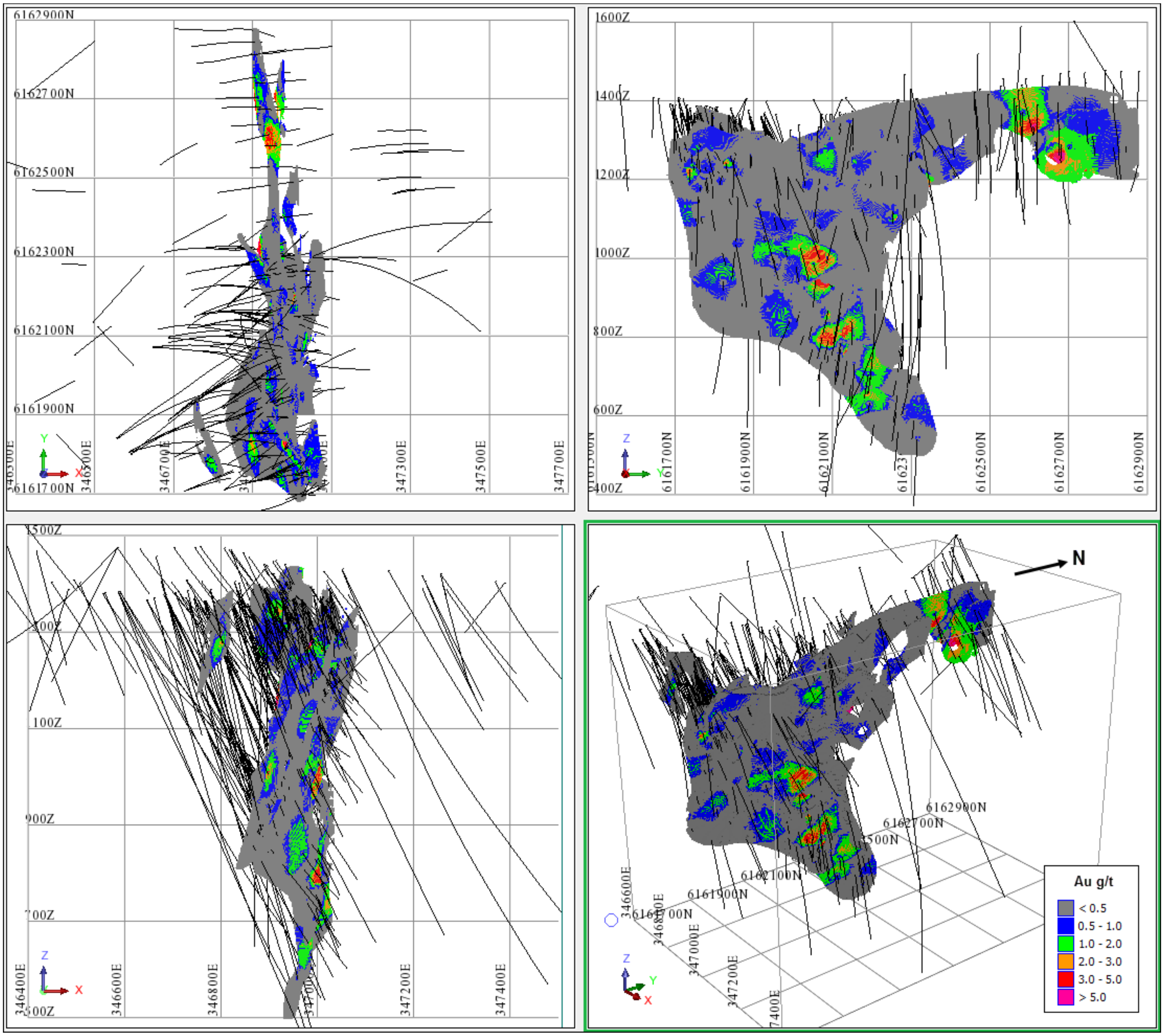
Direction	Min	Max	Dist (m)	size (m)
x	346650	347460	810	0.5
y	6161550	6162900	1350	3.0
z	250	1500	1250	2.5

Inverse distance cubed weighting to the third power (ID3) interpolation was carried out within the zone wireframes in a single pass using a maximum search distance of 100 m in the plane of the zones. A minimum of 3 and maximum of 12 composites were used for grade estimation. Anisotropic interpolation was used with each block being assigned a dip and dip azimuth parallel to a trend surface based on the zone geometry. These values were used as input to define the search ellipse for each block. The major and semi-major axes were the same dimension and the ratio to the minor axis was 3:1.

A nearest neighbour model was also estimated to assist in model validation.

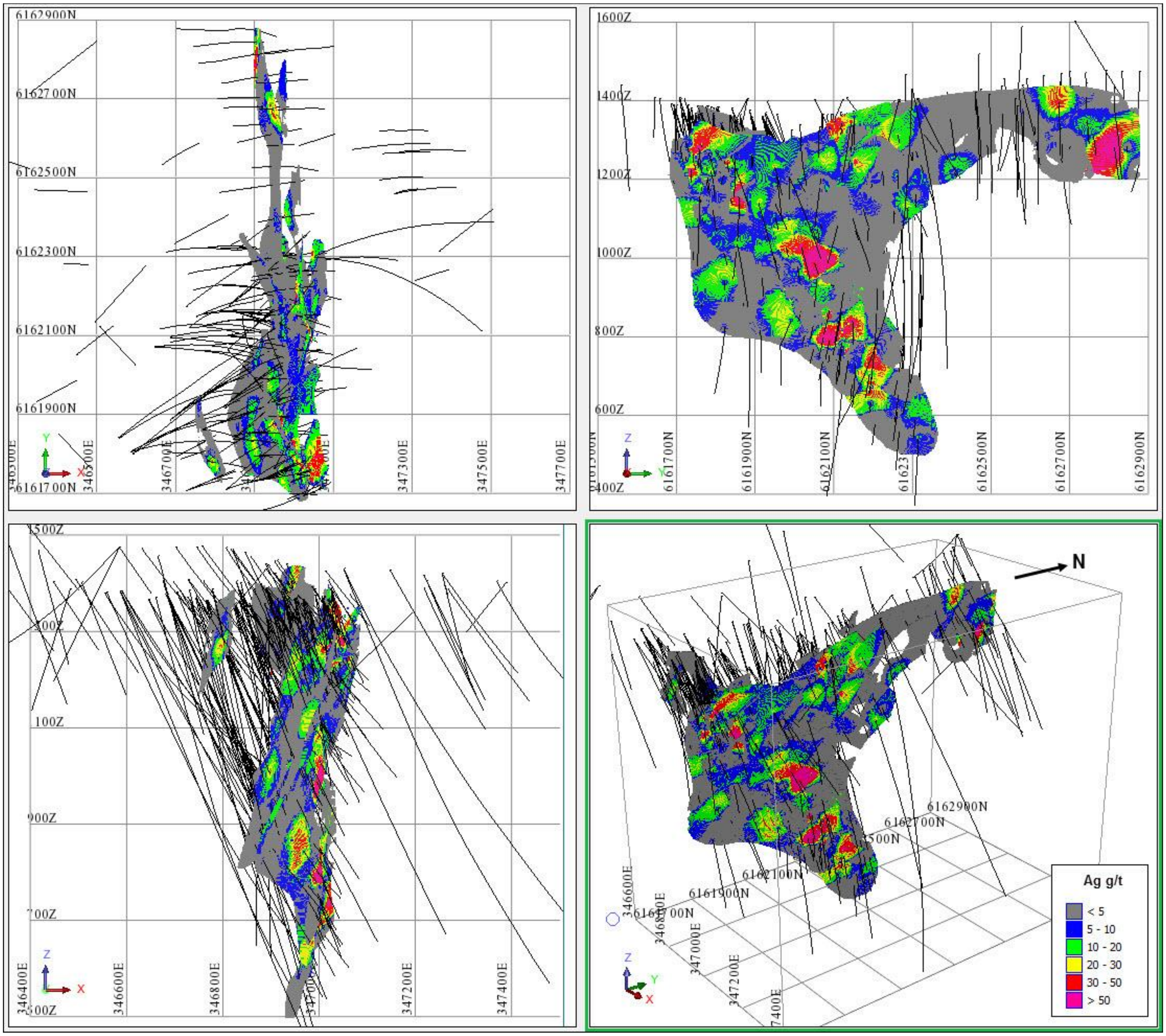
Block model grade distribution is illustrated in Figure 14-28 to Figure 14-30.

Figure 14-28: Block Model Gold Grades



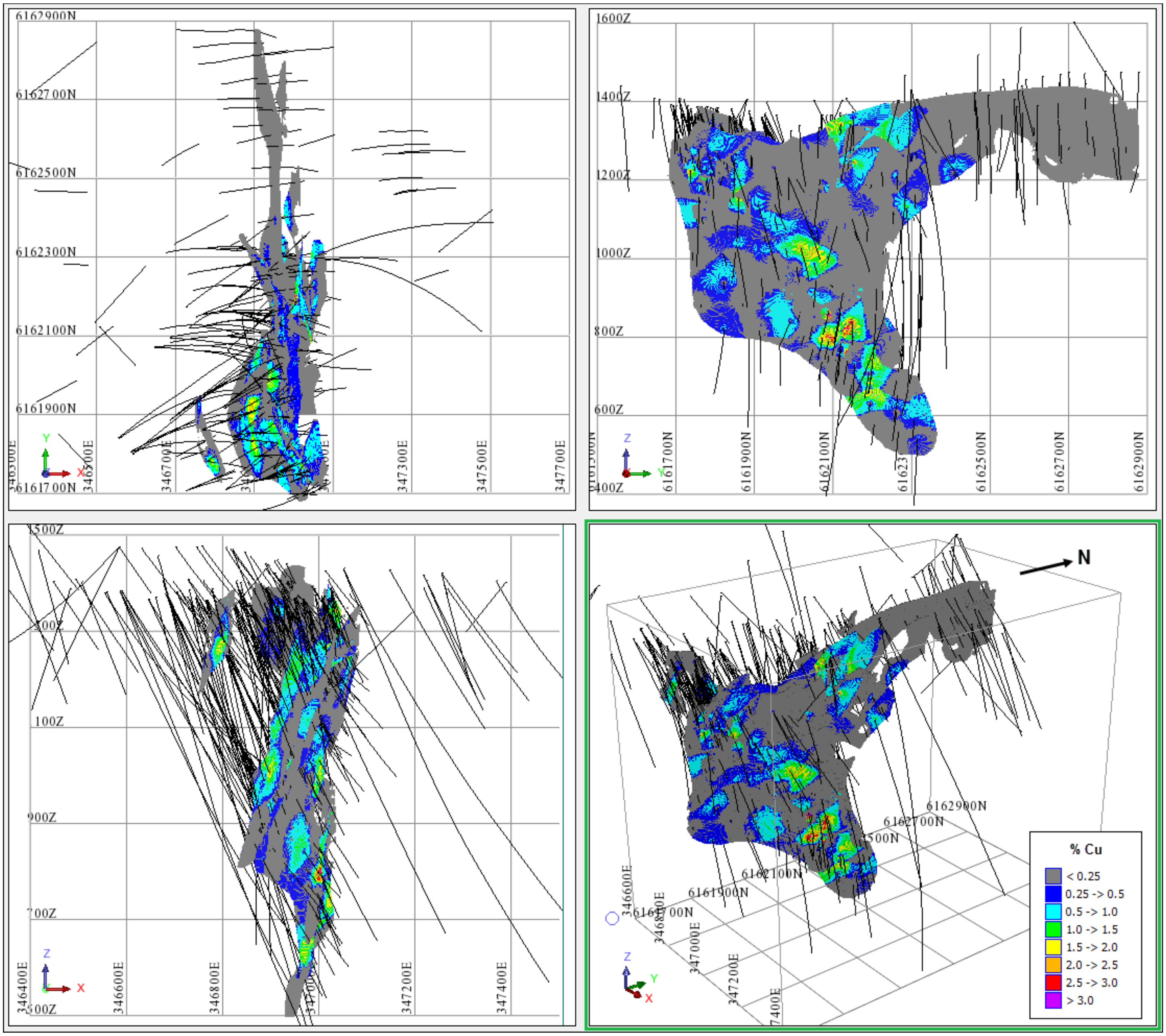
Source: Geosim, 2022.

Figure 14-29: Block Model Silver Grades



Source: Geosim, 2022.

Figure 14-30: Block Model Copper Grades

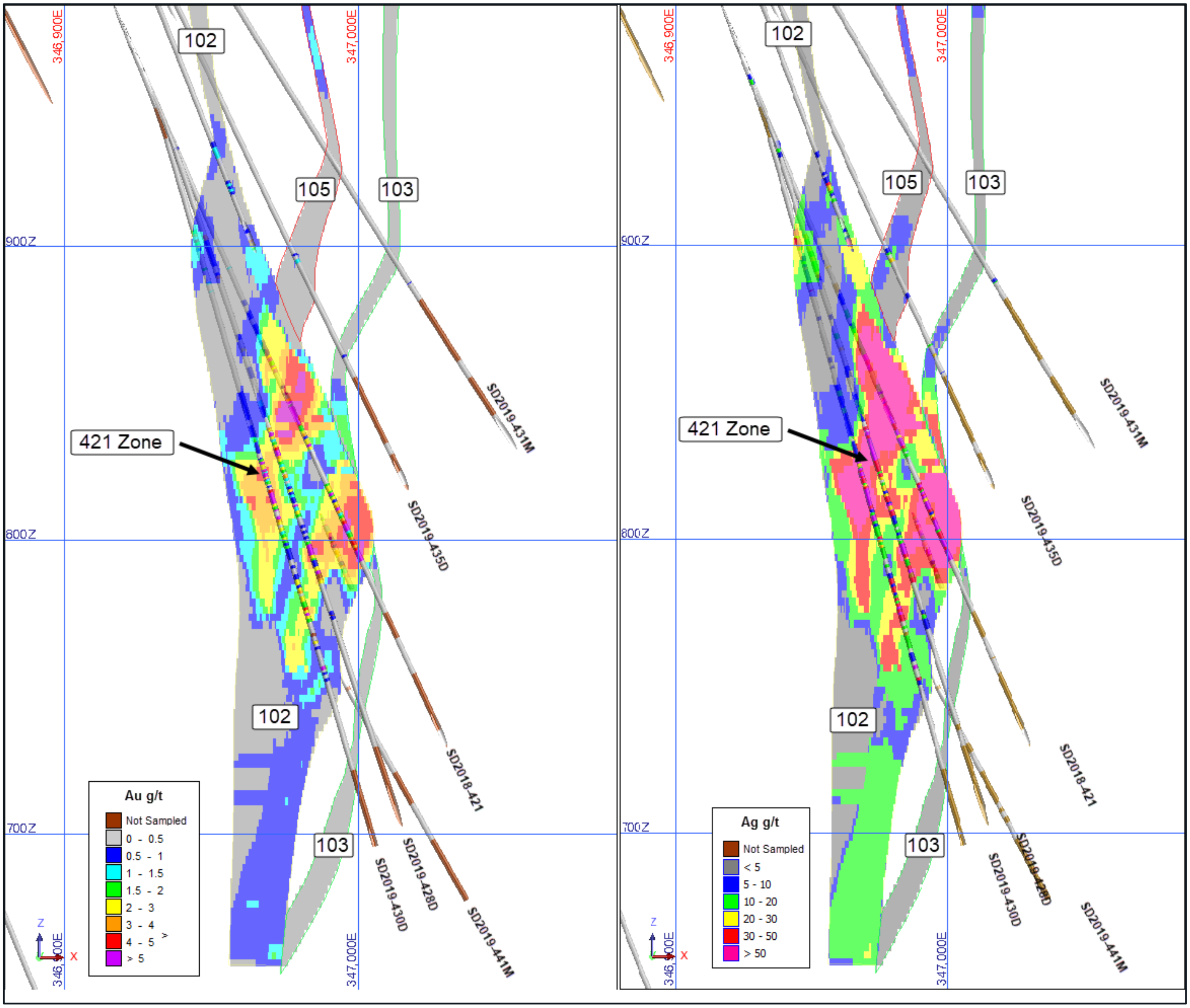


Source: Geosim, 2022.

Grade distribution in the 421 Zone area of Zone 102 is presented in Figure 14-31 and Figure 14-32 as cross sections at 6162125 North.

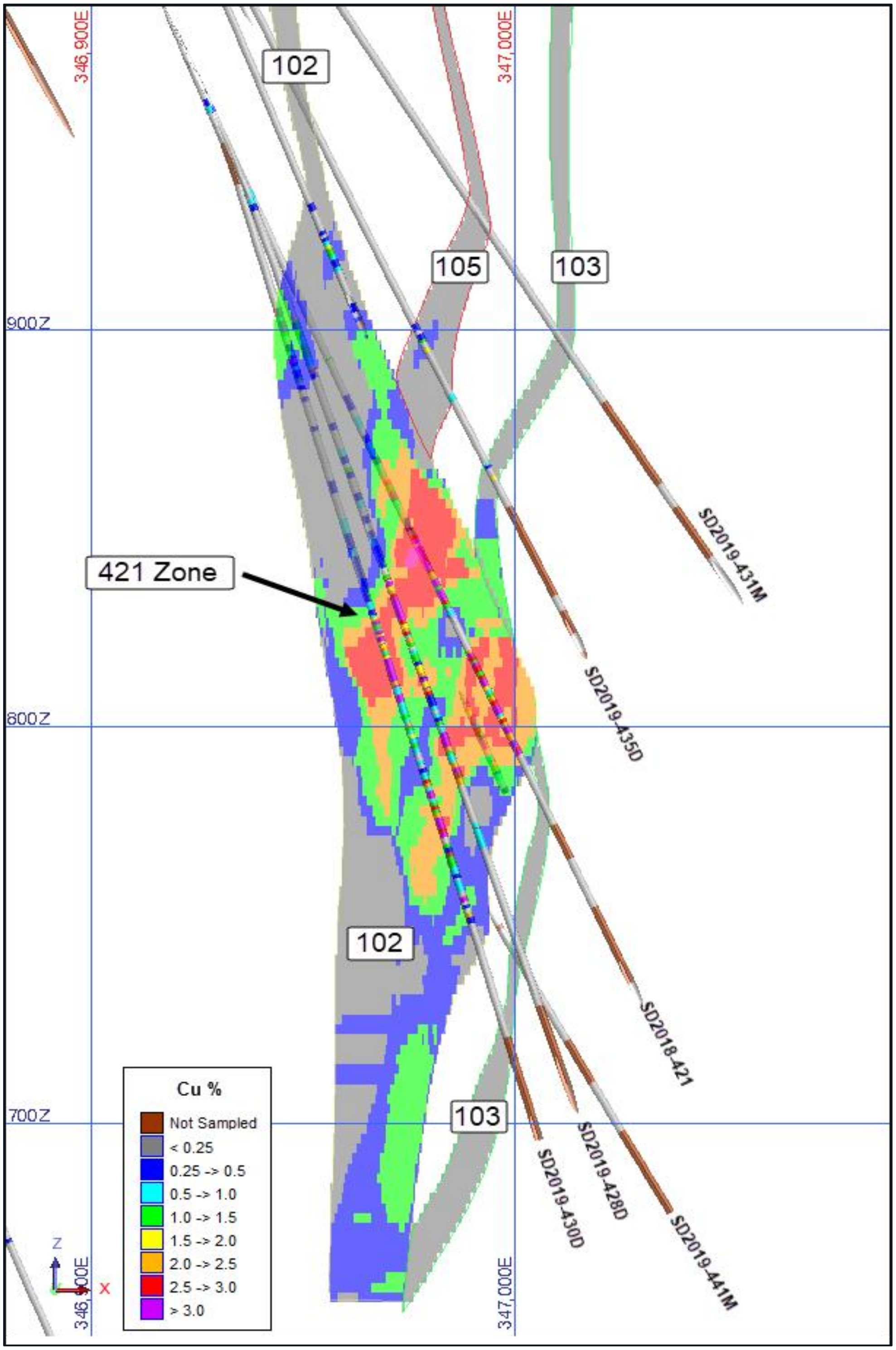
Grade distribution on Section 6162050 through Zones 102, 103, 105, and 106 is illustrated in Figure 14-33 and Figure 14-34

Figure 14-31: Section 6162125N - Au and Ag Grades in 421 Zone



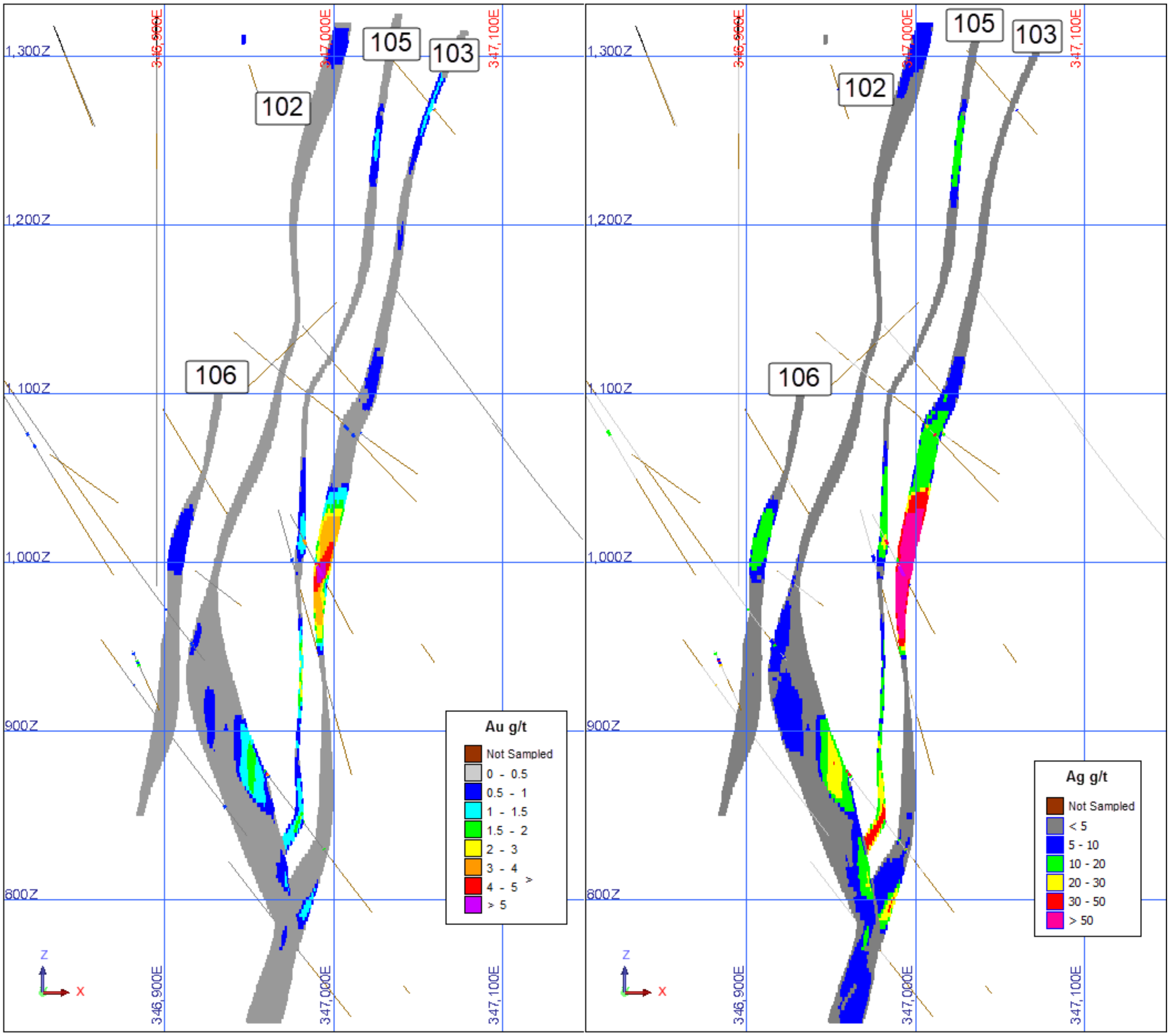
Source: Gesoim, 2022.

Figure 14-32: Section 6162125N - Cu Grades in 421 Zone



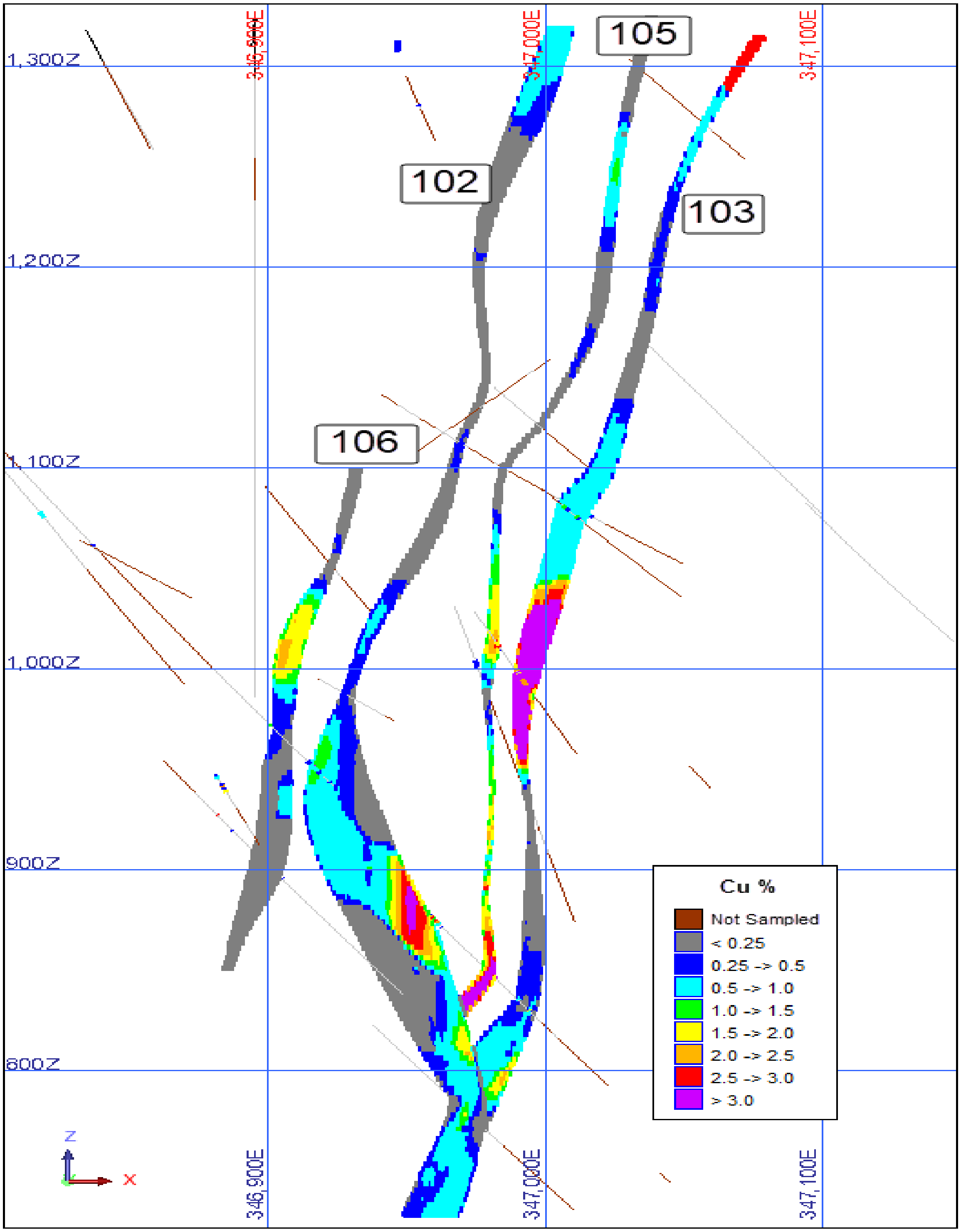
Source: Geosim, 2022.

Figure 14-33: Section 6162050N - Au and Ag Grades



Source: Geosim, 2022.

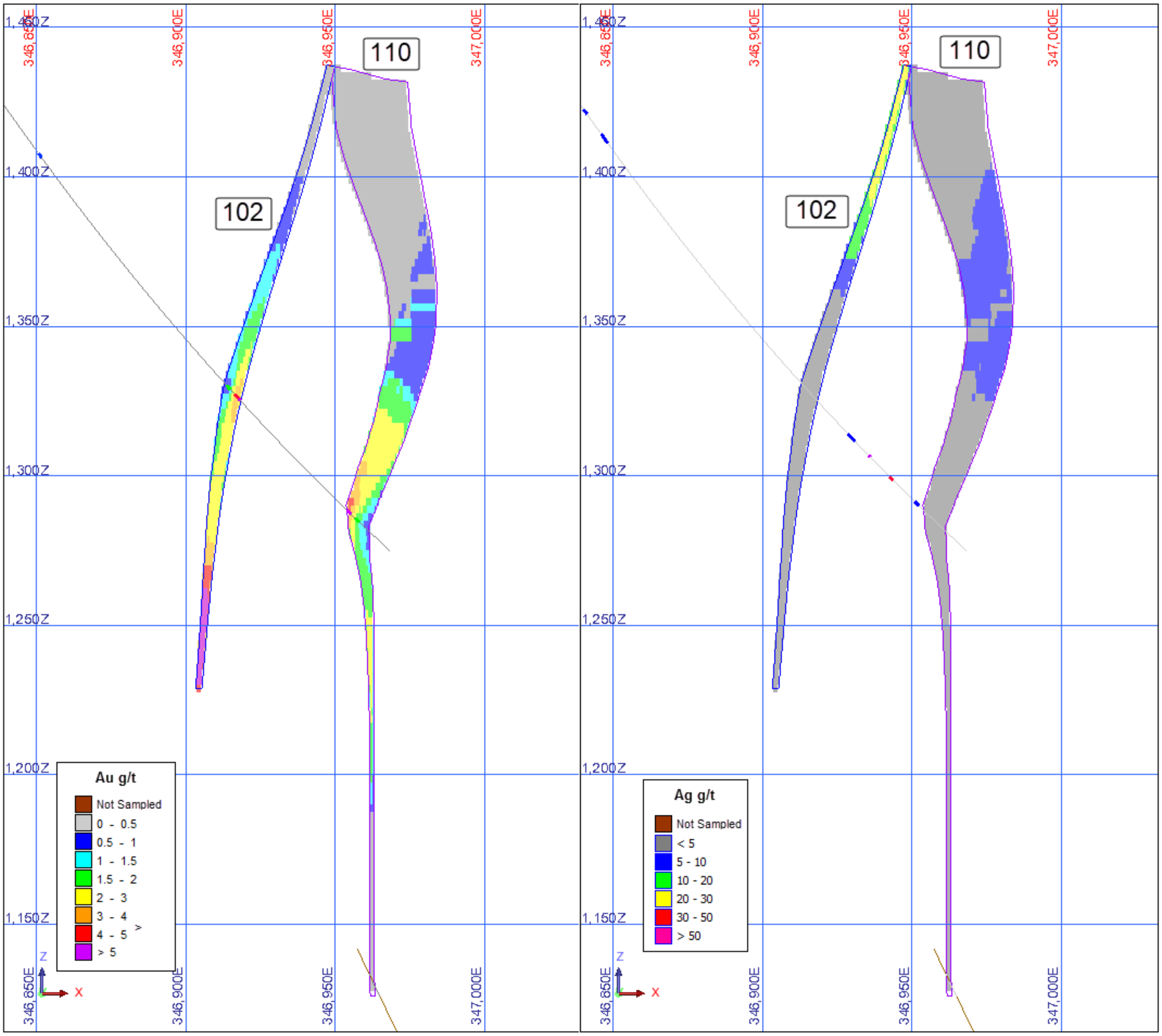
Figure 14-34: Section 6162050N - Cu Grades



Source: Geosim, 2022.

Grade distribution on Section 6162705 through Zones 102 and 110 is illustrated in Figure 14-35. This section contained no significant Cu grades (greater than 0.25%).

Figure 14-35: Section 6162705N - Au and Ag Grades



Source: Geosim, 2022.

14.3.8 Block Model Validation

Model verification was initially carried out by visual comparison of blocks and composite grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

A comparison of global mean values shows a reasonably close relationship with composites and block model values estimated using the nearest neighbour and ID3 interpolation methods (Table 14-28).

Table 14-28: Global Mean Grade Comparison

Item	Au	Ag	Cu
Composites	0.65	11.4	0.51
Capped Composites	0.59	10.7	0.50
Declustered Capped Composites	0.54	9.6	0.42
IDW Grade	0.52	9.2	0.41
NN Grade	0.57	9.9	0.45

14.3.9 Classification of Mineral Resources

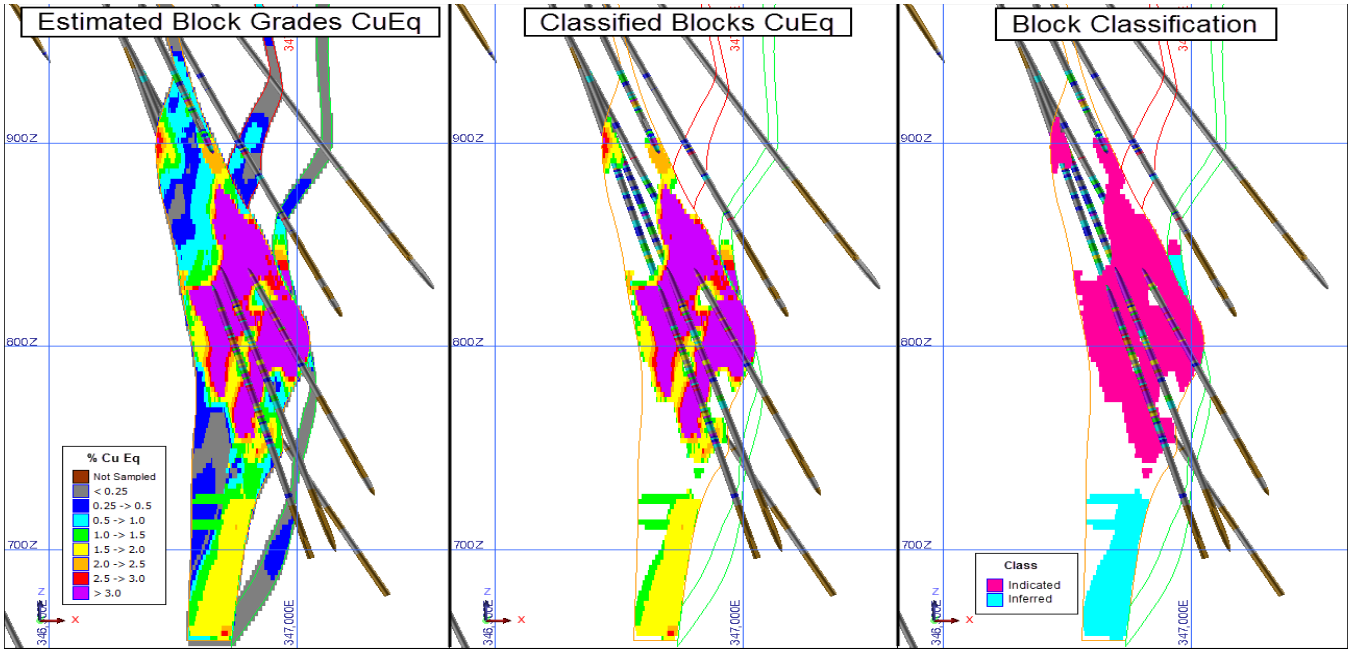
Resource classifications used in this study conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014).

Classified blocks were restricted to a minimum mining width of 2 m using a column processing procedure to differentiate potential economic material from waste. Blocks were further classified based on drill spacing. Blocks falling within a drill spacing of 30 m within Zones 2, 3, and 6 were initially assigned to the Indicated category. All other estimated blocks within a maximum search distance of 100 m were assigned to the Inferred category. Blocks were reclassified to eliminate isolated blocks and clusters and to eliminate small portions of Indicated resources within Inferred resources and vice versa.

Block classification is illustrated in Figure 14-36 to Figure 14-38.

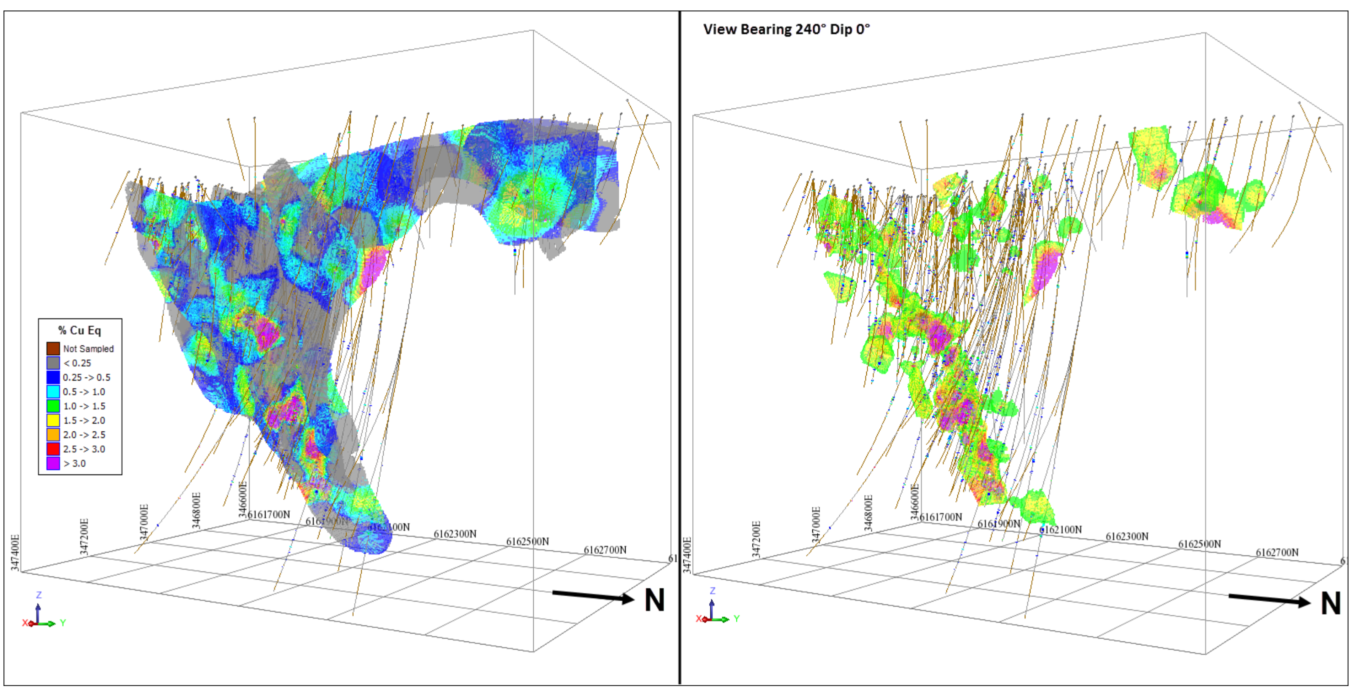
The formula for copper equivalent (CuEq) was based on assumed metal prices of $1,650/oz gold. $21.50/oz silver and $3.50/lb copper (CuEq = Cu + Au * 0.6875 + Ag * 0.009).

Figure 14-36: Section 6162125N Block Classification



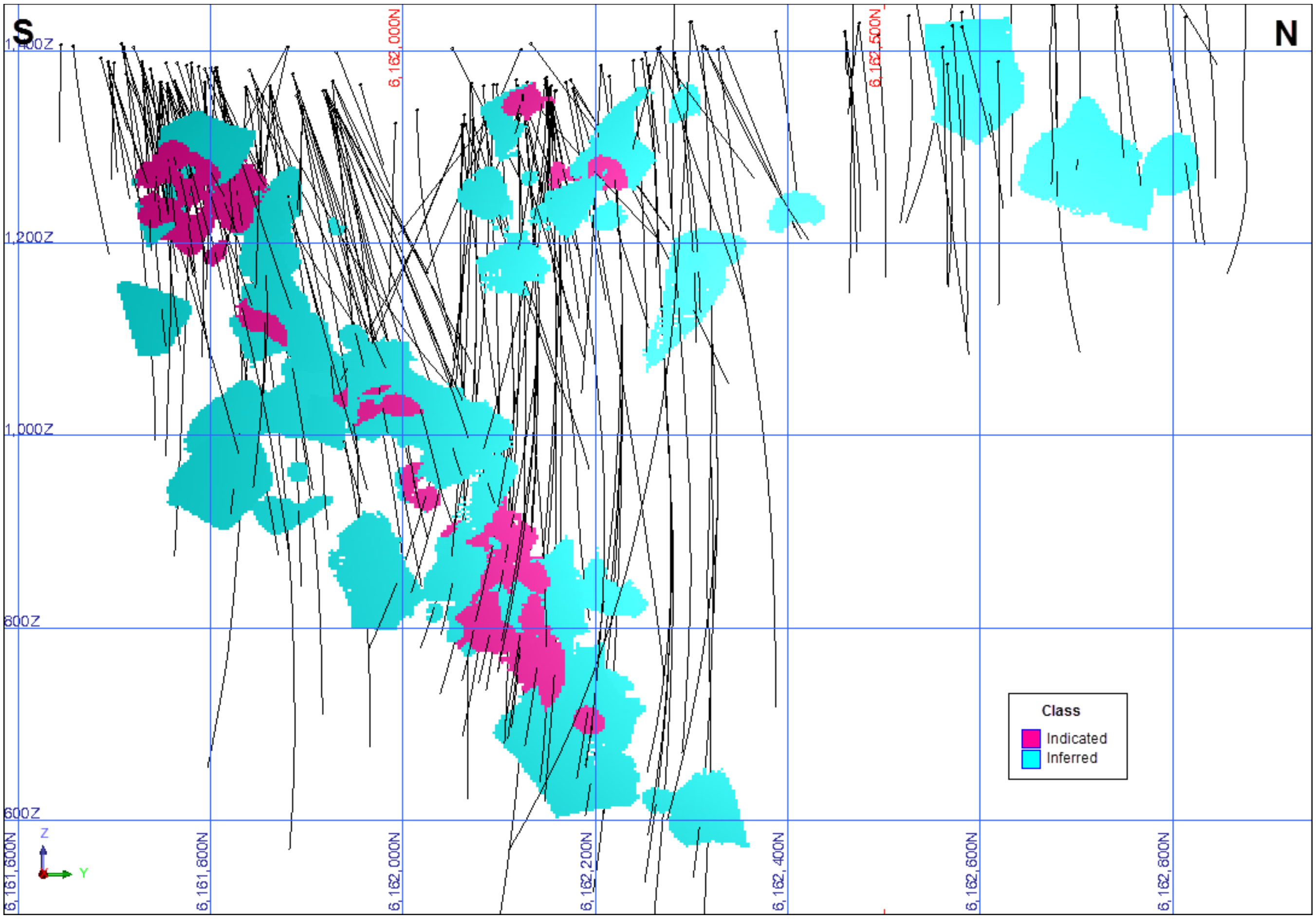
Source: Geosim, 2022.

Figure 14-37: Perspective View of Estimated and Classified Blocks



Source: Geosim, 2022.

Figure 14-38: Longitudinal Section Showing Block Classification



Source: Geosim, 2022.

14.3.10 Reasonable Prospects for Eventual Economic Extraction

Reasonable prospects for economic extraction were determined by applying a minimum mining width of 2 m and excluding isolated blocks and clusters of blocks that would likely not be mineable. The base case cut-off of US$88/t was determined based on metal prices of US $3.50/lb copper, US $1,650/oz gold and US $21.50/oz silver, underground mining cost of US $64/t, transportation cost of $6/t, processing cost of US $8.25/t, and G&A cost of US $9.75/t. Recoveries used in calculation of the base case cut-off were based on recent metallurgical test results. The recovery formulas were as follows:

- Cu Recovery: IF(Cu<0.1, 0.5, IF(Cu<1, 0.94712*Cu^0.0649, 0.95))
- Au Recovery: MIN(0.1*Au+0.66, 0.85)
- Ag Recovery: MIN(IF(Ag<0.5,0.3,0.51169+0.26032*LOG(Ag)),0.72)

Block tonnes were estimated using a density of 3.4 g/cm for mineralized material. Table 14-29 shows the assumptions used in the cut-off determination.

Table 14-29: Cost Assumptions Used in Cut-off Determination

Assumptions	Value
Gold Price (US$ per oz)	$1,650
Silver Price (US$ per oz)	$21.50
Copper Price (US$/lb)	$3.50
Gold Recovery	Variable based on grade – Max 85%
Silver Recovery	Variable based on grade – Max 72%
Copper Recovery	Variable based on grade – Max 95%
Underground Mining Cost (US$ per tonne milled)	$64
Transportation	$6
Processing (US$ per tonne milled)	$8.25
G&A Cost (US$ per tonne milled)	$9.75
Total Operating Cost (US$ per tonne milled)	$88
Cut-off Grade (US$/t)	$88

Mineral resources were estimated for gold, silver, and copper. Significant grades of zinc have been encountered but the distribution is highly irregular and would not likely justify the additional cost of extraction.

14.3.11 Mineral Resource Statement

The updated Stardust mineral resource estimate for the CCS Zone is presented in Table 14-30. It is based on a cut-off of US $88/tonne and 2-m minimum mining width.

Table 14-30: Mineral Resource Statement - Stardust CCS Zone

Underground	Economic Cut-off US$	Classification	Tonnes (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal		
							Cu (Mlbs)	Au (koz)	Ag (koz)
	88.00	Indicated	1.6	1.49	1.63	30.1	52.2	83.1	1,536.4
		Inferred	4.1	1.00	1.38	22.8	90.0	181.1	3,004.3

Notes:

1. The Mineral Resources have been compiled by Mr. B Ronald G. Simpson of GeoSim Services Inc. Mr. Simpson has sufficient experience that is relevant to the style of mineralization and type of deposit under consideration and to the activity that he has undertaken to qualify as a Qualified Person as defined by NI 43-101.
2. Mineral Resources are estimated consistent with CIM Definition Standards and reported in accordance with NI 43-101.
3. Reasonable prospects for economic extraction were determined by applying a minimum mining width of 2.0 m. and excluding isolated blocks and clusters of blocks that would likely not be mineable.
4. The base case cut-off of US$88/t was determined based on metal prices of $1,650/oz gold. $21.50/oz silver and $3.50/lb copper, underground mining cost of US$64/t, transportation cost of US$6/t, processing cost of US$8.25/t, and G&A cost of US$9.75/t. Recovery formulas were based on recent metallurgical test results. Maximum recoveries were limited to 95% for Cu, 85% for Au and 72% for Ag.
5. Block tonnes were estimated using a density of 3.4 g/cm^3 for mineralized material.
6. Six separate mineral domains models were used to constrain the estimate. Minimum width used for the wireframe models was 1.5 m.
7. For grade estimation, 2.0-metre composites were created within the zone boundaries using the best-fit method.
8. Capping values on composites were used to limit the impact of outliers. For Zone 102, gold was capped at 15 g/t, silver at 140 g/t and copper at 7.5%. For all other zones, gold was capped at 6 g/t, silver at 140 g/t and copper at 5%.
9. Grades were estimated using the inverse distance cubed method. Dynamic anisotropy was applied using trend surfaces from the vein models. A minimum of 3 and maximum of 12 composites were required for block grade estimation.

10. Blocks were classified based on drill spacing. Blocks falling within a drill spacing of 30 m within Zones 2, 3, and 6 were initially assigned to the Indicated category. All other estimated blocks within a maximum search distance of 100 m were assigned to the Inferred category. Blocks were reclassified to eliminate isolated Indicated resources within inferred resources.
11. The quantity and grade of reported Inferred Mineral Resources in the 2023 PEA are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral Resources as Indicated or However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
12. The estimate of Mineral Resources may be materially affected by geology, environment, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
13. Totals may not sum due to rounding.

The mineral resource breakdown by zone is presented in Table 14-31 and Table 14-32.

Table 14-31: Indicated Mineral Resources by Zone

Zone	Tonnes (000)	Grades			
		%Cu	g/t Au	g/t Ag	%CuEq
102	1,218	1.57	1.65	28.3	3.14
103	253	1.23	1.90	43.9	3.17
106	82	1.14	1.04	12.9	2.07
110	35	1.41	0.53	33.9	2.12
Total	**1,587**	**1.49**	**1.63**	**30.1**	**3.07**

* Totals may not sum due to rounding.

Table 14-32: Inferred Mineral Resources by Zone

Zone	Tonnes (000)	Grades			
		%Cu	g/t Au	g/t Ag	%CuEq
102	1,914	0.88	1.44	19.7	2.20
103	992	1.06	1.66	38.1	2.70
105	163	0.88	1.34	23.5	2.13
106	795	1.32	0.86	13.6	2.14
110	180	0.54	1.54	12.9	1.83
111	47	1.26	1.31	21.1	2.46
Total	**4,090**	**1.00**	**1.38**	**22.8**	**2.30**

* Totals may not sum due to rounding.

14.3.12 Factors That May Affect the Mineral Resource Estimate

The mineral resource estimate is based on limited information and sampling gathered through appropriate techniques diamond drill core holes. The estimate was prepared using industry standard techniques and has been validated for bias and acceptable grade-tonnage characteristics.

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price falling below the assumed price
- Assumptions that all required permits will be forthcoming
- Actual metallurgical recoveries being lower than assumed recoveries in the resource estimate
- Significant increase in mining and process cost than the current assumptions
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the province of British Columbia in terms of environmental, permitting, taxation, socio-economic, marketing, and political. GeoSim is not aware of any legal or title issues that would materially affect the Mineral Resource estimate.

14.3.13 Comments on the Stardust Mineral Resource Estimate

Mr. Simpson has estimated and classified the mineral resources in a manner consistent with the 2014 CIM Definition Standards. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that most of the Inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

15 MINERAL RESERVE ESTIMATES

This section is not relevant to this report.

16 MINING METHODS

16.1 Introduction

Extraction of the economic mineable inventory is proposed through a combination of open pit and underground mining as this presented the best case from an economic, environmental, and sustainable development perspective. A combination of four different mines will be developed to achieve a production rate of 22,000 t/d for the Kwanika-Stardust project, including:

- Kwanika Central Open Pit: This mine will produce 29.4 Mt mill feed with an average NSR value of 36.67 $/t and strip ratio of 1.87. This mine will be the only source of mill throughput in the production year of 1 to 3, and also the majority of mill feed in year 4.
- Kwanika South Open Pit: This mine will produce 19.1 Mt mill feed with an average NSR value of 23.31 $/t and strip ratio of 1.66. This mine will supplement the Kwanika Block Cave mill feed capacity in production years 9 to 12.
- Kwanika Central Block Cave: This mine will produce 44 Mt mill feed with an average NSR value of 56.79 $/t. This mine will utilize the block cave mining method, with operating costs estimated at 10.62 $/t inclusive of mining, transportation, and G&A. This mine will commence production near the end of year 3 of the project and will be the predominant feed source for years 4 through 9, with a steady state throughput of 20,000 t/d years 5 through 8.
- Stardust Underground Mine: This mine will produce 3.1 Mt mill feed with an average NSR of 195.41 $/t. The mine will utilize sub level open stoping with an estimated operating cost of 111.33 $/t inclusive of mining, transportation, and G&A. The mine will start construction in year 3 and commence production in year 4. Production will finish in year 9 of the project, with an annual throughput of 1,590 t/d.

16.2 Underground Geotechnical Considerations

16.2.1 Kwanika

The Kwanika geotechnical database was primarily developed by Moose Mountain Technical Services (MMTS) as part on an open pit geotechnical advancement project in 2018. Seventeen (17) holes were geotechnically logged. Core sampling was conducted to support a rock property testing program. Alpha/beta measurements were taken on the oriented core supplemented by geotechnical mapping of select/representative rock outcrops.

Customized geotechnical logging guidance provided for assessment of requisite parameters for the NGI Q, Bieniawski RMR (1989) and Laubscher Rock Mass Rating (1990) rock mass classification systems.

The Kwanika geotechnical dataset is considered adequate for conceptual PEA-level designs for both open pit and underground.

16.2.1.1 Geotechnical Data Sources

Section 10.2.1 outlines the historical drilling database for the Kwanika deposit. The 2018 Geotechnical data advancement project included seventeen holes (17) with valuable geotechnical information. In total, 7300.9 m of core was logged for holes K18-179A to K18-195. The geotechnical parameters logged are summarized in Table 16-1. Structural information was also collected on oriented core traverses (alpha/beta angles) for faults, shears, veins, dykes, lower contacts, and upper contacts.

Table 16-1: Kwanika Core Logging Parameters Collected (MMTS 2018)

Geotech Core Logging Parameter	Factors						
RQD%	90-100%	75-90%	50-75%	25-50%	<25%		
	20	17	13	8	5		
Intact Rock Strength (MPa)	R6	R5	R4	R3	R2	R1	R0
	>250	100 - 250	50 -100	25 - 50	5 - 25	1 - 5	<1
Weathering	W1	W2	W3	W4	W5	W6	
	Fresh	Slightly Weathered	Moderately Weathered	Highly Weathered	Completely Weathered	Original rock fabric destroyed	
Joint Surface Roughness	RO	SM	SS				
	Roughness	Smooth	Slickensided				
Joint Surface Shape	PL	CU	UN	ST	IR		
	Planar	Curved	Undulating	Stepped	Irregular		
Core Breakage	Excellent	Good	Fair	Poor	Very Poor		
	Intact Rock	Massive, moderately jointed	Blocky and seamy	Shattered, very blocky and seamy	Crushed		

16.2.1.2 In-situ Stress State

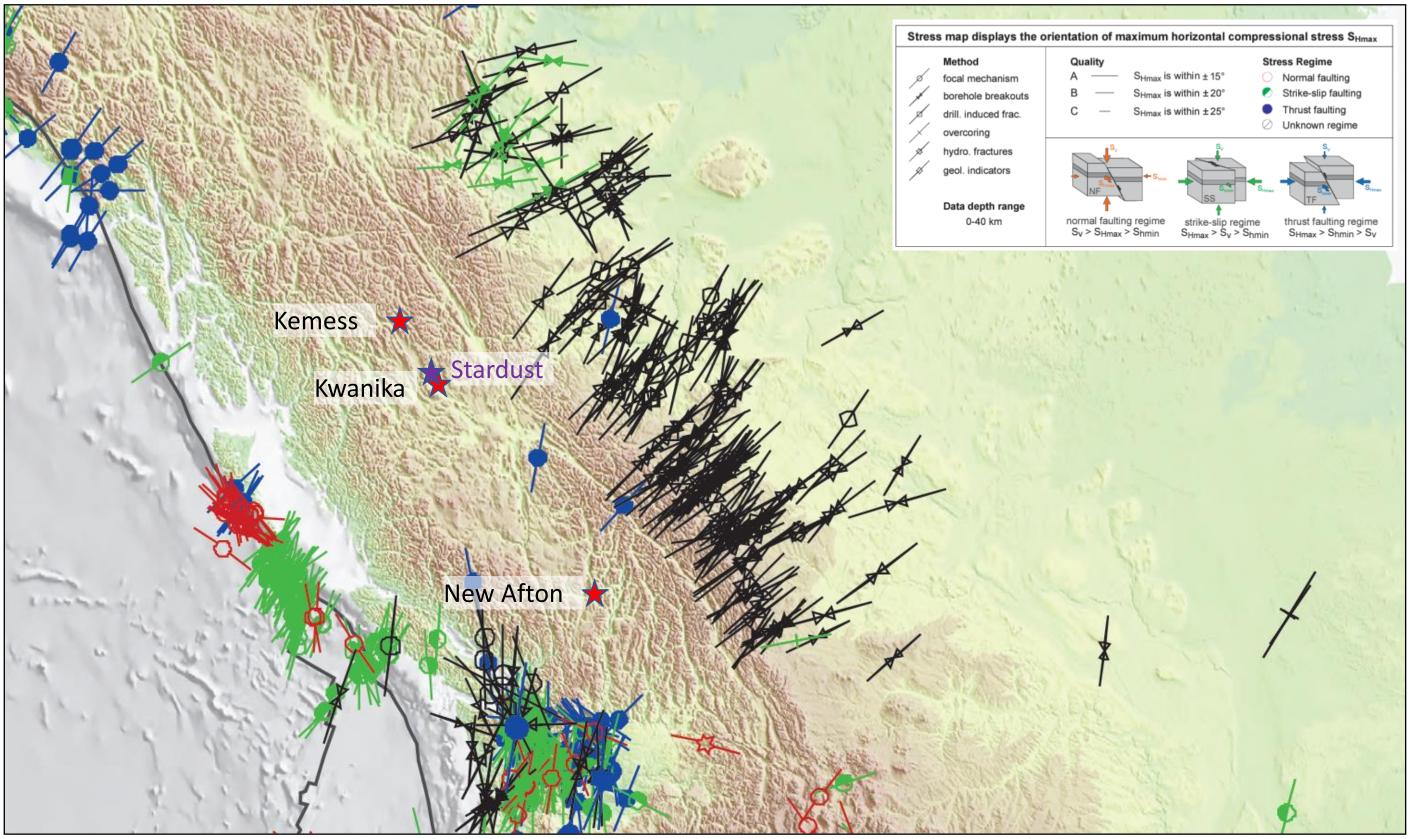
The state of in-situ stresses includes both the lithostatic stress and those from geological processes, including regional tectonics and glaciation which increased the vertical stress.

There have not been any in-situ stress measurements conducted on the property. Experience and analogous projects in the region have been used in the absence of site-specific data.

Three (3) non-local sources have been proposed as shown in Table 16-2. The relative proximity of the two (2) mine locations – Kemess underground and New Afton underground – are illustrated in Figure 16-1. It is recommended that the Kemess underground in-situ stress state parameters be considered until local measurements are obtained. Additionally, the noted uncertainty in the indicated primary and secondary horizontal stress direction suggests that a sensitivity analysis is valuable for any studies where the in-situ stress state is an important input (i.e., excavation/ stope damage and subsidence studies). Where the Central, North and Pinchi faults are local to areas of interest, their presence is likely to impact the in-situ stress field and should be considered for future studies.

The Stardust underground is currently designed to a depth of 775 mbgs while the Kwanika Block Cave development is estimated to be ~ 550 mbgs At these depths, overstressing of the rockmass is unlikely; however there is a potential for overstressing of the rockmass and a high-level review of the induced stresses from mining would be beneficial to provide greater confidence in the development and production scheduling and costing.

Figure 16-1: Comparison of In-situ Stress State Estimate Locations – Kwanika-Stardust Project relative to 2016 World Stress Maps.



Source: World Stress Map, 2016

Table 16-2: In-situ Stress Estimation – Kwanika Project

Western Canadian Basin	Kemess	New Afton
s_H (MPa) = 1.2*s_V	s_H (MPa) = 1.26*s_V	s_H (MPa) = 0.0648 MPa/m * z
s_h (MPa) = s_V	s_h (MPa) = 0.66*s_V	s_h (MPa) = 0.0486 MPa/m * z
s_V (MPa) = 0.026 MPa/m * z	s_V (MPa) = 0.026 MPa/m * z	s_V (MPa) = 0.0284 MPa/m * z
Orientation s_H: N/A	Orientation s_H: North/South (flat)	Orientation s_H: East/West (flat)
Orientation s_h: N/A	Orientation s_h: East/West (flat)	Orientation s_h: North/South (flat)

16.2.1.3 Structural Model

Information from the 2018 drilling campaign is the source of the current structural information. Stratavision Pty Limited analyzed the raw data to determine dominant orientations for each geotechnically logged hole. The data indicates that several orientations are present with a notable trend related to a shallow west-dipping unconformity. Low angle structures dipping 20 to 40°W to West/Northwest are present in most drillholes. These structures include fractures, joints and minor shears which are expected to impact cave propagation behaviour and local ground stability in the underground development.

Moderately dipping Southwest and Southeast structures may represent a conjugate structural fabric developed within the mineralized corridor. Steep structures, particularly faults, are less prominent in the data which may be due to the steepness of the drilling.

Three (3) major faults have been identified on the property:

1. Pinchi fault

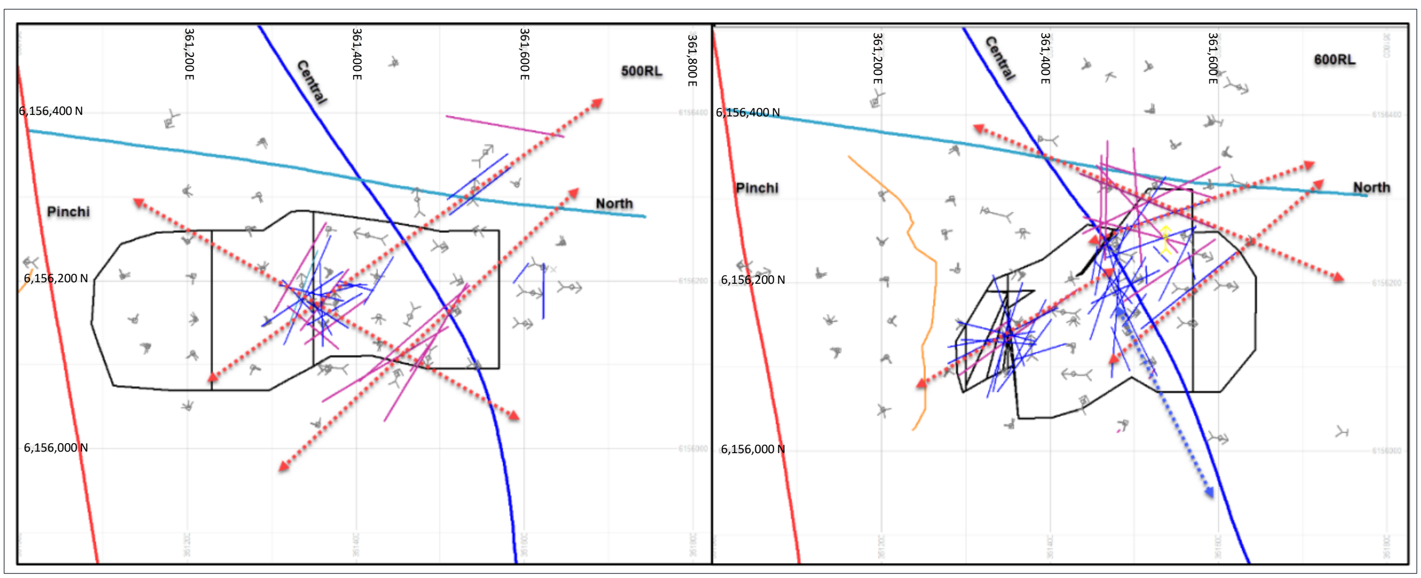
2. North fault (historically referenced as the South fault)

3. Central fault (previously known as the WW fault

Refer to Section 7.2.1 for a detailed description of these primary faults. A general sense of their orientation can be referenced in Figure 7-2. Major regional and local scale faults are interpreted to be steeply-dipping late-stage structures that strike in a general North-South to Northwest/Southeast direction. By contrast, the majority of the smaller shears appear to be influenced by the local stratigraphy and strike in a Northeast / Southwest direction and dip moderately towards the Northwest. Note that these interpretations are based on limited information and require verification once underground access is available. Figure 16-2 shows plan view traces of the major faults and minor shear interpretations while Figure 16-3 illustrates the minor shear traces in section view.

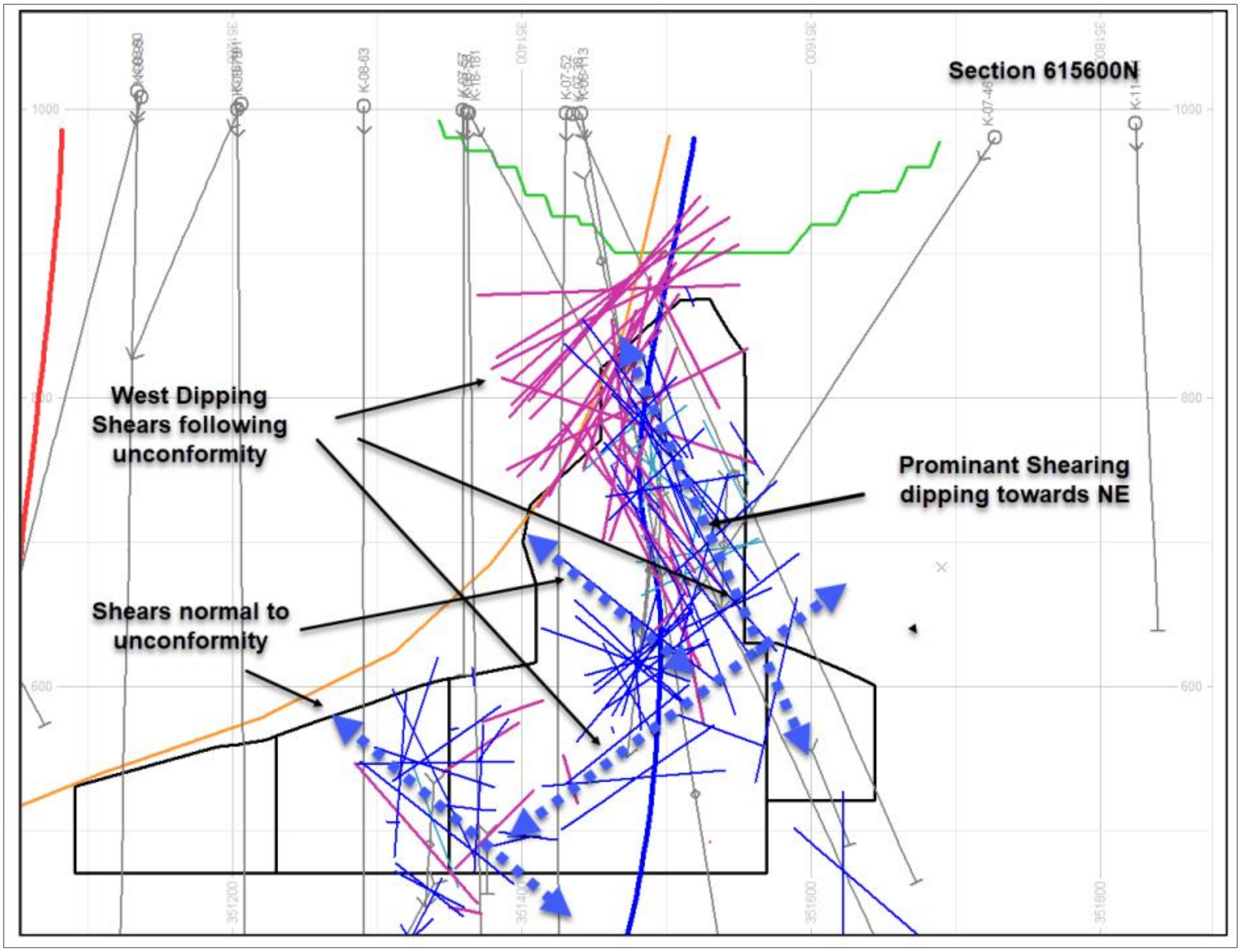
A summary of the dominant structural orientation is summarized below in Table 16-3 and as a stereograph projection in Figure 16-4.

Figure 16-2: Traces of Major Faults and Minor Shears (500RL and 600RL) Plan View – Scale as shown Kwanika.



Source: Kwanika Prefeasibility Study – Wrap-Up Report – WP5 Open Pit Geotechnical, November 2019.

Figure 16-3: Shear Traces on 6,156,000N (View North, Scale as shown)

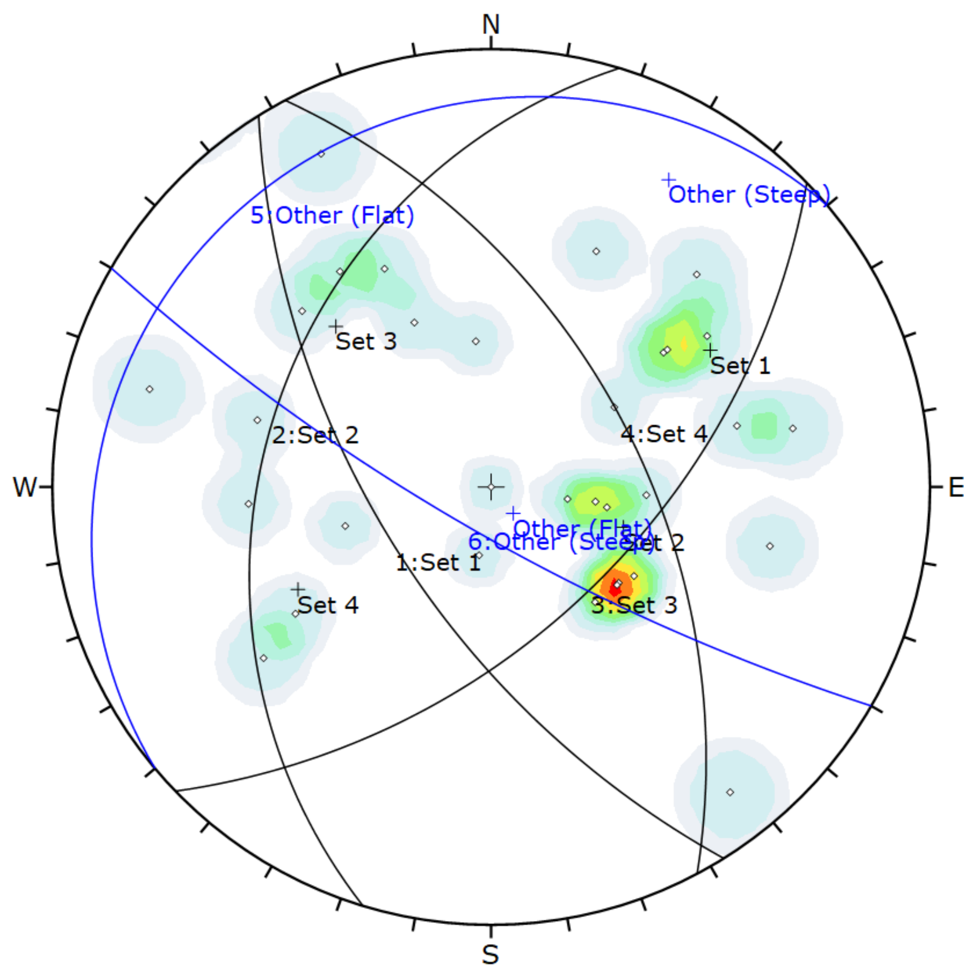


Source: Kwanika Prefeasibility Study – Wrap-Up Report – WP5 Open Pit Geotechnical, November 2019.

Table 16-3: Summary of Structural Mapping Data - Kwanika

Set ID	Dip (°)	Dip Direction (°)	Strike (°)
Set 1	**35** (*20 to 42*)	**287** (*237 to 318*)	**197** (*147 to 228*)
Set 2	**61** (53 *to 68*)	**238** (*204 to 282*)	**148** (*114 to 192*)
Set 3	**54** (37 to 62)	**136** (*86 - 174*)	**46** (*356 to 84*)
Set 4	**53** (38 to 66)	**62** (*53 - 75*)	**332** (*323 to 345*)
Other (Flat)	**9** (0 to 18)	**320** (*270 to 10*)	**230** (*180 to 280*)
Other (Steep)	**78** (70 to 83)	**210** (*106 to 322*)	**120** (16 *to 232*)

Figure 16-4: Stereonet Projection of Kwanika Dominant Structural Orientations (Rocscience Dips v3.0)



	Color	Dip	Dip Direction	Label
User Planes				
1	Black	61	238	Set 1
2	Black	35	287	Set 2
3	Black	54	136	Set 3
4	Black	53	62	Set 4
5	Blue	9	320	Other (Flat)
6	Blue	78	210	Other (Steep)

Source: Kwanika Prefeasibility Study – Wrap-Up Report – WP5 Open Pit Geotechnical, November 2019.

16.2.1.4 Rock Property Data

A rock mechanics laboratory testing program was completed as part of the 2018 MMTS geotechnical study. Unconfined compressive strength (UCS), Triaxial Strength, indirect tensile strength, and direct shear tests were completed on a relatively small number of rock samples. Table 16-4 provides a summary of the test results.

Table 16-4: Summary of Rock Material Testing – Kwanika.

Rock Type	Bulk Density	UCS	Young Modulus, E	Poisson Ratio	Indirect Tensile Strength	Peak Friction Angle	Residual Friction Angle
	(kg/m³)	(MPa)	(GPa)	-	(MPa)	(°)	(°)
Monzonite	2.77	55.22	37.31	0.18	7.25	41.60	40.10
Diorite	2.79	60.45	35.70	0.28	15.66	35.50	34.70
Microdiorite	2.77	32.10	34.30	0.22	-	-	-
Sandstone	2.63	13.60	10.40	0.21	-	-	-
Monzodiorite	2.67	32.55	58.10	0.25	-	-	-
Conglomerate	2.66	73.30	23.60	0.30	8.09	39.10	35.45
	No. of Data Points						
Monzonite	9	9	9	9	5	3	3
Diorite	2	2	2	2	1	1	1
Microdiorite	1	1	1	1	0	-	-
Sandstone	1	1	1	1	0	-	-
Monzodiorite	2	2	1	1	0	-	-
Conglomerate	1	1	1	1	2	2	2
	Standard Deviation						
Monzonite	0.06	22.36	17.43	0.09	3.62	2.81	4.51
Diorite	0.02	44.05	36.20	0.07	-	-	-
Microdiorite	-	-	-	-	-	-	-
Sandstone	-	-	-	-	-	-	-
Monzodiorite	0.05	31.61	-	-	-	-	-
Conglomerate	-	-	-	-	-	0.99	3.89

Source: Rock Strength Laboratory Testing Program for 2018 Geotechnical Investigation of Serengeti Resources Kwanika Project, Jan 2019

The lab test results indicate significant variability in the Unconfined Compressive Strength (UCS) which is attributed to the presence of intense sericite/clay alteration that overprints the mineralized zones. When the UCS data are reviewed spatially against the planned location of the Kwanika underground development, the range of UCS values is between 40 MPa to 60 MPa with potentially higher values (~100 MPa) to the east and south of the deposit.

16.2.1.5 Rock Mass Classification

In advance of the 2019 geotechnical drilling program, MMTS assisted with developing a rock mass classification data collection system for the Kwanika project. The Rock Mass Rating system, RMR (Bieniawski, 1989), NGI Q (Barton et al. 1974, 2017) and Laubscher Mining Rock Mass Rating (MRMR, 1990) were selected for application. The data collected were supplemented with limited mapping of sparse outcrops to verify and complement the existing dataset, including lithology, rock mass strength and discontinuity characteristics. Table 16-5 and Table 16-6 shows a summary of the data collection which is derived from the original raw data logs.

Table 16-5: Kwanika Rock Mass Classification Data Summary (NGI Q, RQD, RMR_{89})

Domain	RQD LQ(Avg)UQ	Q' LQ(Avg)UQ	RMR_{89} LQ(Avg)UQ
Intrusives	87 (89) 99	1.5 (4.00) 5.08	43 (49) 58
Alteration	82 (87) 98	1.60 (3.84) 4.60	39 (48) 58
Mudstone	64 (77) 93	1.10 (3.08 2.25	34 (42) 53
Conglomerate	88 (90) 100	1.40 (2.80) 3.25	44 (50 (61)
Takla	78 (87) 99	1.68 (4.43) 4.33	46 (50) 56
Overburden	N/A	N/A	N/A

Table 16-6: Kwanika Rock Mass Classification Data Summary (Laubscher MRMR 1990)

Domain	FFM *LQ(Avg)UQ*	IRS *LQ(Avg)UQ*	JC_F *LQ(Avg)UQ*	RMRL90 *LQ(Avg)UQ*
Intrusives	2.70 (5.42) 6.30	60 (73) 100	12 (18) 24	36 (46) 50
Alteration	2.70 (5.51) 6.30	60 (65) 100	12 (18) 24	34 (42) 49
Mudstone	6.50 (9.60) 11.30	5 (49) 60	10 (15) 24	24 (34) 46
Conglomerate	6.90 (9.59) 10.78	60 (68) 70	12 (19) 26	30 (38) 45
Takla	4.60 (6.96) 7.78	60 (74) 100	12 (19) 24	35 (42) 49
Overburden	N/A	N/A	N/A	N/A

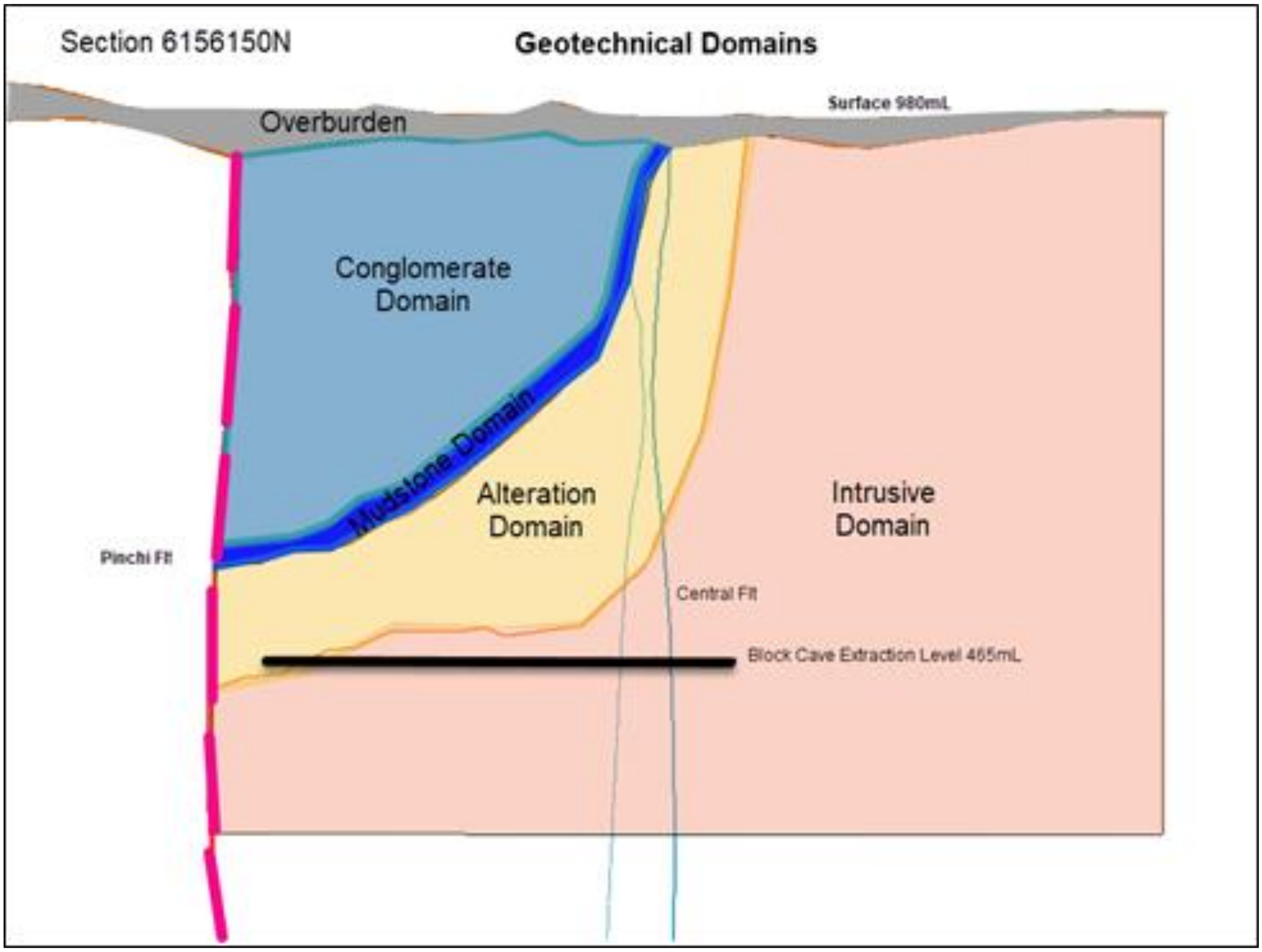
16.2.1.6 Geotechnical Domains

Preliminary geotechnical domains have been developed to capture macro level changes in the rock mass characteristics. This is captured in Figure 16-5. The most notable feature is an alteration domain which comprises relatively weak altered intrusives that follow the unconformity along the mudstone contact. Additional work is required to extend the domain interpretations to encompass the larger open pit development areas. The indicative geotechnical conditions expected within each domain are summarized below in Table 16-7.

Table 16-7: Geotechnical Domains and Faults (Kwanika)

Domain	Comments
Near Surface till + oxidation	Generally 25 m to 35 m in thickness.
Sediments:	Early Cretaceous Sediments unconformably overlies mineralization towards the west.
Conglomerates & Sandstones	Conglomerates and massive sandstone units have been grouped for geotechnical evaluation.
Mudstone (Siltstones)	A dark sometimes highly fissile/fractured siltstone located immediately above unconformity.
Altered Intrusives	Bulk of mineralization subject to supergene and sericite alteration from the Hypogene zone. (Typically Monzonite host)
Unaltered Intrusives	Lower grade and unmineralized intrusives (Monzodiorites, Diorites)
Takla Unit	Andesitic rocks, fine-grained flows and tuffs located on the northern side of the deposit
Eastern Block (East of Central Fault)	Notably stronger rock mass east of Central Fault comprising of Monzodiorites
Faults	**Comments**
Pinchi Fault	Major Regional Fault striking N-S, truncates mineralization along the western boundary at depth.
Central Fault	Regional-scale N-northwest trending fault near eastern boundary of mineralization (at depth). The rock mass between the Pinchi and Central Faults is interpreted to be half-graben structure which has been downfaulted.
North Fault	E-W striking steeply inclined fault.
Low Angle Shears	There are a series of low angle shears which dip shallowly towards the west along the form of the intrusives and unconformity. They have not been explicitly delineated but are worth noting for mine planning and future analysis.

Figure 16-5: Preliminary Geotechnical Domains - Kwanika (2018 MMTS)



Source: Kwanika Prefeasibility Study – Wrap-Up Report – WP5 Open Pit Geotechnical, November 2019.

16.2.1.7 Ground Support Strategy

Preliminary ground support guidance was developed by North Rock Mining Solutions for the underground development considering representative ranges in rock mass quality and material properties. The stability of proposed excavations was assessed and analyzed using the commonly employed empirical methodology after Grimstad and Barton (1993, 2008) and incorporated with the NGI Q data set to develop minimum ground support recommendations. Table 16-8 shows the results for the Class 2 ground conditions.

Table 16-8: Preliminary Ground Support Guidance for Class 2 Ground Conditions

Rock Mass Class	Opening Type	Cross Section (W x H, m)		Ground Support Type	Length (m)	Spacing (m)	Additional Notes
CLASS 2 (1 < Q < 10) (30<GSI), (RMR89<50) (poor to fair) Rock Strength R3/R4 Altered Intrusives Domain - Monzonites	Main access decline, ramps and other haulage routes	5.5 x 5.5	Back	Fully-grouted #7 resin-grouted rebar / D-bolt equivalent.	2.4	1.5 x 1.5	Welded-wire mesh with 50 mm of shotcrete to within 1.0 m of the floor
			Walls				
	Undercut Development	4.0 x 4.0	Back	Fully-grouted #7 resin-grouted rebar / D-bolt equivalent.	1.8	1.5 x 1.5	Welded-wire mesh with 50 mm of shotcrete to within 1.0 m of the floor
			Walls				
	Extraction Development	4.5 x 4.5	Back	Fully-grouted #7 resin-grouted rebar / D-bolt equivalent. Long support: Double strand 0.7" bulbed cable bolts (1.0 m bulb spacing as required)	1.8 6.0	1.5 x 1.5 2.5 x 2.5	Welded-wire mesh with 50 mm of shotcrete to within 1.0 m of the floor
			Walls	Fully-grouted #7 resin-grouted rebar / D-bolt equivalent.	1.8	1.5 x 1.5	

Excavation support ratios of 3.0 for temporary production headings, 1.6 for permanent worker-entry excavations and 1.3 for LOM infrastructure were used. Ground support designs were refined based on operational considerations such as standardization of ground support and referenced with support applied in analogous operations. Cablebolt lengths and spacings were determined using empirical methods. The use of split set bolts appears to be acceptable in the ribs provided the rock mass is sufficiently competent to mobilize the full strength of the bolt.

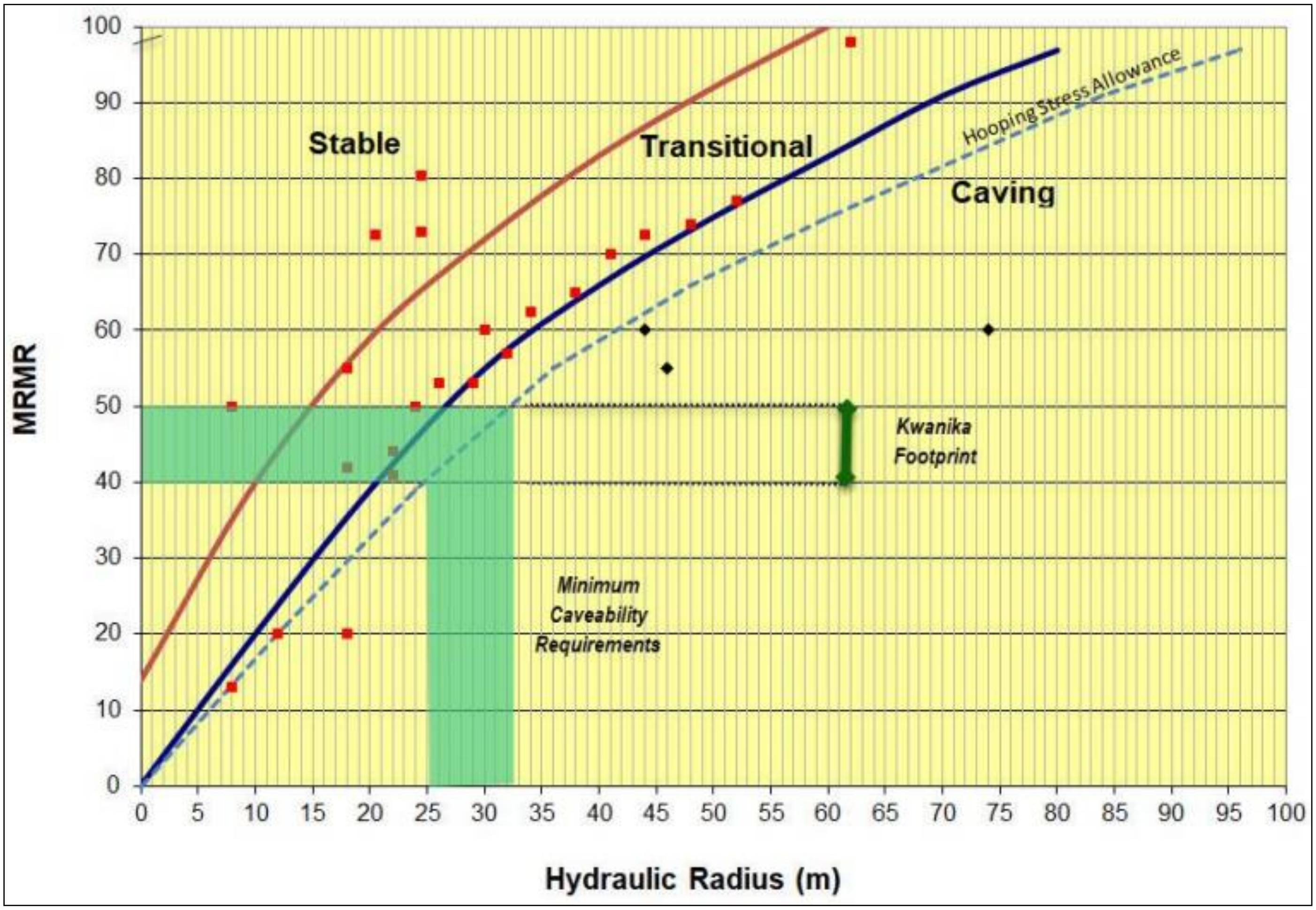
16.2.1.8 Caveability Assessment

Laubscher's Caveability Chart was employed to make a preliminary caveability assessment for the Kwanika Block Caving plan. A limited amount of drilling information was available to estimate Modified (Mining) Rock Mass Ratings (MRMR). Additionally, core photos from selected drillholes from earlier campaigns were reviewed to supplement the data set. The data indicates a progressive change in MRMR from 40 along the western half of the footprint, increasing to 50 towards the eastern side. Some drillholes East of the Central fault suggest MRMR's of 55 to 60 are possible. Figure 16-6 shows the Kwanika Block Cave footprint data against the Laubscher (1999) caveability chart.

The chart suggests that a hydraulic radius (HR) between 25 and 32 is required to induce continuous caving. The planned Kwanika footprint has a HR of 62 which is well in excess of the minimum requirements.

Cave propagation rates could be very rapid as the presence of faults may mobilize the altered intrusives forming an early connection to the surface or open pit bottom. Consequently, the transition between underground and open pit mining will need to be carefully timed.

Figure 16-6: Laubscher's Caveability Chart for the Kwanika Footprint (after Laubscher D., 1999)



Source: Kwanika Empirical Caveability Assessment_Draft. Oct 2019.

16.2.1.9 Subsidence

A preliminary empirical assessment for the extent of the subsidence zone was undertaken by Stratavision during the early states of the 2018 MMTS geotechnical study to assist with drill planning and consider preliminary options between the underground block cave and open pit mine planning. Beck Engineering (2019) facilitated this assessment by contributing a 3D coupled LGCA-DFE analysis using a Newtonian Cellular Automata (NCA) to simulate movements in the muckpile and a discontinuum strain softening Dilatant Finite Element (DFE) model to assess stability, stress and damage in the rock mass.

The reliability of the subsidence forecast is limited due to low resolution pit and domain shapes and structural model in addition to the potential for significant hydromechanical interactions and interaction with weathered zones or glacial till/detritus near surface.

The modelled cave is indicated to break back and subside as predicted, but near surface movement could be exacerbated by hydromechanical effects not captured. While the subsidence model has not been updated to include the revised Kwanika pit designs, it is instructive in suggesting that:

- The planned pit diversion of Kwanika Creek is impacted by minor to moderate disturbance from the pit which could be made worse by hydromechanical effects
- The cave-induced wall failure and caving would take place before the embankment is built, so the main impact of the underlying conditions is on settlement and the volume required to ensure a stable embankment.

16.2.2 Stardust

The Stardust geotechnical data is currently limited to raw geotechnical core logging data that has been interpreted to derive general rock mass classification values. Two hundred and seventeen (217) holes comprise the drillholes with useful geotechnical information. No structural information is associated with the drillhole database.

16.2.2.1 Geotechnical Considerations

Table 10 1 outlines the historical drilling database for Stardust. Table 16-9 provides a summary of the geotechnical core logging data available from historic drilling records. Data available includes RQD, core recovery, intact rock strength, degree of weathering, core break assessment, joint number and joint roughness and shape. The development of rock-type specific geotechnical information for the Stardust deposit is high priority target prior to advancing to a PFS-level study.

Table 16-9: Summary of Geotechnical Core Logging Information by Drill Campaign for Stardust

Description	Total drilled metres	No. of Holes	Geotech Data
DDH18-SD-406 to 427	6837.60	22	RQD, IRS, Weathering, Breakage, No. of Joints, Joints per metre, JR, Joint Shape
DDH19-SD-428D to 455D	21508.57	28	RQD, IRS, Weathering, Breakage, No. of Joints, Joints per metre, JR, Joint Shape
DDH20-SD-456M to 471	11695.00	16	RQD, IRS, Weathering, Breakage, No. of Joints, Joints per metre, JR, Joint Shape
DDH21-SD-472 to 474	2477.50	3	RQD, IRS, Weathering, Breakage, No. of Joints, Joints per metre, JR, Joint Shape
LD1979-01 to 03	615.40	3	Core Recovery
LD1980-01,02 - LD1981-03 to 06	819.14	6	Core Recovery
LD1991-01 to 11	971.20	11	Core Recovery
LD2004-01 to 21	6006.80	21	RQD
LD2005-01 to 17	5152.80	17	RQD
LD2006-01 to 32	6852.40	32	RQD
LD2007-01 to 12	3042.50	12	RQD
LKN2007-01 to 07	1622.00	7	RQD
LM2007-01 to 04	1496.00	4	RQD
AG2008-01B to 05	2654.70	5	RQD
LD2009-01 to 17	6105.72	17	RQD, Core Recovery
LD2010-01 to 10	3125.20	10	RQD, Core Recovery
LD2017-01 to 03	343.51	3	Core Recovery

16.2.2.2 In-situ Stress State

There are no local in-situ stress state measurements for the Stardust deposit. The in-situ stress state assumptions described in Section 16.2.1.2 can be considered applicable to the Stardust deposit. The planned underground development extends to around 775 m, a depth at which there exists the potential for stress related impacts to production and development.

16.2.2.3 Structural Model

The rocks underlying the Stardust property have experienced deformational events with at least two penetrative deformation processes, D1 and D2, which have influenced the current map pattern. A planar S1 fabric associated with the D1 deformation process is states to most likely align axial planar to the tight isoclinal folds that plunge around 40-50° to the north-northwest. The D2 folds are suggested to have similar orientations to the D1 folds, but tend to be slightly more open, and have shallower, 20° northwest plunges. Tight folding is believed to have come about due to buttressing against the Pinchi Fault which is believed to have originally been a major thrust fault. These folds, where observed, have a 10 to 60° N-NW plunge and minor axial plane shears are common.

The entire property has a strong NW-trending grain reflecting bedding, tight asymmetric folding, and bedding plane faults. The most important, and consistent fault structures demonstrated in the drill core are roughly coplanar to bedding. Some of these faults have the appearance of early east-verging reverse faults, which are largely lithologically controlled and mostly identified in the immediate hangingwall to the Canyon Creek Skarn.

The strongest and most strike discordant structural zone on the property is the structural zone and dyke system which hosts the Number 1 veins. This mineralized fault structure has a nearly north-south strike and moderate to steep west dip. In marked contrast, all structures, including lithology and major skarn bodies on the Stardust property have strike relationships which average 150° to 160° and steep westerly dips.

16.2.2.4 Rock Property Data

No rock property testing on core samples from the Stardust drilling inventory has been completed to date.

16.2.2.5 Rock Mass Classification

The Stardust geotechnical data base is thin and currently not attributed to a geological domain model. Table 16-10 shows a summary of the available geotechnical data for the Stardust deposit. This data has been referenced to estimate RMR_{1989} factors to arrive at a general rock mass rating of 65 (Fair to Good rock mass) as summarized in Table 16-11.

Table 16-10: Summary of Core Logging Geotechnical Data

Geotech Core Logging Parameter	Average	Standard Deviation	Count	Description
RQD%	67.5	18.16	193	Fair
Intact Rock Strength factor	6.23	1.37	69	50 - 100 MPa
Weathering factor	4.87	0.17	69	Fresh to Slightly Weathered
Joint Spacing (m)	0.67	0.35	69	
Joint Surface Roughness factor	2.34	0.22	69	Smooth/Rough
Joint Surface Shape factor	1.69	0.5	69	Planar/Curved
Core Break factor	2.51	0.45	69	Fair/Good

Table 16-11: Rock Mass Rating (Bieniawski, 1989) estimate for Stardust Property

ID	Factor	Rating	Description
A.1	Strength of Intact Rock Material	5 - 7	50 - 100 MPa
A.2	RQD	13 - 17	68% +-18
A.3	Spacing of Discontinuities	15	0.7 m avg
A.4	Condition of Discontinuities	20 - 25	Smooth/Rough
A.5	Ground Water	15	Assume Dry
B	Discontinuity Rating Adjustment	-5	Assume Fair
	RMR_{1989} Range	63 - 74	
	RMR_{1989} Design	**65**	

16.2.2.6 Ground Support Strategy

The indication of generally fair to good rock mass quality suggests that ground support requirements for Stardust will be similar comparison to the Kwanika underground estimates. See Section 16.2.1.7 should be referenced in this regard.

16.2.2.7 Underground Mining Considerations

General stope production parameters for the 2022 mine plan are summarized below in Table 16-12 below for longitudinal and transverse stoping (bottom-up sequence). A Matthews' Stability Graph analysis should be completed for the Stardust production stoping once the geotechnical database is updated.

Table 16-12: Stardust Deposit Stope sizing, Dilution, and Pillar Parameters

Parameter	Unit	Value
Stope Height	(m)	25
Stope Width (minimum)	(m)	2
Stope Width (maximum)	(m)	30
Stope Length	(m)	15
Minimum Pillar dimension	(m)	5
Minimum Stope Dip	(°)	50
Stope Dilution	(%)	10

Empirical analyses and numerical modelling should be employed to refine pillar sizing assumptions for the mine plan as well. This should include guidelines for offset distances of capital development and infrastructure from production stoping areas.

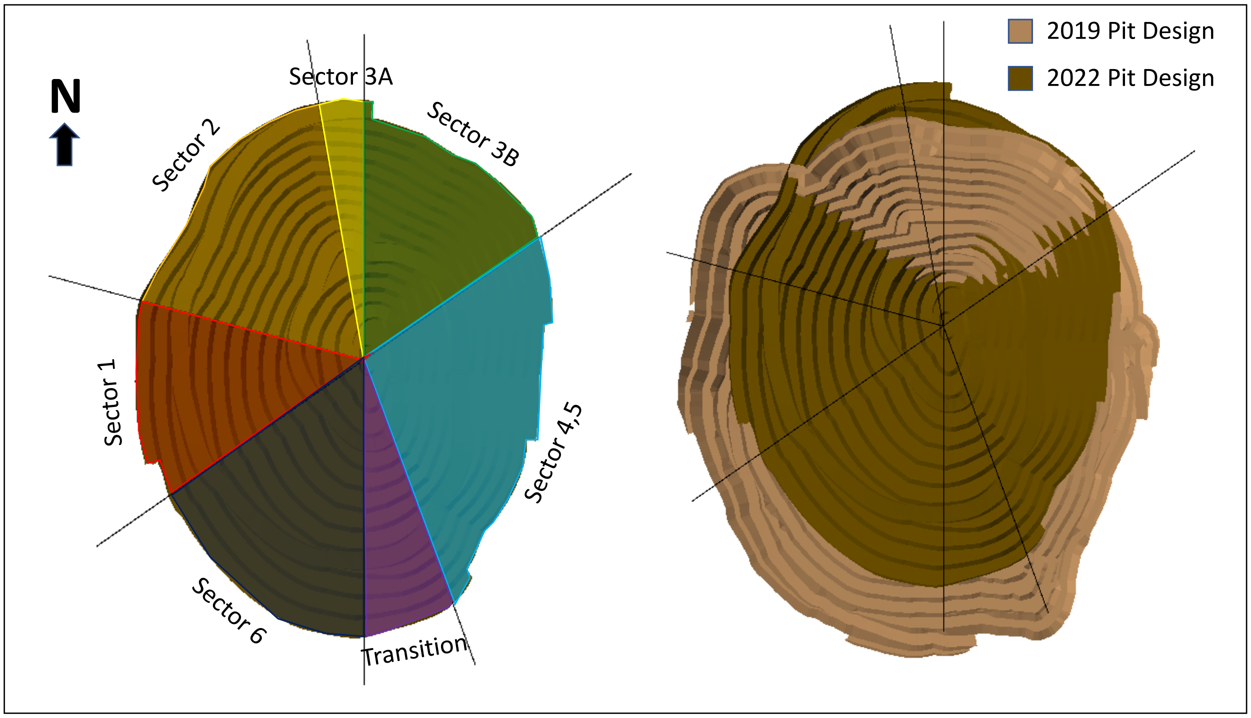
16.2.3 Kwanika Open Pit Geotechnical Considerations

Moose Mountain Technical Services (MMTS) completed a preliminary geotechnical assessment in 2018 to provide suitable pit slope angles for PFS-level mine planning. The assessment is based primarily on past resource drilling data, core photographs, RQD data, detailed geotechnical data from the 2018 drilling program, Whittle pit shells, geologic and structural models, and relevant background reports.

MMTS divided the pit into individual slope design sectors based on the slope height and dominant geology. Estimates of suitable overall slope angles were developed for each sector with an overall slope recommendation ranging between 32° and 44° for the Kwanika Central pit.

Figure 16-7 illustrates the Pit design in Plan View and a side comparison to the 2019 internal report. As shown, the 2022 Pit design is almost entirely within the 2019 design with the exception of the north slope in Sectors 2, 3A and 3B. Table 16-13 and Table 16-14 summarize the current open pit design parameters which honour the 2019 geotechnical design guidelines.

Figure 16-7: (L) 2022 Pit Design (R) Comparison of 2019 and 2022 Pit designs



Date. Source: Mining Plus 2022

Table 16-13: 2022 Kwanika Central Open Pit Design Parameters by Sector

Sector	Azimuth		OSA[1]	Max IRA	Minimum Berm Width
	From	To	(°)	(°)	(m)
1	235	285	37	41	17.7
2	285	350	41	44	15.2
3A	350	0	43	46	13.7
3B	0	55	43	46	13.7
4&5	55	160	44	48	12.2
Transition	160	180	38	42	16.8
6	180	235	32	35	23.7

[1] *2019 Internal Report Pit Design Values*

Table 16-14: 2022 Kwanika Central Open Pit Design Parameters

Parameter	Units	Value
Bench Height	(m)	20.0
Bench Face Angle	(°)	70.0
Minimum Berm Width	(m)	8.0
Ramp Gradient	(%)	10.0
Ramp Width (Double Lane)	(m)	25.2
Ramp Width (Single Lane)	(m)	18.9
Minimum Mining Width at bottom of pit	(m)	30.0

The South pit design honours the parameters in Table 16-14 with an inter-ramp slope angle of 40°.

16.3 Hydrogeological Considerations

Hydrogeological assessments were performed on Klohn Crippen Berger (KCB) during the 2018 MMTS drilling campaign. Water inflow criteria provided by KCB indicates peak ground water inflows of 10 L/s during mine development which taper off to 2 L/s under steady state conditions. This data forms the basis for site water balance. No additional hydrogeology studies were completed on this Project.

16.3.1 Kwanika

Hydrogeological assessments were performed by KCB during the 2018 MMTS drilling campaign. Water inflow criteria provided by KCB indicates peak ground water inflows of 10 L/s during mine development which taper off to 2 L/s under steady state conditions.

16.4 Underground Mine Design

16.4.1 Stardust Underground

16.4.1.1 Mine Access

The Stardust mine will be accessed via one decline located 7 km away from the Kwanika Underground and Open Pit Site. This will serve as the primary access point for haulage, men, and materials. A haul road will be built between the two mine sites, and camp for Stardust will remain at Kwanika. The decline accesses the mine at 1,374 m above sea level. A raise bore shaft will be located at 1,380 m above sea level, 273 m away from the portal for exhausting ventilation. The shaft will be connected to the ramp on the 1,290 L. Mineral and waste stockpile will be located on the surface before transport to the main Kwanika stockpiles, where the material will be hauled. A small office trailer will also be located near the portal for supervisors and safety.

16.4.1.2 Underground Mine Development

The main ramp continued from the decline will access the mine sub-levels on 25 m sublevel spacing. Mineral and waste will be transported utilizing 30-tonne haul trucks to surface. The maximum gradient on the ramp is 13.5%. The underground ramp dimensions (5.0 m high x 4.5 m wide) will accommodate 30-tonne trucks. The ramp extends from the

1290L to the 590L. Lateral Development dimensions (4.5 m high x 4.5 m wide) will support all truck haulage and each profile will be flat back. The lateral development will support 1,650 t/d operation over the 7-year mine life.

Levels will accommodate a combination of Transverse and Longitudinal Stoping. Trucks will be side loaded and travel up the main ramp to surface. Avoca Mining is incorporated where possible in the longitudinal stoping areas. Secondary development is generally flat with gradient where appropriate. Development at Level Accesses will include electrical Cut-outs, sumps, borehole cut-outs, and ventilation accesses.

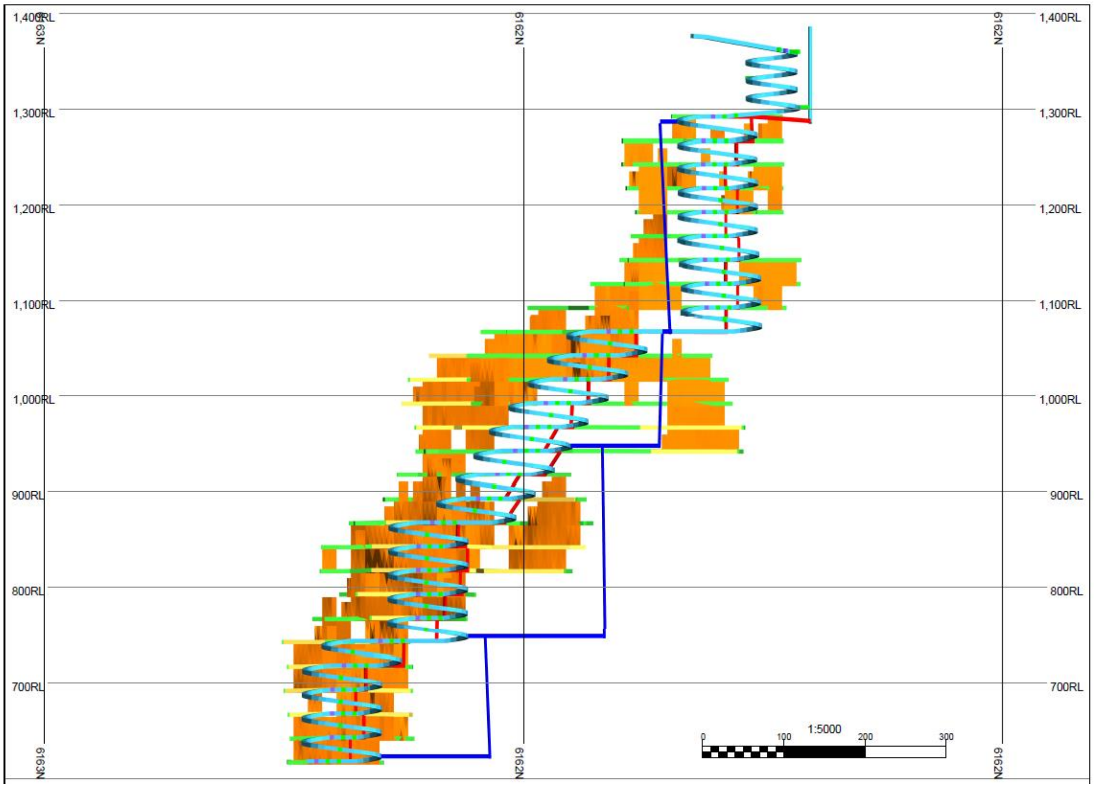
Table 16-15 summarizes the lateral and vertical development for Stardust over the life of mine. Most of the development will be completed in years 1 through 4 with some secondary development in the final 2 years. Figure 16-8 shows the long section for the underground design. As shown the bulk of the tonnes will start at the 1100 L.

Standard ground support of rebar and split sets will be utilized for supporting underground development. Wire mesh will be installed on the back and ribs for all development. Cable bolts will be installed in the hanging wall of stopes and intersections where necessary.

Table 16-15: Lateral and Vertical Development

Development	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Lateral Development (m)								
Level Access	2,224	370	807	561	402	85		
Borehole Cut Out	98	22	24	28	25			
Ramp	6,534	1,927	1,471	1,780	1,356			
Electrical Cut Out	196	42	49	56	49			
Fresh Air Drive	418	20	28	102	268			
Footwall	2,821	0	140	1,698	614	369		
Muck Bay	336	72	84	96	84			
Mineral Drive (N/A)	9,971	354	2,732	3,742	1,991	925	227	
Return Air Drive	1,376	320	387	407	226	36		
Sump	196	42	49	56	49			
Total	24,170	3,169	5,771	8,526	5,062	1,415	227	
Vertical Development (m)								
Fresh Air Raise	690	0	238	120	277	55		
Return Air Raise w/ Manway	733	159	169	230	150	25		
Surface Vent Raise	105	105	0	0	0	0		
Total (m)	1,528	264	407	349	427	80		

Figure 16-8: Stardust Long Section



Source: Mining Plus, 2022.

16.4.1.3 Materials Handling

Mineral and waste will be hauled to surface utilizing 30-tonne haul trucks. From the portal the material will be transported with surface haul trucks to the Kwanika Mill via the 7 km haul road to the Stardust Portal. Waste and Cemented Rock Fill (CRF) will be back hauled into Stardust for stope backfill.

16.4.1.4 Mine Ventilation

Vertical Development will include one raise to surface as well as four internal intake raises, and 27 return raises between levels. The internal raises will be developed by raise bore. The raise to surface will be connected into the mine at the 1290L, first stoping level, for exhaust and start the ventilation circuit. The main raise will be a 4 m diameter raise bore. Internal ventilation raises will be 3 m diameter raise bores for return air. Fresh air raises in the levels will be 3 m by 3 m square drop raises or equivalent raise bore.

16.4.1.5 Mine Dewatering

A main pump station infrastructure has been accounted for in the capital budget and underground sumps have been included in the design. No additional information has been provided for the PEA.

16.4.1.6 Mine Operations

The mine will be contractor-operated for the short mine life within the entire project. The mine will start by developing the main decline to 1290L where the bottom of the main ventilation raise will be established, and excavation will commence. Once the mine has established the main ventilation circuit, production will commence.

Mine technical services, such as geology, engineering, and management will be the owner's responsibility and are captured in the general mine expense area. In this regard, this cost has been accounted for in the mining G&A. The technical services team will share resources with the Kwanika Block Cave and will utilize a small team for the Stardust Mine.

16.4.1.7 Mine Equipment

Table 16-16 details the stardust underground equipment list.

Table 16-16: Stardust Mobile Equipment List

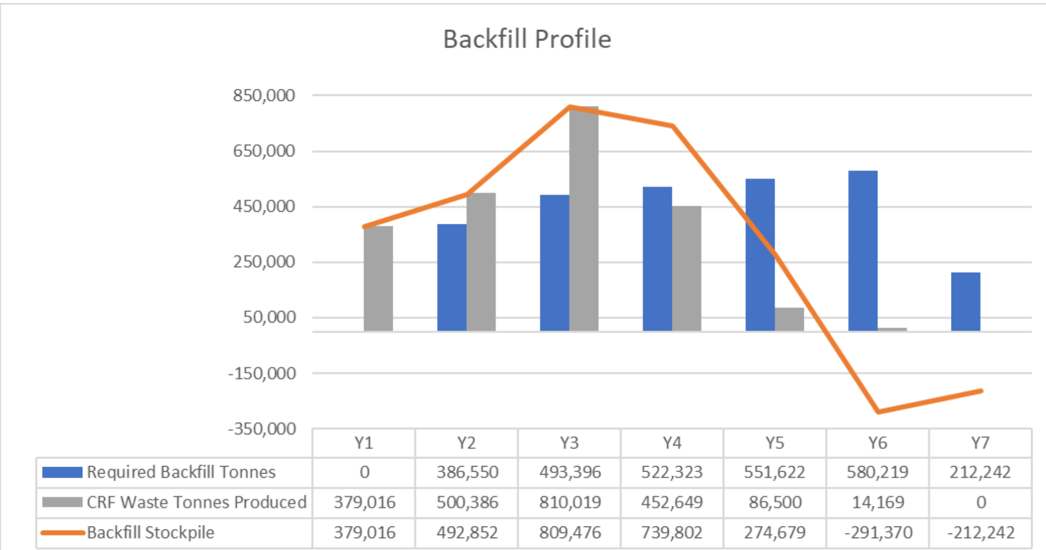
Mobile Equipment	Quantity
Contractor Lease Total	**33**
Jumbo Drill Rig (2-Boom electric-hydraulic)	2
Rockbolting Jumbo	2
Development Haul Truck - 40t	6
Development LHD 5.4 m^3	2
Development Charge Up	1
Scissor Truck	1
Shotcrete Sprayer (for rehab)	1
Concrete Transmixer (for rehab)	1
Longhole Drill	2
Stope Charge Up	1
Production LHD 5.4 m^3	2
Service Truck	1
Grader	1
Personnel Carrier	1
Small Personal Vehicles (22 person)	7
Telehandler	2
Owner Purchased Subtotal	**8**
Grader	1
Surface Truck	4
Small Personal Vehicles	3

16.4.1.8 Underground Mine Production & Schedule

Stoping will be primarily longitudinal for the years 1 through 3 and will be split between longitudinal and transverse stoping the remainder of the mine life. Stopes lengths were limited to 15 m long on longitudinal stopes long on transverse. Stopes less than 15 m wide were mined by the longitudinal method and stopes wider than 15 m wide were mined in the transverse method. As shown in Figure 16-10, 68% of the stoping will be mined utilizing a Longitudinal Stoping method and 32% will utilize the Transverse method. The entirety of the transverse stoping will occur between the 890L and 765L.

Mine backfill as shown in Figure 16-9, will be supplied by waste from Stardust for years 1 through 5. Years 6 and 7 will need waste back hauled from Kwanika waste storage. Stardust will utilize CRF for backfilling primary Transverse Stopes and Longitudinal Stopes. Cement content is assumed to average 5% by mass as required. Secondary stopes will be backfilled with waste rock.

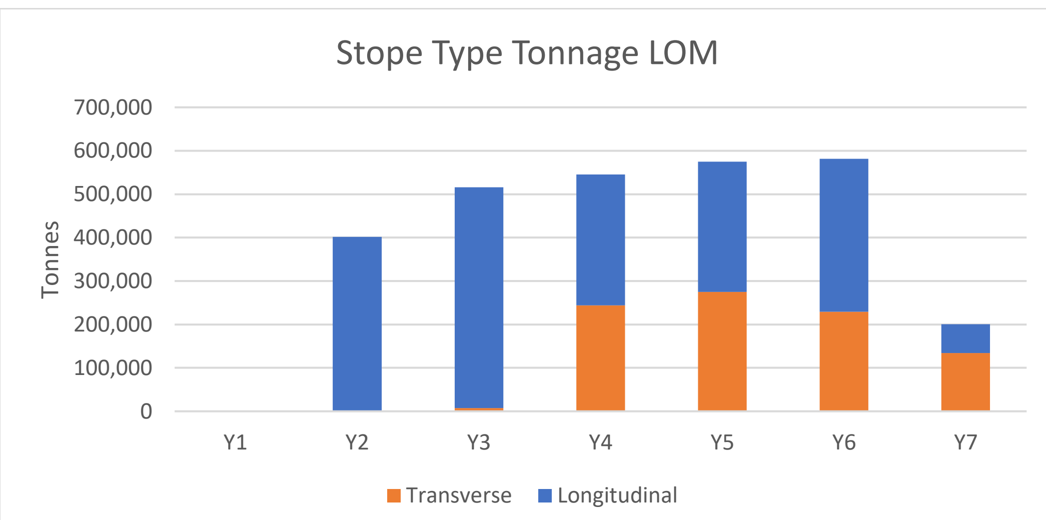
Figure 16-9: Backfill Profile



	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Required Backfill Tonnes	0	386,550	493,396	522,323	551,622	580,219	212,242
CRF Waste Tonnes Produced	379,016	500,386	810,019	452,649	86,500	14,169	0
Backfill Stockpile	379,016	492,852	809,476	739,802	274,679	-291,370	-212,242

Source: Mining Plus, 2022.

Figure 16-10: Stope Tonnage Profile



Source: Mining Plus, 2022.

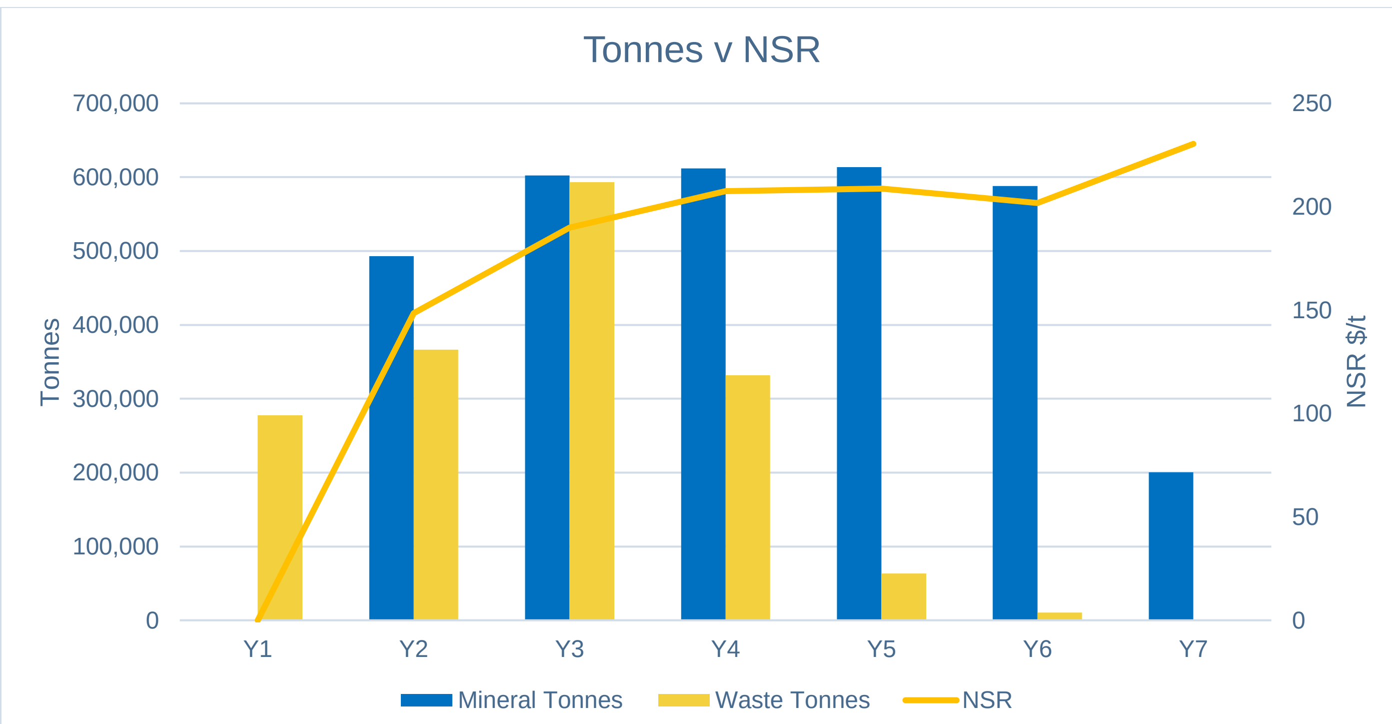
Year 1 will be entirely waste development with the decline and main vent raise being established. Mineral Production will commence in Q1 of Year 2. Development to the main portion of the mineral body at the 1065 L. Most capital development will be completed by the end of year 4. The final 3 years of development will be mineral drives and footwall drives. Stardust will average 1,600 mineral tonnes per day for all but Year 7 of the mine plan. As shown in Figure 1611, the mine will reach steady state production in year 2 and maintain and steady average of 604,000 tonnes until the final year of mine life.

Table 16-17 shows the life of mine Production Profile and Metal concentrations.

Table 16-17: Production Profile

	Tonnes (t)	Y1	Y2	Y3	Y4	Y5	Y6	Y7
Mineral Tonnes	3,109,824	0	493,134	602,325	612,008	613,720	588,081	200,556
Waste Tonnes	1,643,033	277,668	366,583	593,421	331,611	63,370	10,381	0
Cu (%)	1.329	0	1.106	1.227	1.302	1.449	1.445	1.568
Au (g/t)	1.466	0	1.031	1.501	1.657	1.539	1.436	1.707
Ag (g/t)	27.793	0	20.358	28.990	33.043	27.372	26.397	31.842
NSR/t	195.41	0	148.58	189.93	207.53	208.77	201.83	230.40

Figure 16-11: LOM Production Profile

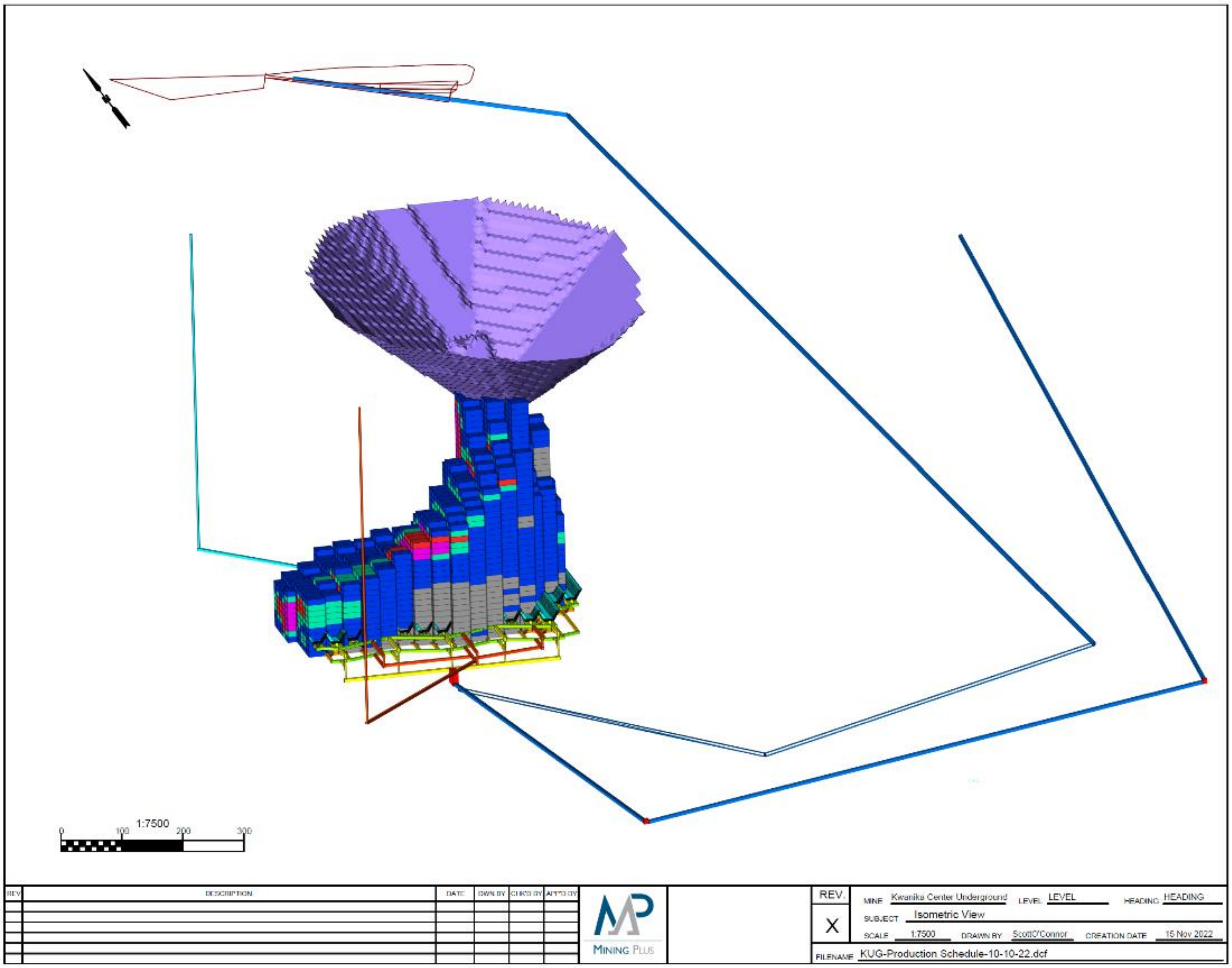


Source: Mining Plus, 2022.

16.4.2 Kwanika Central Block Cave

The mine design for Kwanika Central Block Cave was generated for NorthWest Copper as part of an internal report. At the conclusion of the study in 2019, mine planning was estimated at 60% complete relative to prefeasibility standards and is viewed as sufficient to scoping level accuracy. The modifying factors assumed to determine the block cave inventory are 15% dilution at 38.41% of in-situ grade and 88.52% mining recovery. These factors were derived from a 2019 calibration of Kwanika underground in-situ material to flow model recovered material utilizing PCBC software. The block cave inventory is limited to material above cut-off and outside of the selected 29 Mt Kwanika Central Open Pit shell described in Section 16.5.2.1. A cut-off value 21.25 ($/t) was used to determine the footprint boundary, which is representative of the total opex inclusive of mining, transportation, G&A and processing costs. Additionally, a $2,500 per square metre hurdle, or $675,000 per drawpoint, was applied to account for the incremental footprint and drawbell development capital costs. The resulting mineral inventory for Kwanika Central Block Cave is 44 Mt and represented by 301 drawpoints at a nominal spacing of 15 m x 18 m, as shown in Figure 16-12.

Figure 16-12: Kwanika Central Block Cave Isometric View



Source: Mining Plus, 2022.

The Kwanika Central Block Cave mineral inventory is presented in Table 16-18.

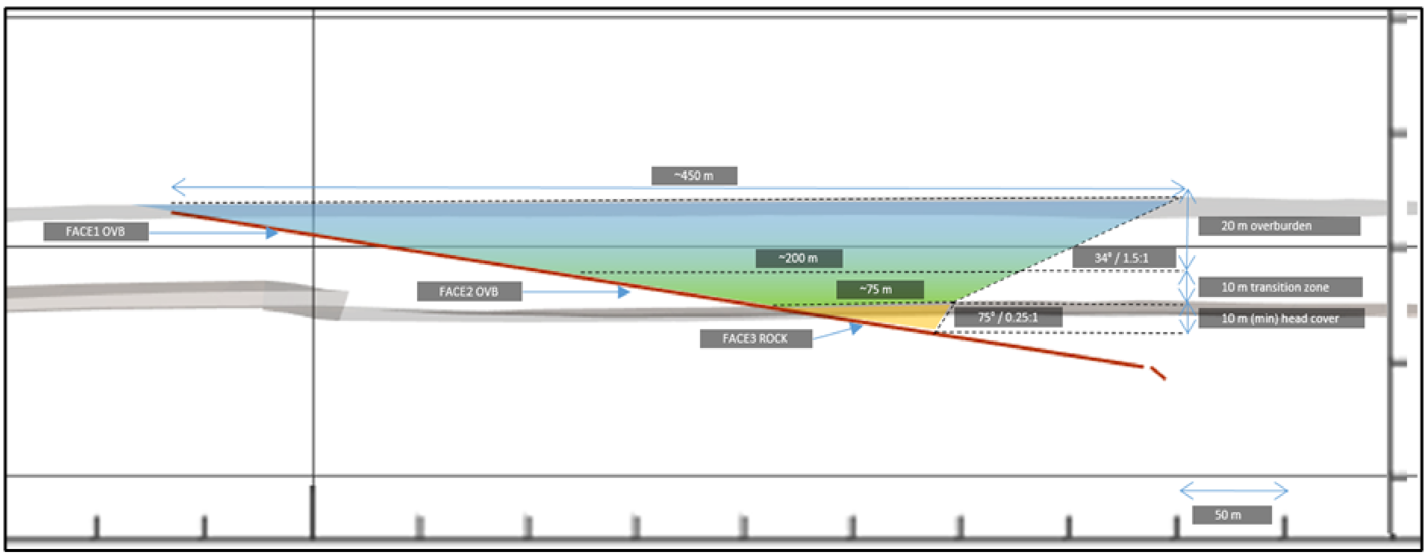
Table 16-18: Kwanika Central Block Cave Inventory

Kwanika Centre Block Cave	ktonnes	Au (g/t)	Ag (g/t)	Cu (%)	NSR ($/t)
Measured	31,016	0.50	1.35	0.42	55.58
Indicated	12,598	0.57	1.38	0.45	61.65
Unclassified Waste	424	0	0	0	0

16.4.2.1 Mine Access

The Kwanika Centre Block Cave will be accessed by a box cut and portal located on the south side of camp at around 1,020 m above sea level and is shown in Figure 16-13. The around 4 km single-heading ramp is designed at 5.5 m x 5.5 m profile and -15% gradient and will serve as the primary access for personnel and equipment. A ventilation raise from surface equipped with an escape hoist will be used as the secondary means of egress.

Figure 16-13: Kwanika Portal Box Cut.



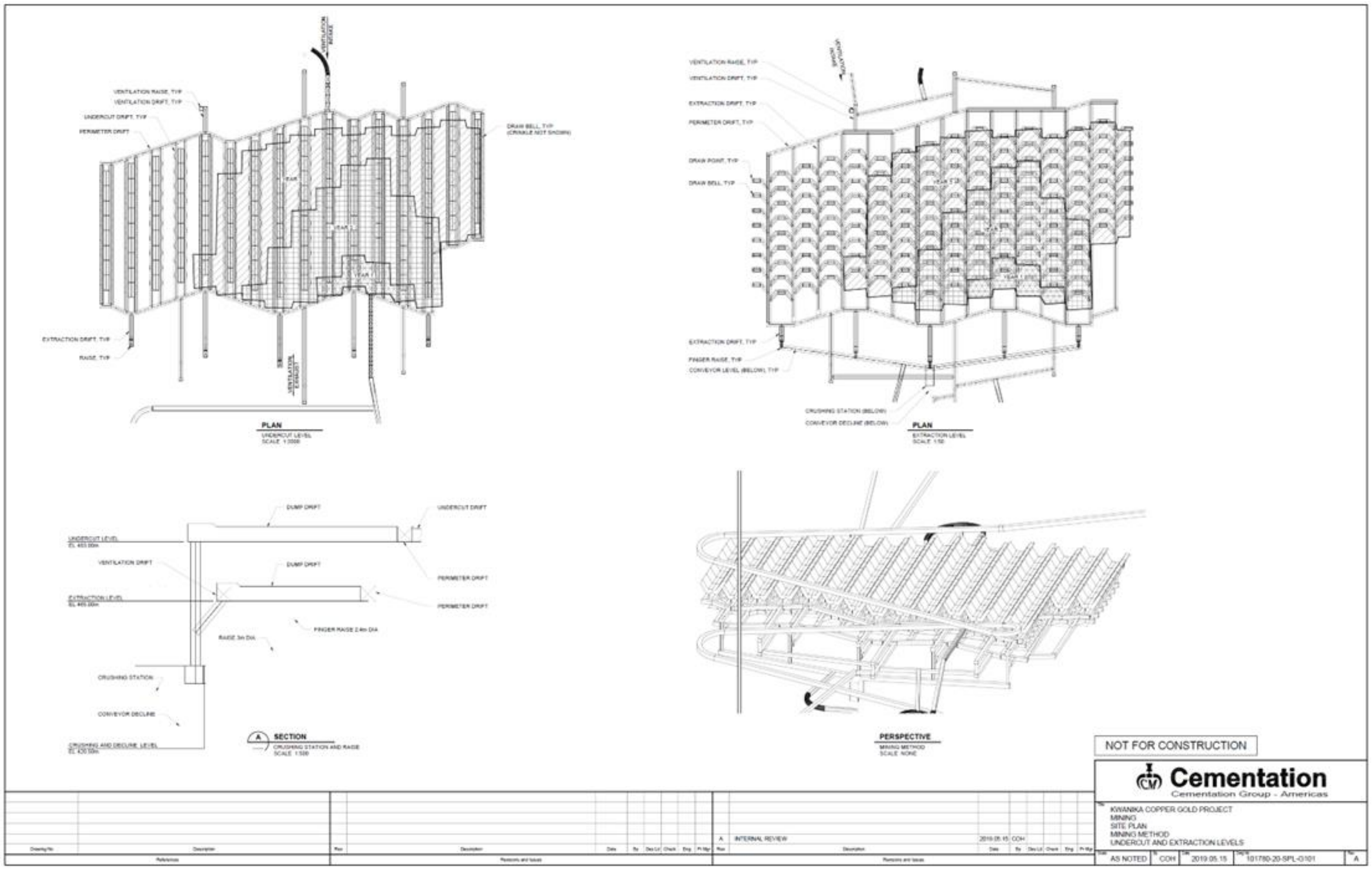
Source: Internal Report, 2019

16.4.2.2 Underground Mine Development

The Kwanika Central Block Cave is typical block cave layout comprised of a main ramp, interlevel ramps, undercut level, extraction level, conveyor transfer level, main conveyor and ventilation raises from surface. Total horizontal development metres are shown in Table 16-19.

Table 16-19, which has been scaled up from the original 2019 Internal Report estimate to account for the 6 additional planned drawpoints of the main conveyor drive. Extraction drive spacing is 30 m and drawbell spacing is 18 m. A schematic of the block cave footprint is shown in Figure 16-14.

Figure 16-14: Kwanika Central Block Cave Layout



Source: Internal Report, 2019

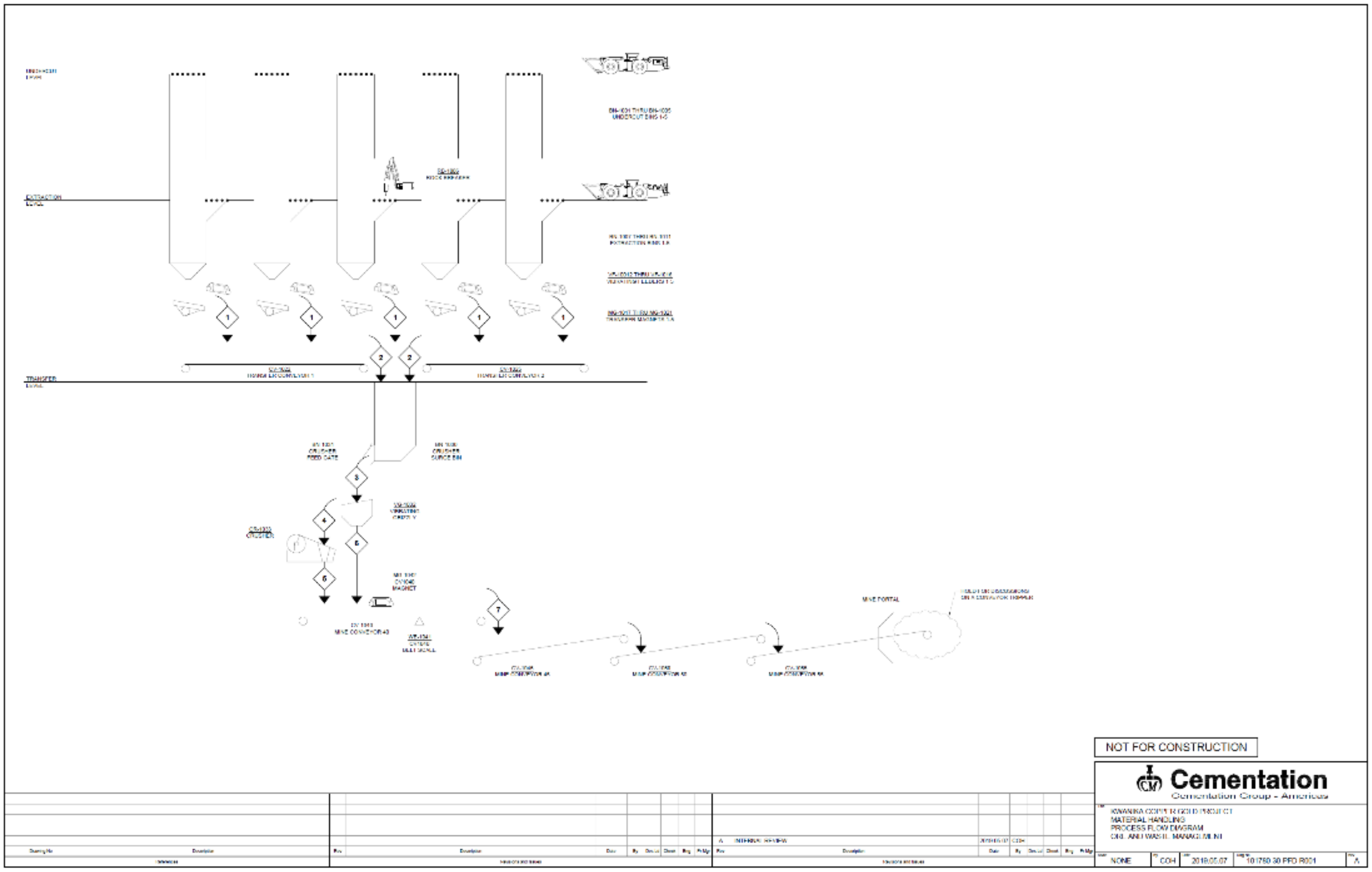
Table 16-19: Kwanika Central Horizontal Development

Horizontal Development	Heading Size (metres)	Metres
Main Access Ramp	5.5 x 5.5	4,248
Interlevel Ramps	4.5 x 4.5	1,248
Undercut Level Waste Development	4.5 x 4.5	1,706
Undercut Level Mineral Drives	4.5 x 4.5	5,070
Extraction Level Waste Development	4.5 x 4.5	1,406
Extraction level Mineral Drives	4.0 x 4.0	2,902
Ventilation Level Development	4.5 x 4.5	1,517
Conveyor Level Development	4.5 x 4.5	519
Extraction Level Drawpoint Development	4.5 x 4.5	4,310
Main Conveyor	4.5 x 4.5	3,890
Total Metres (excluding mass excavations)		**26,816**

16.4.2.3 Materials Handling

The Kwanika Central Block Cave materials handling system consists of five mineral passes between the undercut level and transfer level with extraction level finger raises and grizzlies located at each collar. The mineral passes feed two converging transfer conveyors that feed a single crusher. Crushed material is conveyed to surface and mill. A schematic of the material handling system is shown in Figure 16-15.

Figure 16-15: Kwanika Central Material Handling System



Source: Internal Report, 2019

16.4.2.4 Mine Ventilation

The ventilation for Kwanika Central Block Cave was modelled by R. E. Ackermann in 2019 as part of the Internal Report and has not been reviewed other than to confirm the input assumptions are still valid. Modelling was executed using VentSIM software and assumed a 6 m concrete lined intake raise from the surface on the north side of the footprint and 5 m concrete lined exhaust raise located on the south side of the footprint. Two parallel main Howden fans, fit with heaters, are proposed feeding the intake raise. Secondary fans located underground serve to direct airflow to the various levels, shops, and crusher. The extraction level assumes battery operated LHDs, each requiring 18,000 CFM to operate. The overall airflow requirement for the mine is 800,000 CFM, which satisfies up to 15 concurrent production drives in operation.

16.4.2.5 Mine Dewatering

Ground water inflows during mine development and production were determined in the 2019 Internal Report to be of minor consequence with peak inflows in the order of 10 L/s and steady state inflows of 2 L/s. No additional information has been provided since the 2019 Internal Report was completed.

16.4.2.6 Mine Operations

The Kwanika Central Block Cave is assumed to be developed and constructed by contractor. This includes earthworks, portal, surface facilities near the portal, ventilation facilities, underground excavations, and underground construction.

The mining development will begin with two mining crews performing the main ramp access and main conveyor decline development until the interlevel ramp intersection is reached. The priority of this crew will be to develop to the bottom of the South ventilation raise. A third mining crew and equipment fleet will be added after development to the interlevel ramp has been reached, and multiple headings will be available. One crew will focus on advancing development to the bottom of the North ventilation raise, while the other crew will focus on development required on the conveyor level. Once the ventilation access and the conveyor development has been completed, the mining crews will focus on developing the Undercut and Extraction Levels.

A truck haulage crew will be required to remove waste rock until the conveyor is installed in the ramp. After the conveyor is installed, the truck haulage personnel are demobilized from site, and waste rock is carried by the conveyor to surface. The conveyor construction is assumed not to impact critical path of the development mining.

One development mining crew will demobilize from site after the Undercut Level development is complete, while the other crew continues the Extraction Level development and the drawbell drilling and blasting. After all drawbells have been completed, the development crew and the contractor's indirect personnel will demobilize from site. The only remaining contractor crew on site will be the drawbell rehabilitation crew, which will remain on site until the end of production.

The mine owner will be responsible for production LHD and material handling operation, including support services such as rock breaker operation and maintenance. Assumed use of 8 yd^3 (6.1 m^3) LHDs will be used to transport material from drawpoint to mineral passes. Mineral passes range from 50-250 m from drawpoints. Operating time is based on nominal 7 km/h which includes speed-up, slow-down time for entering and exiting drawpoints, corners and mineral passes. A total of 20 daily productive hours is assumed, spread evenly across two 12-hour shifts. The mine will operate with a 14-days on, 14-days off, fly-in/fly-out camp rotation.

16.4.2.7 Mine Equipment

The underground mobile equipment required to develop, construct, and operate Kwanika Central Block Cave is shown in Table 16-20. Contractor equipment is leased, and owner equipment is purchased.

Table 16-20: Kwanika Mobile Equipment

Mobile Equipment	Quantity
Contractor Lease Subtotal	**45**
Jumbo Drill Rig (2-Boom electric-hydraulic)	3
Development Haul Truck – 40 t	5
Development LHD – 9 m^3	4
Rockbolting Jumbo	2
Development Charge Up	2
Scissor Truck	3
Shotcrete Sprayer battery powered (for rehab)	2
Concrete Mixer Truck battery powered (for rehab)	3
Longhole Drill	3
Cablebolt Drill battery powered (for rehab)	2
Telehandler	1
Water Truck	1
Service Truck	1
Mobile Crane	1
Production Charge Up	1
Integrated Tool Carrier	2
Grader	1
Small Personnel vehicles UG	5
Skid Steer	1
Personnel Carrier battery powered (22 person)	1
Water Truck	1
Owner Purchase Subtotal	**23**
LHD (8yd^3) Battery powered	9
Grader	1
Mobile Rockbreaker	2
Blockholer	2
Boom Trucks battery powered	1
Small Personnel Vehicles	3
Personnel Carrier battery powered (22 person)	1
Telehandler	1
Pickup Trucks	3

16.4.2.8 Underground Mine Production & Schedule

The pre-production period for Kwanika Central Block Cave is estimated at around 55 months duration, prior to commencement of production activities. The activities shown in Table 16-21 are overall project years, starting at year –2 when the processing facility construction begins.

Table 16-21: Kwanika Central Block Cave Development and Production Timeline

Activity	Project Year
Start Date: UG Mining Contractor Mobilization	Year -2
Portal Construction Complete	Year -1
Main Access Ramp Complete	Year 2
Main Ventilation Network Complete (Surface Vent Raises and UG connection)	Year 3
Crushing/Conveying System Complete	Year 3
Start Production Year 1	Year 3
UG Mine Development Complete	Year 6
Production Complete	Year 12

The production ramp-up commences in Year 3 quarter 3 at a draw rate of 130 mm/d and is maintained until the 25 m critical HR is established in Year 4 quarter 3. Residual material from the Kwanika Central Open Pit is exhausted prior to critical HR induced caving. After the open pit is exhausted and critical HR is established the draw rates shown in Table 16-22 apply.

Table 16-22: Kwanika Centre Block Cave Draw Rates

Percentage of Column Extracted (%)	Draw Rate (mm/day)
0 to 15	130
15 to 30	190
30 to 100	250

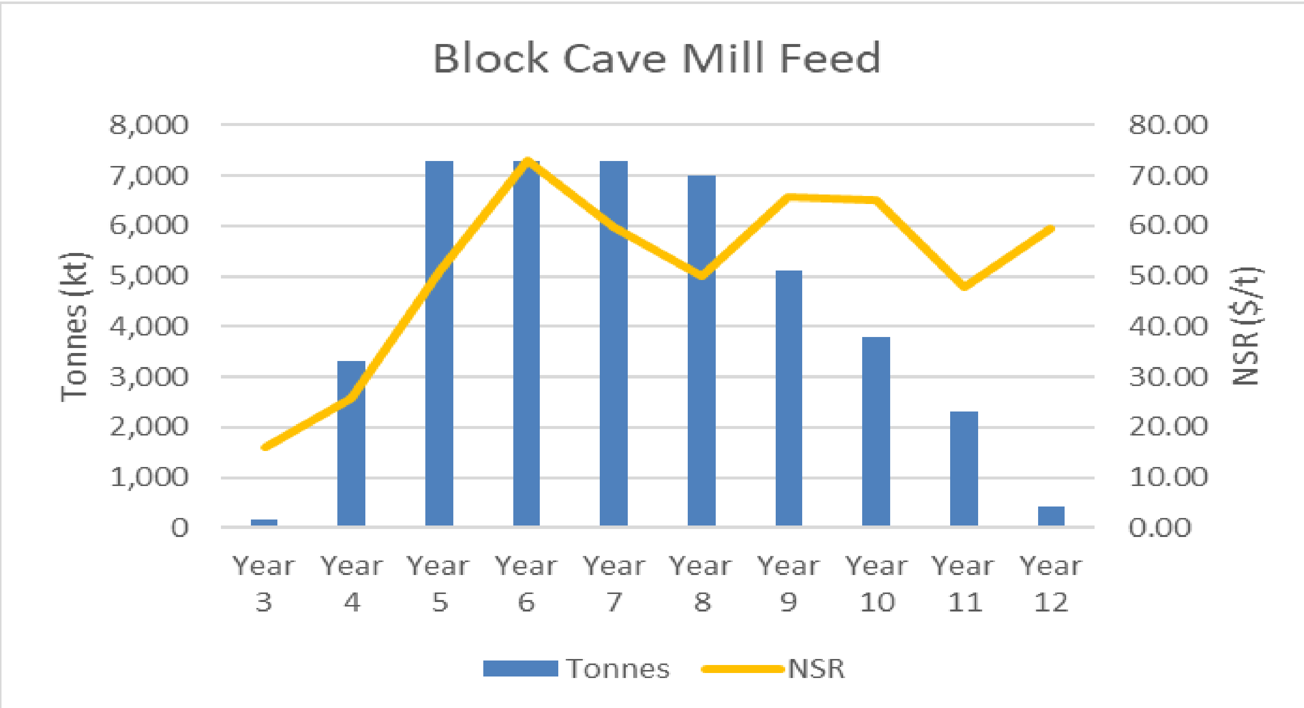
Drawbell construction activities commence with production at 1 drawbell per month and slowly increase to a maximum rate of 5 drawbells per month over the mine life as shown in Table 16-23.

Table 16-23: Kwanika Centre Drawbell Construction Rate

Block Cave Production Month	Drawbell Construction Rate (DB/month)
1	1
2 through 3	2
4 through 6	3
7 onwards	5

After an 18-month ramp-up period nameplate 20,000 t/d production for the block cave is achieved and maintained for 45 months before beginning ramp-down as drawpoints are exhausted. The overall Kwanika Central Block Cave production schedule is shown in Figure 16-16.

Figure 16-16: Kwanika Centre Block Cave Production Schedule



Source: Mining Plus, 2022.

16.5 Open Pit Mine Design

Two open pit mines have been studied to maximize the NPV of the project in conjunction with the Kwanika Block Cave and Stardust underground mines. Open pit mineral resources are reported in relation to a conceptual pit shell considering all resource categories including measured, indicated, and inferred materials.

16.5.1 Net Smelter Price and Cut-Off Grade

The NSR was calculated for each block in the block model to reflect the value of each block based on copper, gold, and silver grades. The input data used for the NSR calculation is shown below in Table 16-24.

Cut-off Grade is determined using an estimated NSR in $/t, which is calculated using Net Smelter Prices (NSP). The NSR (net of off-site charges and mill recovery) is used as a cut-off item for the break-even economic selection of mineralized material.

The calculated NSR Cut-off grade for the open pit is $11.19/t.

Table 16-24: Input data for NSR calculation

Parameters	Unit	Value
Price_Cu	US$/lb	3.5
Price_Au	US$/oz	1650
Price_Ag	US$/oz	21.5
Con_Cu	1/n	0.25
Treatment	US$/dmt	75
Refine_Cu	US$/lb	0.075
Refine_Au	US$/oz	5
Refine_Ag	US$/oz	0.45
Transport	US$/wmt	109.03
FX	$US/$C	0.7722
Payable_Cu	1/n	0.965
Min_Deduct_Cu	1/n	0.01
Con_Ag	g	30
Payable_Ag	1/n	0.9
Payable_Au	1/n	0.95
lbtonne	lbs/tonne	2204.626
goz	grams/oz	31.10348
Moisture	1/n	0.08
Process recoveries		
Cu Processing recovery (curecop)	IF([Cu]<0.1, 0.5, IF([Cu]<1, 0.93833*[Cu]^0.0655, 0.95))	
Au Processing recovery (aurecop)	MIN(0.10*[Au]+0.66, 0.85)	
Ag Processing recovery (agrecop)	MIN(IF([Ag]<0.5, 0.1, 0.32493 + 0.25676 * LN([Ag])), 0.62)	
NSR Coding in the block model		
metconcuop	[*Volume]*[Density]*([Cu]/100)*[curecop]*GC("lbtonne")	
metconauop	[*Volume]*[Density]*[Au]*[aurecop]/GC("goz")	
metconagop	[*Volume]*[Density]*[Ag]*[agrecop]/GC("goz")	
condmtop	[metconcuop]/GC("lbtonne")/GC("Con_Cu")	
paycuop	[metconcuop]*GC("Price_Cu")*GC("Payable_Cu")*(GC("Con_Cu")-GC("Min_Deduct_Cu"))/GC("Con_Cu")/GC("FX")	
payauop	[metconauop]*GC("Price_Au")*GC("Payable_Au")/GC("FX")	
payagop	IF([metconagop]*GC("goz")>GC("Con_Ag"),[metconagop]*GC("Price_Ag")* GC("Payable_Ag")/ GC("FX"),0)	
payop	[paycuop]+[payauop]+[payagop]	
refineop	IF([condmtop]>0, ([metconcuop]*GC("Refine_Cu")*((GC("Con_Cu")-GC("Min_Deduct_Cu"))/GC("Con_Cu"))+([metconauop]*GC("Refine_Au"))+([metconagop]*GC ("Refine_Ag")))/GC("FX"), 0)	
treatop	IF([condmtop]>0, [condmtop]*GC("Treatment")/GC("FX"),0)	
transportop	IF([condmtop]>0, [condmtop]/(1-GC("moisture"))*GC("Transport")/GC("FX"),0)	
nspop	[payop]-[refineop]-[treatop]-[transportop]	
NSR_OP	[nspop]/([*Volume]*[Density])	

16.5.2 Kwanika Central Pit

16.5.2.1 Central Pit Optimization

The pit optimization was conducted using Datamine NPVS software and the Lerchs-Grossman (LG) algorithm. The pit optimization inputs used in this study are shown below in Table 16-25.

Table 16-25: Input Data for Pit Optimization

Block Model		
Block model	**Kwanika Central:** Kwanika_Mar2022_RR	
Resource Categories	**Measured, Indicated and Inferred**	
Technical Factor		
Overall slope angles	**Kwanika Central:** Geotech sectors	
	Wall Azimuth	**Pit slope**
	0.00	43.00
	55.00	44.00
	145.00	38.00
	165.00	32.00
	235.00	37.00
	285.00	41.00
	350.00	43.00
	360.00	43.00
Mining dilution	2%	
Mining loss	5%	
Cu Processing recovery	IF([Cu]<0.1, 0.5, IF([Cu]<1, 0.93833*[Cu]^0.0655, 0.95))	
Au Processing recovery	MIN(0.10*[Au]+0.66, 0.85)	
Ag Processing recovery	MIN(IF([Ag]<0.5, 0.1, 0.32493 + 0.25676 * LN([Ag])), 0.62)	
Cost Factors		
OP mining cost reference	C$ 3.09/t	
Additional OP mining cost	0.08 CAD/t/bench	
UG mining	32 CAD/t	
Milling (processing and G&A)	10.976	
Sale Factor		
Gold price	1650 USD/oz	
Copper price	3.5 USD/lb	
Silver price	21.5 USD/oz	
Other marketing terms	Refer to NSR calculation in section 16.5.1	

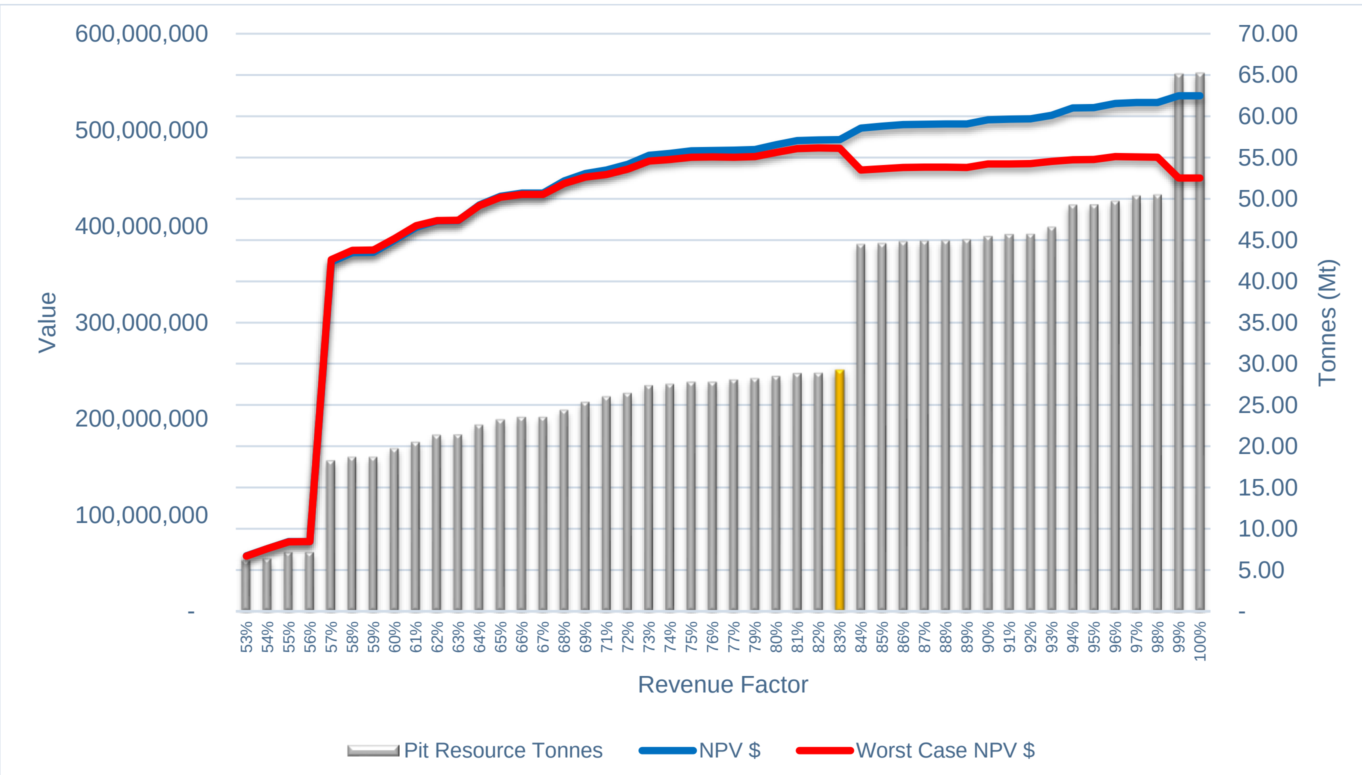
*This is a contractor-operated Open Pit mine for both Kwanika Central and South. The basis of open pit mining costs were derived from previous studies: "NI 43-101 Technical Report for the Kwanika Project, Preliminary Economic Assessment Update 2017" and "Kwanika Prefeasibility Study – WP3/WP9 Close-out Report - MMTS, 2019".

The first step in the process of evaluating the economic potential of the deposit was to undertake an open pit optimization analysis to assess the pit resource tonnage and grade and maximize the economical outcomes of the project.

The pit shell at Revenue Factor (RF) 83% was selected from various LG pit shells (Figure 16-17 below) to optimize the following goals:

- Maximum NPV in the pit
- Maximizing the NPV of the total project
- Providing the mill throughput during the pre-production years of Kwanika block cave mine.
- Maximum NSR value within open pit LOM

Figure 16-17: Optimization outcome for Central Pit



Source: Mining Plus, 2022.

16.5.2.2 Mining Loss, Dilution and SMU Optimization

The resource model was reblocked to an SMU of 10x10x10. The SMU size is considered optimal for the level of project study, the equipment proposed, and the nature of the mineral body.

A 2% external dilution factor is applied to account for mixing of mineralized material and waste during the mining process.

A 95% mining recovery is also applied to account for inefficiencies and operating challenges such as blasting losses, carry-back in truck boxes due to wet material, misdirected materials, and other unforeseen exceptions.

A detailed study is necessary to better understand and quantify the internal dilution. This analysis is recommended for the next phase of project development.

16.5.2.3 Central Pit Slope Angle

An overall slope angle in the range of 32 degrees to 44 degrees was used in the Central pit walls based on the Kwanika geotechnical sectors which were provided in the 2019 Internal Report study.

The pit walls are designed to be shallower in the south and west zones, and steeper from the north to south walls (Table 16-26).

Table 16-26: Overall Slope Angle in Central Pit

Sector*	Azimuth		Overall slope angle (Degrees)	Bench face angle (Degrees)	Berm width (m)
	From	To			
1	235	285	37	70	17.7
2	285	350	41	70	15.2
3	350	0	43	70	13.7
3	0	55	43	70	13.7
4	55	160	44	70	12.2
5	160	180	38	70	16.8
6	180	235	32	70	23.7

Source: Kwanika Prefeasibility Study – WP3/WP9 Close-out Report - MMTS, 2019

16.5.2.4 Central Pit Design

The open pit has been designed using 20 m bench heights and a 70-degree bench face angle. To meet the maximum overall slope angle in each geotechnical sector, the berm widths were calculated for each sector.

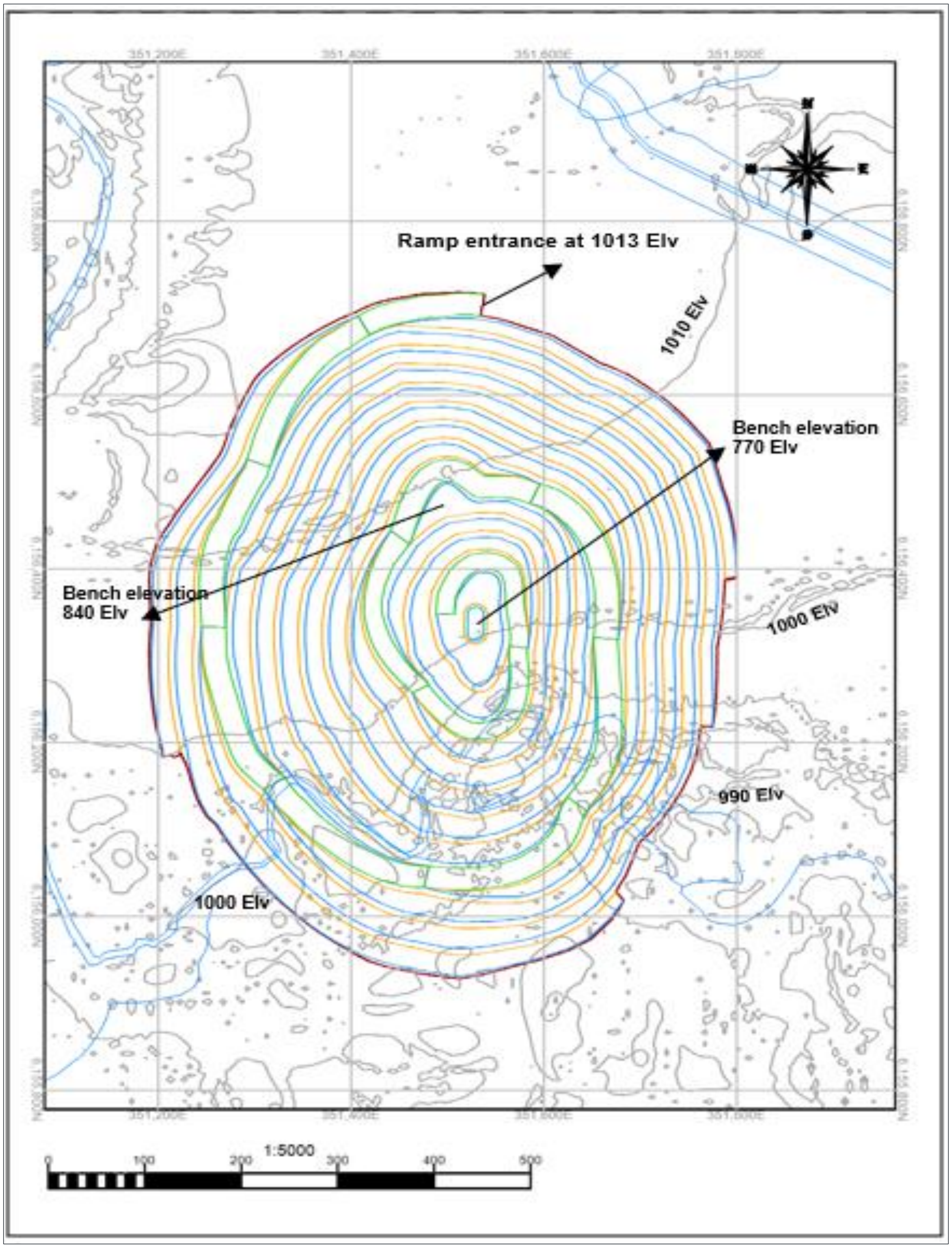
All in-pit ramps are designed to be suitable for the operation of the widest mining equipment being used. In general, in-pit ramps have been designed to be 25.2 m wide for two-way traffic with a maximum gradient of 10%, which includes provision for a safety berm. The in-pit haul ramp from the last 40 m to the pit floor is 18.9 m wide for single-way traffic.

The ultimate pit was designed utilizing Deswik software. The final pit design is around 780 m long in the north-south direction and up to 590 m in width. It extends from the top at 1015 m elevation to the pit floor at 770 m elevation. The highest elevation exposed by the final pit design is at 1015 m elevation. Mine inventories in the designed Central Pit are presented in Table 16-27. The final design for Central Pit is illustrated in Table 16-27.

Table 16-27: Central Pit Inventories

Total Mined Mt		Waste Mt	Overburden Mt	Pit resource (diluted & recovered) Mt	Au g/t	Ag g/t	Cu %	NSR $/t
84.62	25.18	29.99	29.45	Inferred= 0.13 Indicated= 9.47 Measured=19.85	0.29	1.16	0.32	36.74

Figure 16-18: Kwanika Centre Final Pit Design



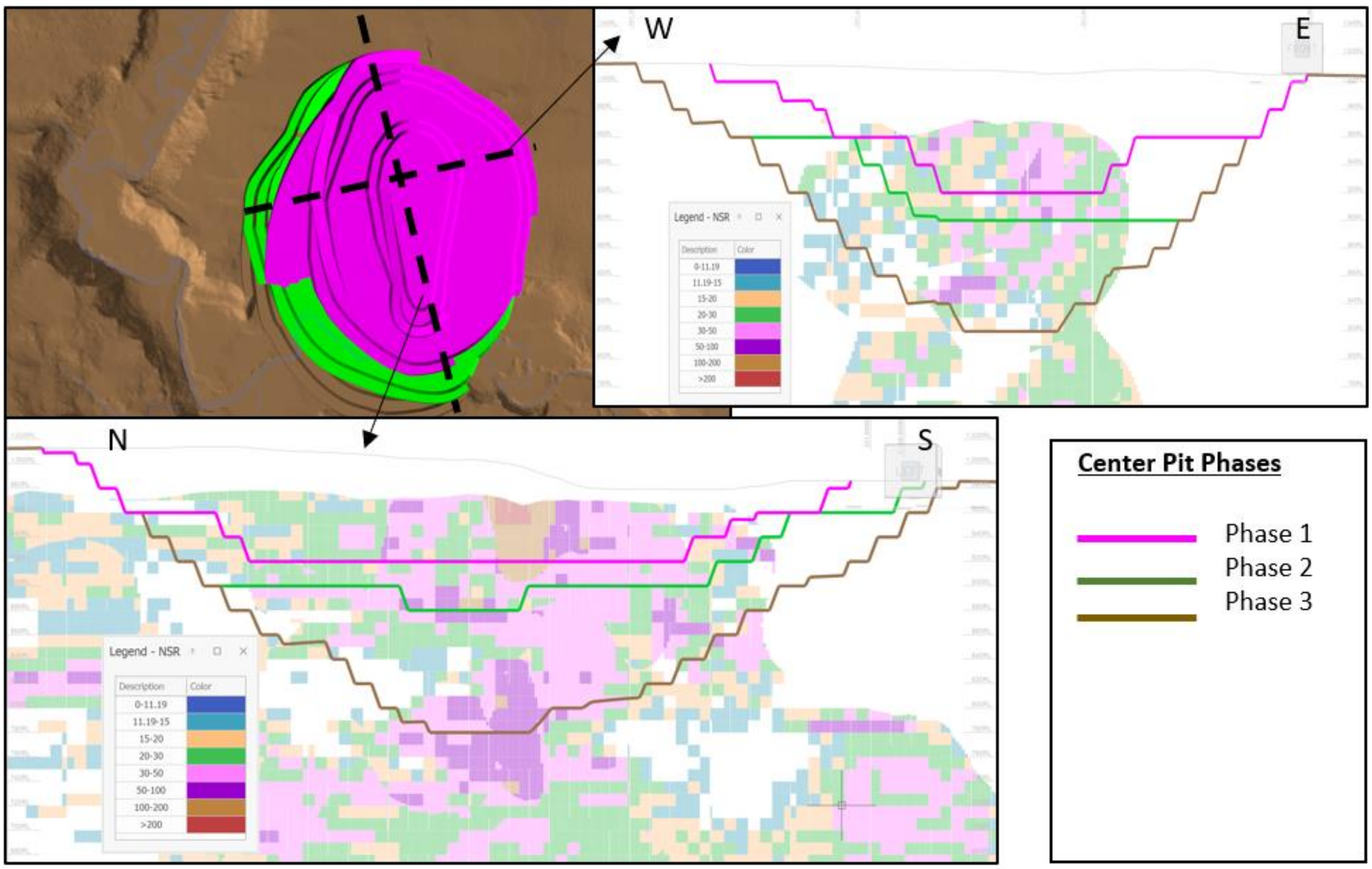
Source: Mining Plus, 2022.

16.5.2.5 Centre Pit Mine Phases

Three phases were defined for the Centre Pit to maximize the value of each phase considering the required material from each period of life of mine (Figure 16-19).

- **Phase 1:** The first phase aimed to provide the required waste rock for construction at year -1 and the high-grade material for Mill feed at Year 1.
- **Phase 2:** The second phase is mainly responsible for providing material for Mill feed at year 2.
- **Phase 3:** Final pit.

Figure 16-19: Central Pit Phases Plan View and Sections



Source: Mining Plus, 2022.

16.5.2.6 Central Pit Mine Production Schedule

The mine production schedule was produced using Deswik software, which links the block model and ultimate pit design with the required production criteria. A mine production schedule based on 10 m high benches was generated indicating annual mineral resource and waste tonnages and grades for one year of pre-stripping and four years of production.

16.5.3 Kwanika South Pit

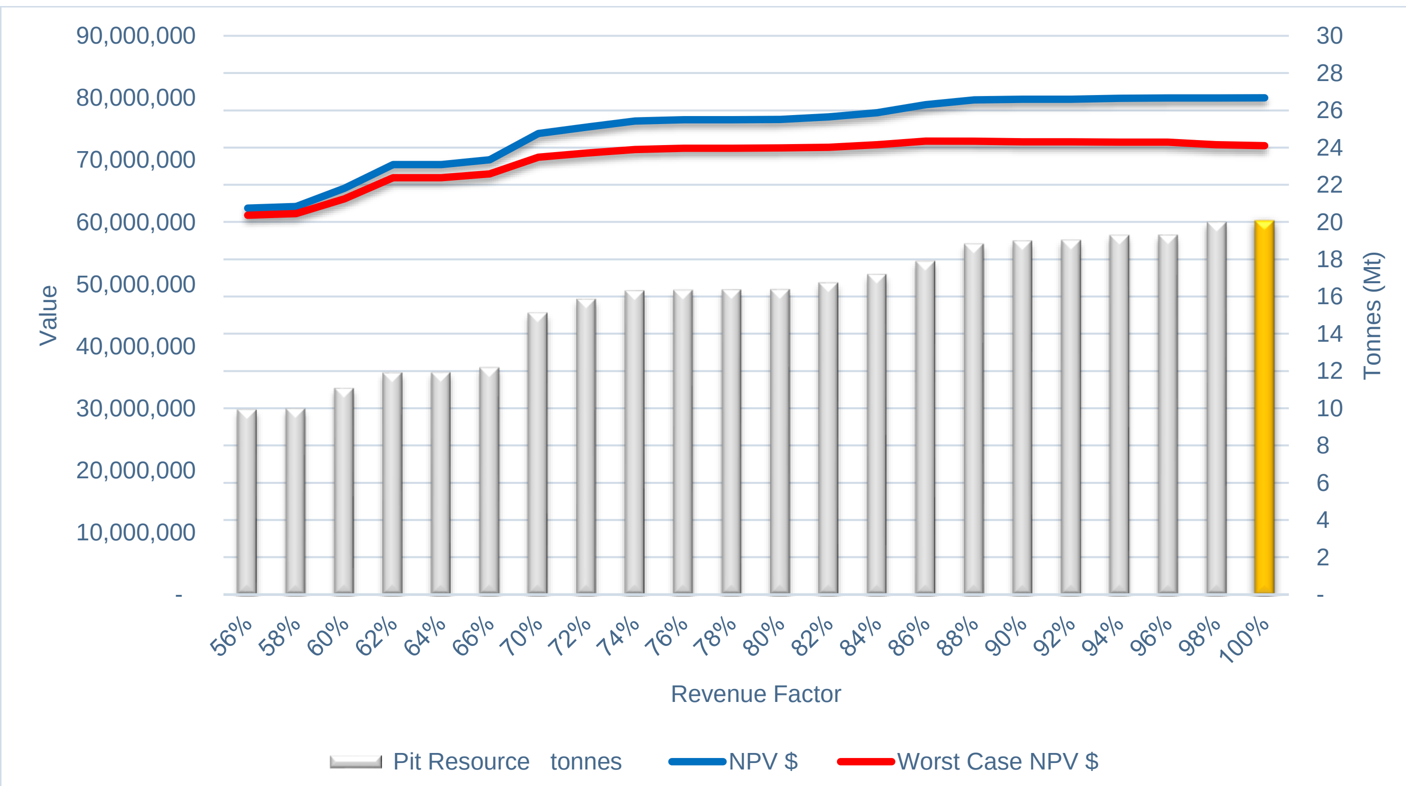
16.5.3.1 South Pit Optimization

The pit optimization for South Pit used the same input parameters as Central Pit, shown in Table 16-25, utilizing Datamine NPVS software. The pit shell at RF 100% was selected as the Kwanika South final pit to optimize the following goals:

- Maximum NPV in-pit
- Provide the maximum resource to supplement the Kwanika block cave with full mill throughput capacity in the last years of the project.

The optimization outcomes are presented in Figure 16-20.

Figure 16-20: Optimization outcomes for South Pit



Source: Mining Plus, 2022.

16.5.3.2 South Pit Mining Loss, Dilution and SMU Optimization

The resource model has a block size of 20x20x10; therefore, no further SMU analysis was conducted. Mining Plus recommends that a smaller block size is used in further studies, in order to better define the mineral body.

A 2% external dilution factor is applied to account for mixing of mineralized material and waste during the mining process.

A 95% mining recovery is also applied to account for inefficiencies and operating challenges such as blasting losses, carry-back in truck boxes due to wet material, misdirected materials, and other unforeseen exceptions.

A detailed study is necessary to better understand and quantify the internal dilution. This analysis is recommended for the next phase of project development.

16.5.3.3 South Pit Slope Angle

A geotechnical assessment has not yet been completed for Kwanika South. A benchmarking study was conducted based on nearby mines to determine the appropriate pit slope angles to use for this study. An overall pit slope angle of 45° was selected based on previous studies.

A targeted program of geotechnical borehole drilling, and analysis is recommended for the next phase of project development in order to better understand pit slope design criteria and overall pit slope angles in all sectors of the deposit.

16.5.3.4 South Pit Design

The open pit has been designed using a 20 m bench height and a 70-degree bench face angle with a minimum berm of 8 m per bench. All in-pit ramps are designed with the same parameters as Central Pit. The ultimate pit was designed utilizing Deswik software. The final design includes 3 separate pits, around covers a distance of1900 m in the north-south direction and up to 450 m in width. Each pit geometry is as follows:

South Pit1: Located in the northern zone of the South Pit area. The design is around 420 m long in the north-south direction and up to 380 m in width. It extends from the top at 1009 m elevation to the pit floor at 822 m elevation.

South Pit2: Located in the centre zone of the South Pit area. The design is around 439 m long in the north-south direction and up to 334 m in width. It extends from the top at 1024 m elevation to the pit floor at 842 m elevation.

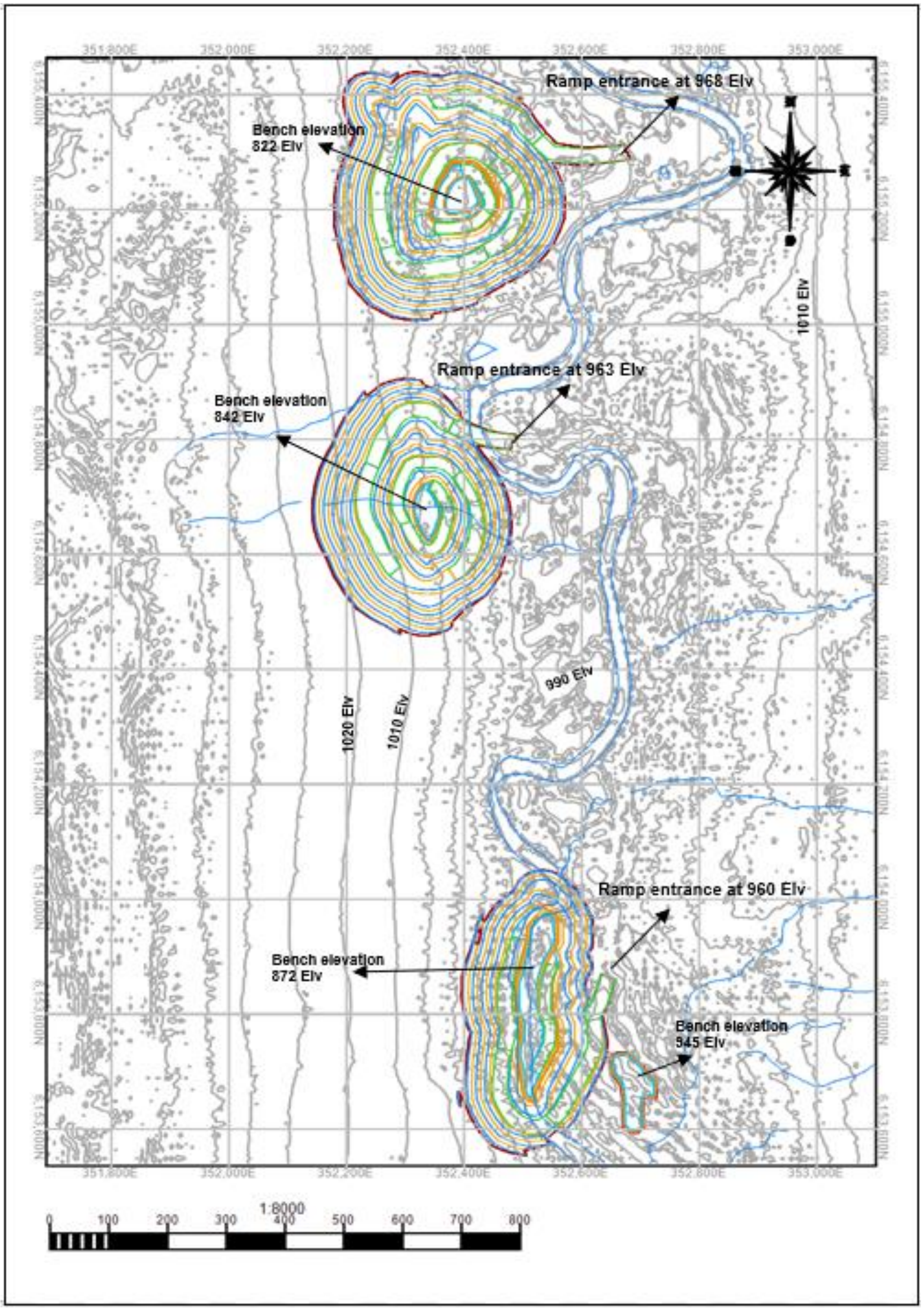
South Pit3: Located in the southern zone of the South Pit area. The design is around 490 m long in the north-south direction and up to 213 m in width. It extends from the top at 990 m elevation to the pit floor at 872 m elevation.

The mine inventories in the designed South Pit are presented in Table 16-28. The final design for South Pit is illustrated in Table 16-28.

Table 16-28: South Pit Inventories

	Total Mined (Mt)	Waste (Mt)	Overburden (Mt)	Pit resource (diluted & recovered) (Mt)		Au (g/t)	Ag (g/t)	Cu (%)	Mo (g/t)	NSR ($/t)
South pit	50.76	20.85	10.83	19.05	Inferred=19.05	0.07	1.68	0.29	98.48	23.40

Figure 16-21: Kwanika South Final Pit Design

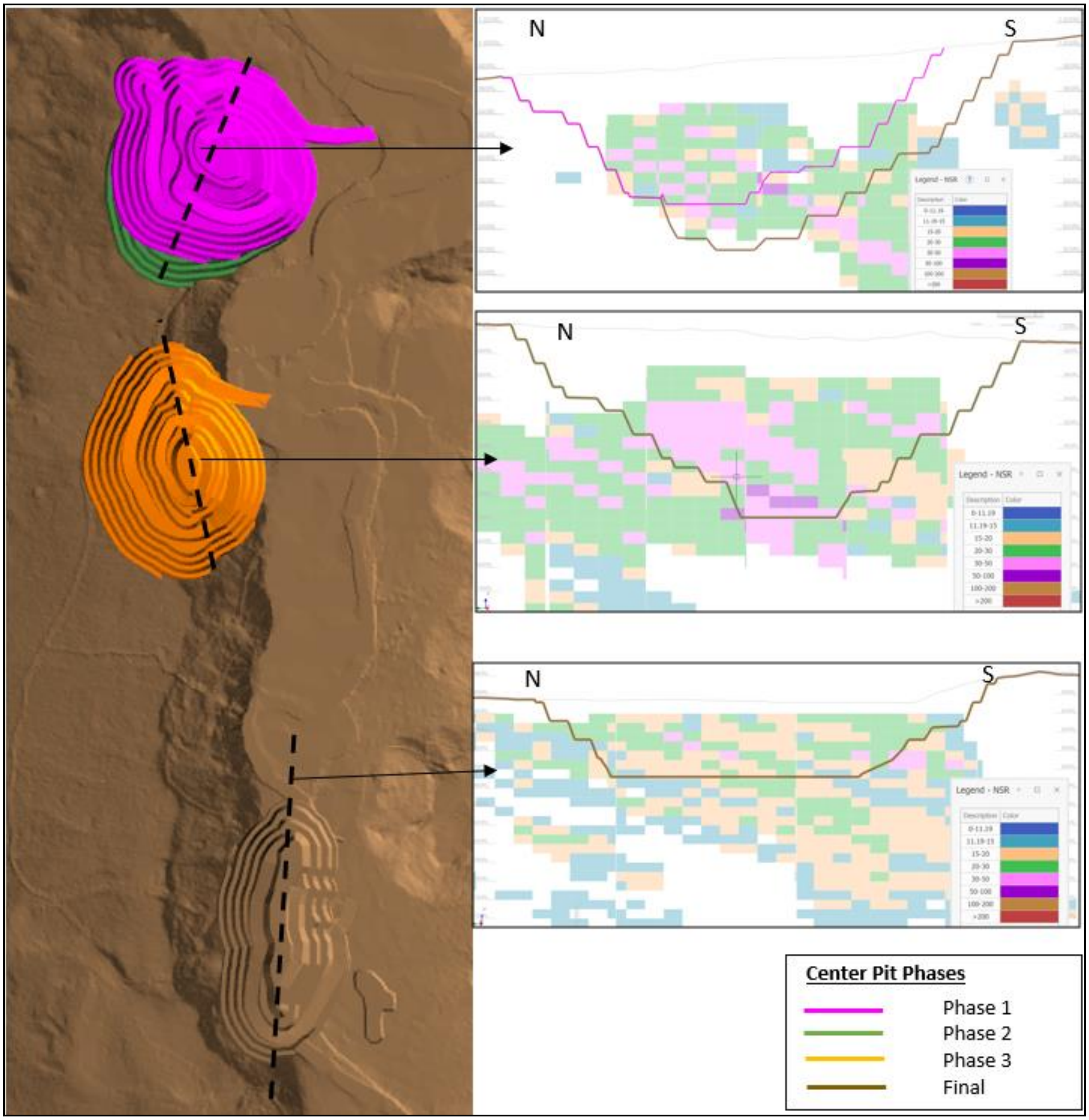


Source: Mining Plus, 2022.

16.5.3.5 South Pit Mine Phases

Four phases were defined for the South Pit to maximize the value of each phase considering the required material from each period of life of mine. These phases have the potential for backfilling with mine waste to avoid the long distance of the haul cycle to the waste dump facility (Figure 16-22).

Figure 16-22: South Pit Phases Plan View and Sections



Source: Mining Plus, 2022.

16.5.3.6 South Pit Mine Production Schedule

The mine production schedule was produced using Deswik software, which links the block model and ultimate pit design with the required production criteria. A mine production schedule based on 10 m high benches has been generated indicating annual mineral resource and waste tonnages and grades for eight years of production.

The main goals for the South Pit mine plan are:

1. Provide the minimum resource which is required to supplement the block cave production to achieve 22 kt/d from years 9 to 12.
2. Provide a smooth material movement over the production years to avoid mobilization and demobilization costs for equipment.

16.5.4 Mineralized Rock Stockpiles

Some stockpiling of low-grade material will occur over the life of mine. The block cave mine will send low-grade material to the stockpile in year 3. Also, low-grade material will be sent to the stockpile from the Kwanika Central Pit. This resource will be reclaimed later in years 5 to12 to fulfill the mill feed requirements. The stockpile was designed to fit a capacity of 5 Mt of rock.

16.5.5 Waste Rock and Overburden Storage Facilities

The waste rock and overburden storage facilities are located to the North of the Kwanika Central and South deposits. A slope angle of 32 degrees was applied, which is assumed to consider berms and batter angles. The waste dump was designed to fit 41 Mt of rock, and the overburden stockpile was designed to fit 46 Mt of overburden.

16.5.6 Open Pit Operations

The project is conceived as a contract mining operation. The reference contractor mining cost from the 2017 PEA includes the responsibility for all mining areas including direct mining, mine maintenance, and dewatering activities. The costs from the 2017 PEA have been inflated to the year 2022.

Mine technical services, such as geology, engineering, and management will be the owner’s responsibility and are captured in the general mine expense area. In this regard, an extra G&A cost is added to the inflated contractor mining cost.

16.5.7 Open Pit Equipment

Considering the maximum total material movement of 21 Mt/y from the open pit, a large capacity operation is designed, and large-scale equipment is specified. The major open pit fleet will consist of 15 m^3 diesel-hydraulic shovels matched with 136t trucks.

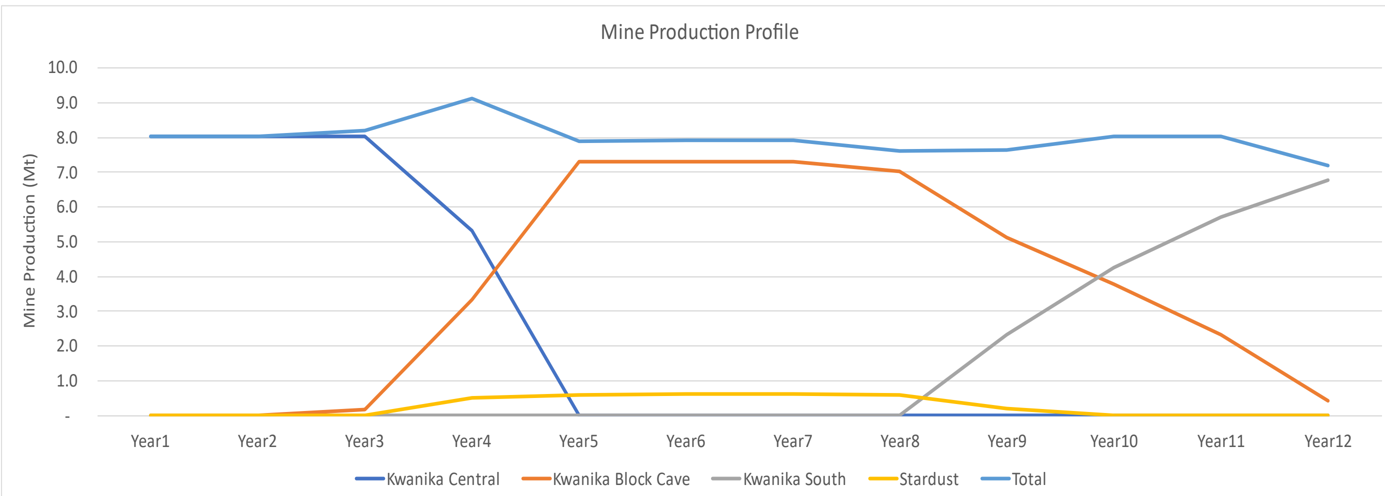
It is assumed the open pit mining will be by contractor and a contractor’s uplift has been applied to the mining costs of this fleet.

16.6 Combined Mine Production Plan

The combination of 2 open pits and 2 underground mines provide 11 years of mill feed at a full capacity of 22 kt/d and 1 year of production with 93% of the mill's full capacity. The mine production main goal is to maximize the NPV of the operation.

Figure 16-23 shows the Production Plan for the Project. Tonnages are ROM from the resources.

Figure 16-23: Mine Production Plan



Source: Ausenco, 2023

Central Pit provides the required waste rock for construction at year -1 and then provides the full mill capacity from year 1 to 3. At year 4, Central Pit provides 50% of mill feed and the rest is provided by Stardust and Block cave. From years 5 to 8 almost 90% of the mill feed provides by Block Cave and the rest comes from Stardust and the stockpile. Then at year 9 Kwanika South Pit ramp-up and it goes to year 12 the Block cave contribution to mill feed decreases and the South Pit contribution increase (Table 16-29).

Table 16-29: Project Timetable and Mines Contribution to The Mill Feed

Production Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Processing Plant			100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	91%
Kwanika Centre Pit			100%	100%	100%	53%								
Kwanika Block cave						41%	91%	91%	91%	87%	64%	47%	29%	6%
Stardust						6%	8%	8%	8%	7%	2%			
Stockpile							2%	1%	1%	5%	5%			1%
Kwanika South Pit											29%	53%	71%	93%
pre-production year	Production year	Mine development	Mine production (% contribution to mill feed)											

17 RECOVERY METHODS

17.1 Overview

The project flowsheet has been selected based on recovery methods required for processing the project material, supported by preliminary testwork and financial evaluations. The basis of the selected design is presented in Section 17.2. The plant is designed to process, on average, 22,000 t/d (8.03 million t/y) of mineralized material. Mined material is crushed, ground, and processed by bulk rougher flotation. Rougher flotation concentrate is then reground and upgraded by cleaner flotation to produce a copper concentrate which is then dewatered. Flotation tailings are sent to a tailings dam.

The flowsheet has been developed from engineering studies and recent testwork. An overall process flow diagram and mechanical equipment list have been developed. Kwanika process plant includes the following unit processes and facilities:

- primary crushing
- crushed mineralized material stockpile and reclaim
- gyratory crushing
- SAB grinding circuit
- copper flotation comprises rougher flotation, concentrate regrind, and three-stage cleaning flotation and cleaner scavenger flotation
- concentrate thickening
- concentrate thickening and filtration
- concentrate load out and storage
- tailings thickening, filtration and tailings.
- reagents storage and distribution (including lime slaking, flotation reagents, water treatment and flocculant).

17.2 Process Design Criteria

The project flowsheet and unit operations have been selected based on preliminary testwork and financial evaluations. Unit operations used to build the plant flowsheet are standard technologies widely used in copper flotation plants. The basis of the selected design and recovery data are presented in Table 17-1. Figure 17-1 shows the overall process flow diagram.

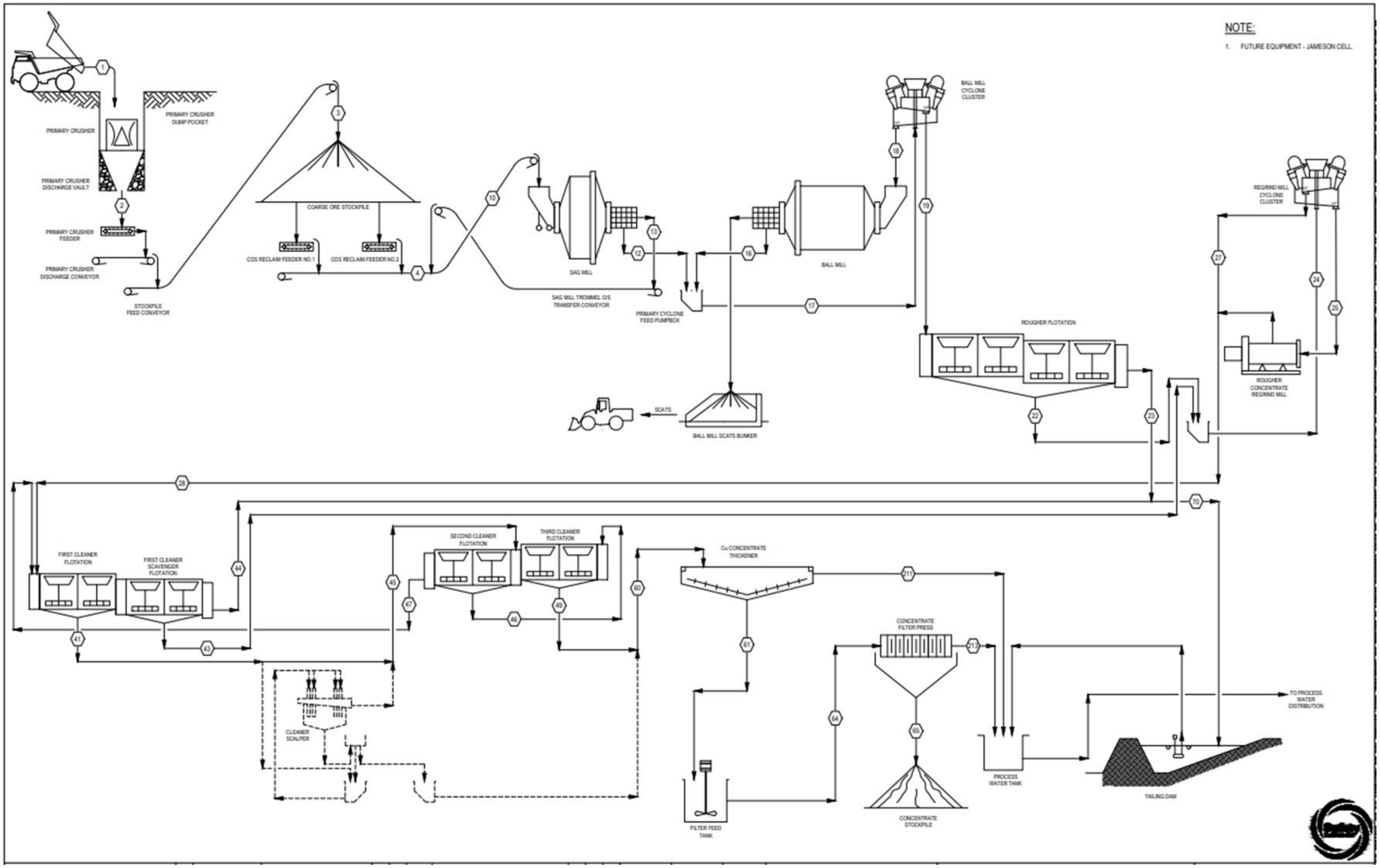
Table 17-1: Process Design Criteria

Design Parameter	Units	Value
Plant Operating Summary		
Total Material Milled	Mt	94
Plant design capacity	Mt/y	8.03
Plant Operating Basis		
Crushing Plant		
operating hours per year, at availability	%	75
Grinding/Flotation Plant		
operating hours per year, at availability	%	92
Production		
Plant Capacity		
Annual Processing Rate	t/a (dry)	8.03
Daily Processing Rate	t/d	22,000
Crushing Plant, nominal	t/h	1,222
Grinding/Flotation Plant, nominal	t/h	996
Head Grade		
Head Cu Grade, Design	%	0.8
Head Au Grade, Design	g/t	0.8
Comminution Characteristics		
Bond crushing work index Design	kWh/t	12.8
Bond Rod Mill Work Index Design	kWh/t	19.2
Crushing Plant		
Primary Crusher		
Crusher Type		Gyrator Crusher
Number of Crushers		1
Processing Rate, nominal	t/h	1,222
Feed Top Particle Size	mm	1,000
Product Particle Size, P_{80}	mm	73
Grinding Circuit		
Circuit Description		SAB
Grinding circuit capacity, design (dry)	t/h	996
Grinding circuit product size, P_{80}	µm	100
SAG Mill		

Design Parameter	Units	Value
Type	-	SAG
SAG mill discharge pulp density	% solids (w/w)	75
SAG mill ball charge	% v/v	12
SAG mill total load	% v/v	26
Ball Mill		
Type	-	Overflow Ball Mill
Number of ball mills	-	1
Circuit	open / closed	closed
Circulating load, nominal	%	300
Ball mill ball charge	% v/v	29
Classifying Cyclones		
Overflow density	% solids (w/w)	35
Underflow density	% solids (w/w)	75
Copper/Gold Flotation		
Circuit Description	Rougher + Regrind + 3-stage Cleaner	
Rougher Flotation		
Cell type	-	Tank
Number of banks	-	1
Rougher flotation pH	-	10
Residence time, design	min	14
Regrind Circuit		
Solids Feed Flow Rate Nominal	t/h	135
Feed Size, F80	µm	75
Product Size, P_{80}	µm	25
Regrind Classification		
Underflow Solids Density	%	55
Mass Split to Undersize	%	54
Regrind Mill		
Mill Type		Regrind Mill
Number of Mills		1
Cleaner Flotation		
First Cleaner Flotation		
Cell type	-	Tank

Design Parameter	Units	Value
Number of banks	-	1
Residence time, design	min	7
First Cleaner Scavenger Flotation		
Cell type	-	Tank
Number of banks	-	1
Residence time, design	min	7
Second Cleaner Flotation		
Cell type	-	Tank
Number of banks	-	1
Residence time, design	min	5.5
Final Cleaner Flotation		
Cell type	-	Tank
Number of banks	-	1
Copper Concentrate Dewatering		
Thickening		
Thickener Type		High-Rate
Number of Thickeners		1
Thickener U/F Solid	%	60
Concentrate Stock Tank		
Feed Solids Density	%	60
Pulp Feed Flow Rate	m^3/h	26
Concentrate Stock Tank Retention Time	h	24
Filtration		
Filter Availability	%	75
Filter Cake Moisture	%	8
Filter Type		Filter Press
Number of Filters		1

Figure 17-1: Process Flowsheet



Source: Ausenco, 2022.

17.3 Process Description

17.3.1 Overview

The Process Plant receives mineralized material from the deposit at the crushing plant by haul truck. Material is dumped directly into the gyratory crusher and crushed material is conveyed to a stockpile. Two reclaim feeders transfer crushed material from the stockpile to the SAG feed conveyor. The crushed material is processed in a semi-autogenous grinding (SAG) mill. After exiting the SAG mill, material passes across a trommel screen and oversize material is returned to the SAG mill via a conveyor. SAG trommel undersize material discharged from the SAG mill is sent to a pumpbox where it is combined with ball mill discharge. This combined material is sent to a cyclone pack for classification. Cyclone underflow reports to the ball mill and cyclone overflow advances to rougher flotation.

The rougher flotation circuit produces a bulk copper concentrate. The tailings from the rougher flotation circuit are sent to the TSF. The rougher copper concentrate reports to a pumpbox that sends material to a pack of regrind cyclones. Overflow from the regrind cyclones advances directly to the cleaner flotation circuit. The cyclone overflow reports to the regrind ISA mill where it is reground before entering the cleaner flotation circuit. The regrind cyclone overflow and regrind ISA mill product are combined in the first stage of cleaner flotation. Tailings from the first cleaner are sent to cleaner scavenger flotation. Concentrate from cleaner scavenger flotation is recirculated to the regrind circuit; tailings are sent to the tailings pond. First cleaner flotation concentrate advances to second cleaner flotation, and second cleaner flotation concentrate is sent to third cleaner flotation. The third cleaner produces a final copper concentrate for dewatering. Second cleaner tailings are recycled to the first cleaner flotation feed. Third cleaner tailings are recycled to the second cleaner flotation feed.

17.3.2 Crushing Plant

ROM material is dumped directly into the gyratory crusher by haul truck. The crusher reduces the material from a P_{80} of 528 mm to a product with a P_{80} of 73 mm. A crusher discharge apron feeder, discharge conveyor, and stockpile feed conveyor move crushed material onto the stockpile which provides a buffer between circuits with different availabilities.

The major equipment in this area include:

- ROM feed bin
- mobile rock breaker
- primary gyratory crusher
- crusher discharge feeder
- crusher discharge conveyor
- stockpile feed conveyor.

17.3.3 Crushed Material Handling

Two apron feeders reclaimed crushed material from the stockpile and deposit it onto the SAG mill feed conveyor which transfers material to the SAG mill. A weightometer is used to control the flow of material into the SAG mill.

The major equipment associated with this area include:

- two crushed product apron reclaim feeders
- SAG mill feed conveyor
- SAG mill feed weightometer.

17.3.4 Grinding and Classification

Process water and crushed mineralized material enters the SAG mill. The 9.75 m ID x 4.57 m SAG mill has an installed power of 9 MW. The processed slurry is discharged across a trommel screen and is collected via gravity into a pumpbox. The trommel screen has a passing size of 12 mm. Oversized material is recirculated back to the SAG mill feed, and undersized material is pumped to a cyclone cluster for classification. Cyclone overflow advances to the rougher flotation circuit. Cyclone underflow reports to the 8.53 m ID x 11.73 m ball mill. The ball mill has an installed power of 18 MW. The size of the material passing through the ball mill is reduced to a product with a P_{80} of 100 µm. The discharge from the ball mill is collected in the same pumpbox that SAG mill undersize material reports to.

The major equipment in this area include:

- SAG mill – 9.75 m ID x 4.57 m, 9 MW
- SAG mill trommel screen – 12 mm opening
- hydrocyclone pack
- cyclone feed pump
- SAG grinding media handling system
- SAG mill liner handling system
- ball mill – 8.53 m ID x 11.73 m, 18 MW
- ball mill grinding media handling system
- ball mill liner handling system.

17.3.5 Rougher Flotation

Underflow slurry from the grinding circuit cyclones reports to the rougher flotation circuit where it is conditioned with lime, A208 collector, 3418A promotor, and MIBC frother. The rougher flotation circuit consists of a single bank of five 305.4 m^3 volume tank cells. The nominal density in the feed pulp is 35% solids by weight. Process air is blown into each cell to create froth bubbles. Copper sulphide minerals adsorb onto the bubbles and float to the top of the tanks, where the froth overflows into concentrate launders. The tailings from each rougher cell move into the next subsequent cell and the flotation process continues. Tailings from the final rougher cell are pumped to the tailings pond. The rougher froth concentrate is sent to the regrind area.

The major equipment in this area include:

- Five 305.4 m^3 conventional rougher flotation tank cells in series

17.3.6 Rougher Concentrate Regrind

The pumpbox receiving the rougher froth concentrate also receives cleaner scavenger flotation tailings. The combined slurry is pumped to a cyclone pack for classification. Cyclone overflow reports directly to the cleaner flotation circuit. Cyclone underflow is discharged to the 2.2 MW Regrind Mill where the P_{80} of the material is reduced to 25 µm. Product from the Regrind Mill is sent to the cleaner circuit along with the regrind cyclone overflow.

The major equipment associated with this area include:

- Hydrocyclone cluster
- 2.2 MW Regrind Mill.

17.3.7 Cleaner Flotation

Slurry from the regrind circuit is pumped to the first cleaner flotation circuit where it is conditioned with lime, A208 collector, 3418A promotor, and MIBC frother. The conditioned slurry is combined with second cleaner flotation tails and floated in the first cleaner circuit. The first cleaner circuit consists of three 110.4 m^3 conventional tank cells. Tailings from the first cleaner circuit are fed to the cleaner scavenger flotation circuit. The cleaner scavenger circuit is comprised of four 58.1 m^3 conventional tank cells. Concentrate captured in the cleaner scavenger circuit is recirculated to the regrind area. Tailings produced by the cleaner scavenger circuit are sent to the tailings pond.

The concentrate from the first cleaner stage advances to second cleaner flotation. The second cleaner circuit consists of three 23.3 m^3 conventional tank cells. The tailings from this circuit is recirculated to the first cleaner stage. The concentrate is pumped to the third cleaner flotation circuit.

The third cleaner flotation circuit is made up of three 23.3 m^3 conventional tank cells. Tailings from this circuit are recirculated to the adjacent second cleaner stage flotation cells. The third cleaner concentrate is pumped to the concentrate thickener.

The major equipment for this area include:

- three 110.4 m^3 conventional tank flotation cells
- four 58.1 m^3 conventional tank flotation cells
- six 23.3 m^3 conventional tank flotation cells.

17.3.8 Concentrate Handling

The final concentrate from the cleaner circuit is pumped to a 14 m diameter high-rate copper concentrate thickener. The density of the thickener underflow is 60% solids by weight. This underflow is pumped to an agitated concentrate storage tank which has a nominal compacity of 24 hours' slurry volume. Thickener overflow is captured and sent to the process water tank.

The thickened slurry from the storage tank is pumped to the concentrate filter at a rate of 13 m^3/h. The concentrate filter produces a final product with a targeted moisture of 8% by weight. Filtrate from this dewatering process is pumped back to the process water tank. The final dewatered copper concentrate is discharged to the stockpile for weighing, loadout by front-end loader, and shipment. The stockpile has a working capacity of 12 hours.

The major equipment associated with this process include:

- one 14 m diameter high-rate copper concentrate thickener
- one filter press with 13 m^3/h capacity.

17.3.9 Tailings Handling

The final tailings are a combined product of the rougher flotation tailings and cleaner scavenger flotation tailings. The majority of this combined slurry is pumped from the mill to the tailings booster station then the tailings pond. Once the underground deposits are being actively mined, 20% of this slurry is planned to be sent back underground as paste. Process water reclaimed from the tailings facility is pumped back to the process water tank.

The major equipment associated with this area include:

- two tailings pumps each with a capacity of 1,205 m^3/h
- two tailings booster pumps each with a capacity of 1,205 m^3/h
- three tailings reclaim pumps each with a capacity of 584 m^3/h.

17.3.10 Reagents Handling and Storage

17.3.10.1 Lime

Lime is received on site in the form of dry powder from bulk road tankers. The lime will be mixed with water to create a slurry that is distributed to the grinding, rougher, and cleaner circuits. Annual consumption of lime is estimated at 12,286 t/y.

17.3.10.2 A208 Collector

A208 collector is delivered to site as a liquid in bulk isotainers. Undiluted collector is distributed using dosing pumps to the rougher flotation circuit. Annual consumption of A208 collector is estimated at 112 t/y.

17.3.10.3 3418A Promoter

3418A promoter is delivered to site as a liquid in bulk isotainers. Dosing pumps deliver this reagent without dilution to the rougher and cleaner flotation circuits. Annual consumption of 3418A promoter is estimated at 132 t/y.

17.3.10.4 MIBC Frother

Methyl Isobutyl Carbinol (MIBC) is received on site in liquid form in bulk isotainers. Dosing pumps will deliver this reagent without dilution to the rougher and cleaner flotation circuits. Annual consumption of MIBC frother is estimated at 385 t/y.

17.3.10.5 Flocculant

Flocculant is received on site as a dry powder in bulk bags. The powder is mixed with water to a concentration of 0.1-0.2 g/L. The solution is then pumped to the concentrate thickener. Annual consumption of flocculant is estimated at 2 t/y.

17.3.11 Consumables

17.3.11.1.1 Grinding Media

The media consumption of SAG mill and ball mill is 8,841 t/y.

17.3.11.1.2 Liners

The consumption of crusher liners is three sets per year, and the consumption of SAG mill liner is 1,205 t/y. The ball mill liner consumption is 1,124 t/y.

17.3.12 Plant Services

17.3.12.1 Process Water

Process water is recovered from the concentrate thickener overflow, concentrate filter filtrate, and the tailings storage facility for reuse. The total process plant water demand is 1,852 m^3/h. Around 47 m^3/h of water is recycled from the concentrate thickener, 8 m^3/h on the concentrate filter press, 1,461 m^3/h is reclaimed from the TSF, and 336 m^3/h from the raw water supply source.

17.3.12.2 Raw Water

Raw water will be pumped from Kwanika Creek into the raw water tank, which has a live capacity of 24 hours. The raw water tank will also serve as the source for the processing plant's distribution of fire water, potable water, and gland seal water. 336 m^3/h of raw water makeup is required for the process plant.

17.3.12.3 Air Services

Plant air compressors supply air at 750 kPa to various processing plant equipment as needed. An air dryer is used to remove moisture before use in instrumentation. There are dedicated air compressors for concentrate filter, and low power air compressors for the flotation cells.

17.3.12.4 Power

The installed power for the process plant is 39,651 kW and the operating load is 27,756 kW. Further discussion on the power requirements are included in Section 18.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Infrastructure at the Kwanika-Stardust project includes on-site infrastructure such as civil, structural, and earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, power supply, piping, camp and tailings storage facility. The site infrastructure will include:

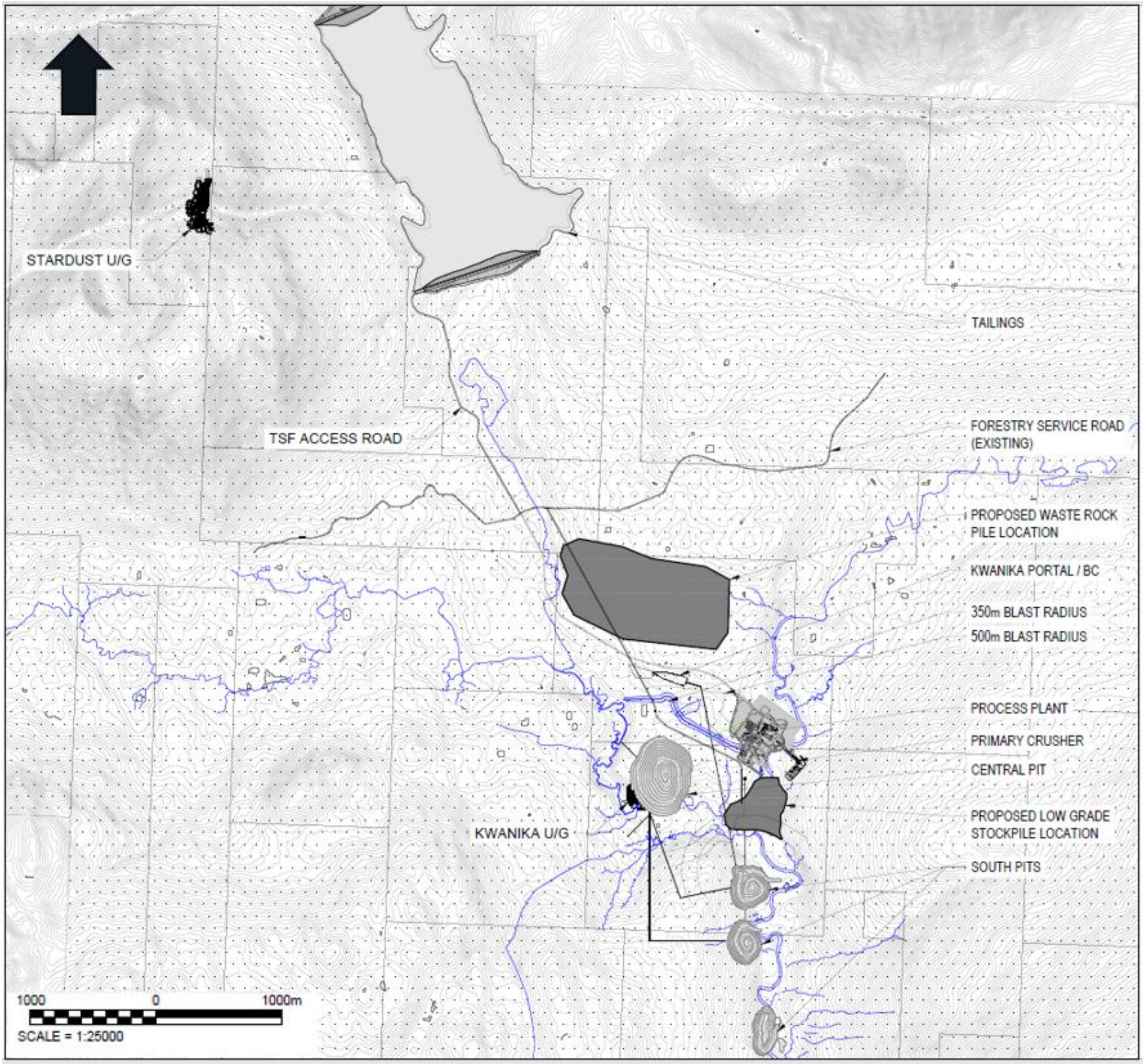
- mine facilities, including mining administration offices, a mine fleet truck shop and wash bays, a mine workshop, and a mine water treatment plant
- common facilities, including an entrance/exit gatehouse, a security/medical office, overall site administration building, potable water and fire water distribution systems, compressed air, power generation and distribution facilities, diesel reception and combustion plants, communications area, and sanitation systems
- process facilities, housed in the processing plant, including crushing, grinding and classification, flotation, product regrind, concentrate handling, thickening, dewatering, and filtration, reagent mixing and distribution, assay laboratory and process plant workshop and warehouse
- other infrastructure includes the on-site man-camp, TSF and WRSF.

The overall site layout was developed using the following criteria and factors:

- The facilities described above must be located on a site within the Kwanika-Stardust project boundary.
- The location of the process plant must be close to Kwanika open pit and underground mine which is the major source of feed, to reduce haul distance but outside of the 500 m blast radius.
- The location of the WRSF must be close to the open pits to reduce haul distance.
- The location of the primary crushing and ROM stockpile must be close to the Kwanika deposits to reduce haul distance.
- The TSF should be located at a site that takes advantage of sloped natural terrain to adequately drain entrained water and reduce earthworks, concrete, and structural development if possible.
- The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.

The Kwanika-Stardust project layout is shown in Figure 18-1.

Figure 18-1: Kwanika-Stardust Project Overall Site Layout



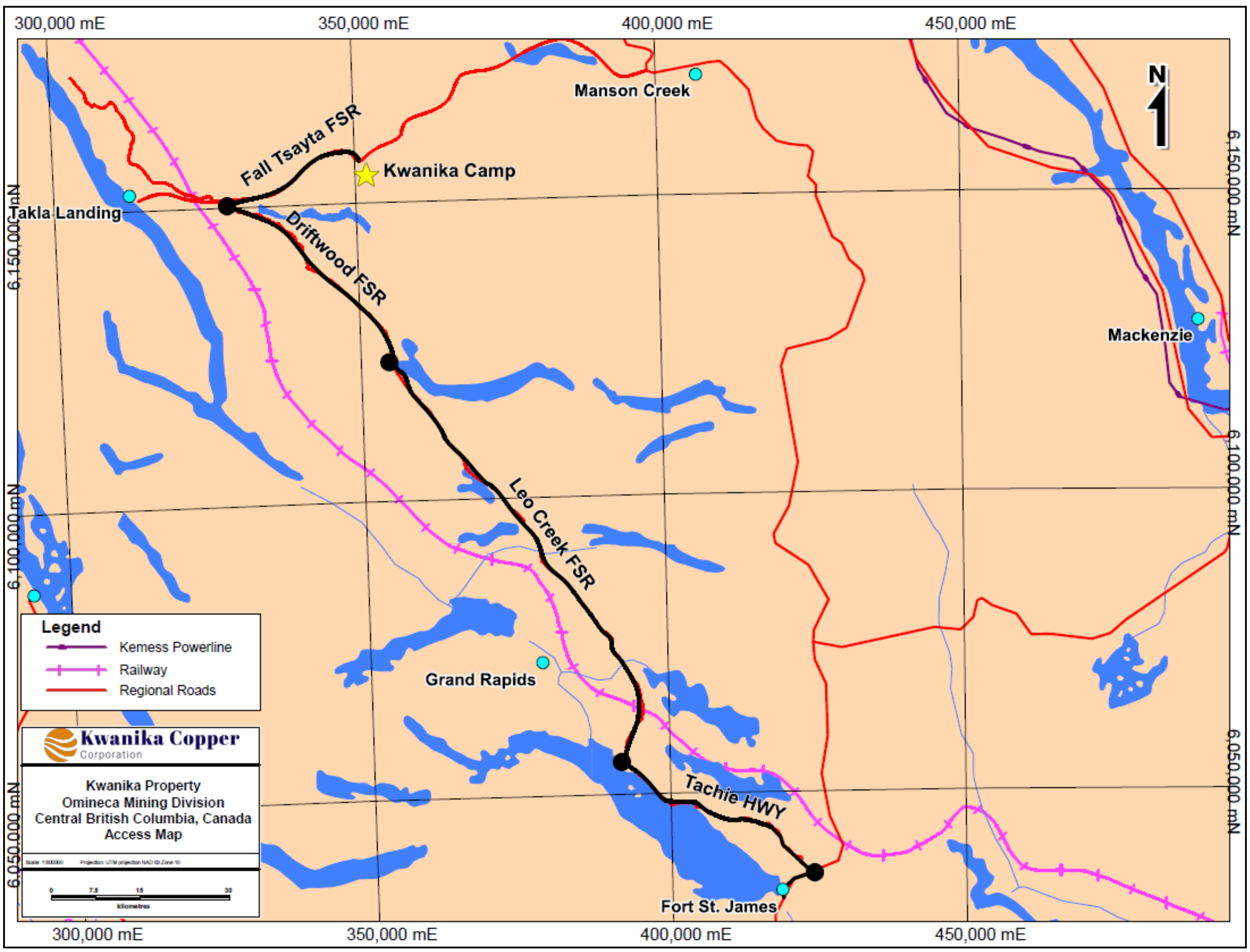
Source: Ausenco, 2022

18.2 Off-site Infrastructure (WBS 5000)

18.2.1 Site Access

The Kwanika-Stardust project is located 140 kilometres northwest of Fort St. James. Surface access to the property is now provided via an existing FSR between Fort St. James and Tsayta Lake Road (see Figure 18-2). In addition, around a 26.5-kilometre-long Tsayta Lake Road will be improved to meet operational requirements and allow for the delivery of bulk freight by tractor-trailer units; these improvements will include widening the road from the current 4-metres width to 19-metres.

Figure 18-2: Site Access



Source: Kwanika Copper Corp., 2017

18.2.2 Water Supply

Freshwater will be sourced from wells and Kwanika Creek. The water will be transported through pumps. Around 400 m of an insulated pipeline will be installed from Kwanika Creek to the processing plant where freshwater tanks will be located. This water will be the source of potable water on site, used for the building facilities and the process plant.

18.2.3 High Voltage Power Supply

Permanent electrical power is provided by a transmission line from the Kemess Power Line, which is 75 km away from the Kwanika-Stardust project to the site's substation. The project will connect to B.C. Hydro's electrical grid, and this assumes the right-of-way for transmission lines and the estimated alignment and design. The 230-kV transmission line will connect to a substation at the site before being stepped down to 25 kV for distribution to different power requirements across the project site.

18.2.4 Logistics

The concentrate will be trucked from the project site to Mackenzie, where it will be loaded in lots onto bulk material carriers and then transferred onto railcars of the CN Railway to port storage facilities at Vancouver Wharves in North Vancouver. The concentrate will subsequently be transported by sea to clients.

18.3 On-site Infrastructure (WBS 4000)

18.3.1 Site Preparation

The infrastructure area will be cleared, and the topsoil will be removed before construction. Drains, safety bunds, and backfilling with granular material and aggregates for road construction are all elements of the initial site development.

Site civil work includes design for the following infrastructure:

- roads for light vehicles and heavy equipment
- access roads
- topsoil and overburden stockpile area
- mine facility platforms and process facility platforms
- ROM stockpile area
- WRSF area
- Water Management Facilities, ditches, and drainage channels
- TSF area.

18.3.2 On-site Roads

The project site has unpaved roads connecting the access road to the gatehouse. In addition to the existing roads on site, new roads will be constructed to link the gatehouse to the administration building, from the process plant to the TSF, and from the access road to the magazine.

18.3.3 Fuel

The diesel storage facility consists of five bulk storage tanks. Each tank has a 100,000 L of capacity, for a total storage capacity of 500,000 L, which represents around two weeks of storage capacity.

18.3.4 Mining Infrastructure

18.3.4.1 Truck Shop/Wash (WBS 1200)

The truck shop/wash is a pre-engineered building with a concrete floor, overhead crane, and overhead doors with fire protection and alarm systems. There will be a total of four maintenance bays. Two maintenance bays will be assigned to preventative maintenance, one will be for corrective maintenance, and the last bay will be multipurpose. Additionally, a single welding bay and truck wash will be located at the front of the truck workshop building.

18.3.4.2 Explosives Magazine (WBS 1200)

The explosive magazine is a modular building that will be shared between both the open pit and underground operations. The magazine stores boosters, detonators, and packaged explosives. The magazine sits on a 20 m x 30 m pad and is sized for 6,000 kg of storage capacity. The magazine pad is kept separate from the bulk emulsion pad and other infrastructure.

18.3.4.3 Mine Office (WBS 1200)

The mine office is a modular building shared by both open pit and underground operations. The building is sized to accommodate 75 people. 45 for underground operations and 30 for open pit operations.

18.3.4.4 Mine Dry Facilities - Kwanika Block Cave (WBS 1400)

The mine dry facilities for the Kwanika Block Cave have the capacity for 140 men and 50 women. The men's area has 140 lockers, washrooms, and showers. The women's area has 50 lockers, washrooms, and showers.

18.3.5 Process Plant Infrastructure

18.3.5.1 Plant Warehouse/Shop (WBS 4600)

The plant warehouse/shop is a pre-engineered with concrete floor, overhead doors, fire protection, and alarm systems. This building will be used for general storage, to store equipment spares for the process plant, to maintain and store light vehicles assigned to the plant, and repair and maintain process plant equipment as necessary.

18.3.5.2 Process Plant Control Room (WBS 2900)

The process plant control is a modular office sized for 18 employees. This building is attached to the process plant and contains dual operator stations.

18.3.5.3 Assay Laboratory (WBS 4600)

The assay laboratory is a one-story modular building comprised of storage area, office, scale room, AA room, wet lab, and met labs. This building is equipped with fire protection and an alarm system. The laboratory requires bottled nitrogen and hoods with ventilation.

18.3.6 On-site Infrastructure

18.3.6.1 Gate House (WBS 4600)

The gate house is a security trailer office with a lockable gate and communications to the main site.

18.3.6.2 Security/Medical Facilities (WBS 6100)

The security/medical facilities are a modular part of the general office building. The security facilities include rooms for luggage and personnel screening during rotations in and out of site. The medical facilities consist of first aid and emergency response rooms for on-site treatment and headquarters for mine rescue team. These facilities are equipped with fire protection and an alarm system.

18.3.6.3 Main Administration Building (WBS 4600)

The main administration building is a modular, multiple level building comprised of a change/lunch facility, offices, meeting rooms, washrooms, desks, fire protection, and alarm systems. The offices will have space for 41 employees. There will be 20 processing plant offices and 21 general and administrative offices.

18.3.6.4 Accommodation (WBS 5500)

Permanent accommodations on site will be in a camp of 525 individual dormitories. The camp will be a modular building with multiple levels and will include a kitchen and dining area as well as recreation room. There will be a boot and jacket room for personnel entering and leaving the accommodations. The camp will be built to accommodate workers for the construction phase of the project and will be converted thereafter for use by operational workers.

18.3.6.5 Incinerator

The incinerator is a diesel-driven, containerized package with skid-frame. It will have the capacity to burn 1270 kg/day of solids waste during construction and 400 kg/day of solid waste during operation.

18.3.6.6 Water Management (Potable Water)

The water treatment plant will have the capacity to treat 75 L/s of water.

Table 18-1: On-Site Buildings Description

WBS	Building Name	Building Type	L (m)	W (m)	H (m)	Area (m^2)	Volume (m^3)
1200	Truck Shop/Truck Wash	Pre-Engineered	67.5	50	16	3375	54000
1200	Explosive Magazine (Open Pit and UG)	Modular	20	30	4	600	2400
1200	Mine Office (Open Pit and UG)	Modular	33.75	20	4.3	675	2902.5
1200	Mine Dry Facilities	Modular	27	26	-	702	-
4600	Plant Warehouse/Shop	Pre-Engineered	48	20	-	960	-
2900	Process Plant Control Room	Modular	16.2	10	4.3	162	696.6
4600	Assay Laboratory	Modular	20	20	4.3	400	1720
4600	Gate House	Modular	10	5	4.3	50	215
6100	Security/Medical Facilities	Modular	5	3	4.3	15	64.5
4600	Main Administration Building	Modular, Multiple Level	23	15.8	4.3	369	1586.7
5500	Permanent Accommodation	Modular, Multiple Level	-	-	-	-	-

18.3.7 Waste Rock Storage Facility

The project will require a waste rock storage facility to store all non-mineralized material from the pits. This material will be deposited on a waste dump north of the process plant. A slope angle of 32 degrees was applied, which is assumed to consider berms and batter angles. The waste dump was designed to fit 87 Mt of rock. The project will also have a low-grade stockpile used to blend mill feed with high-grade underground material. The locations of the waste rock pile and low-grade stockpile are shown in Figure 18-1.

18.3.8 Tailings Storage Facility (TSF)

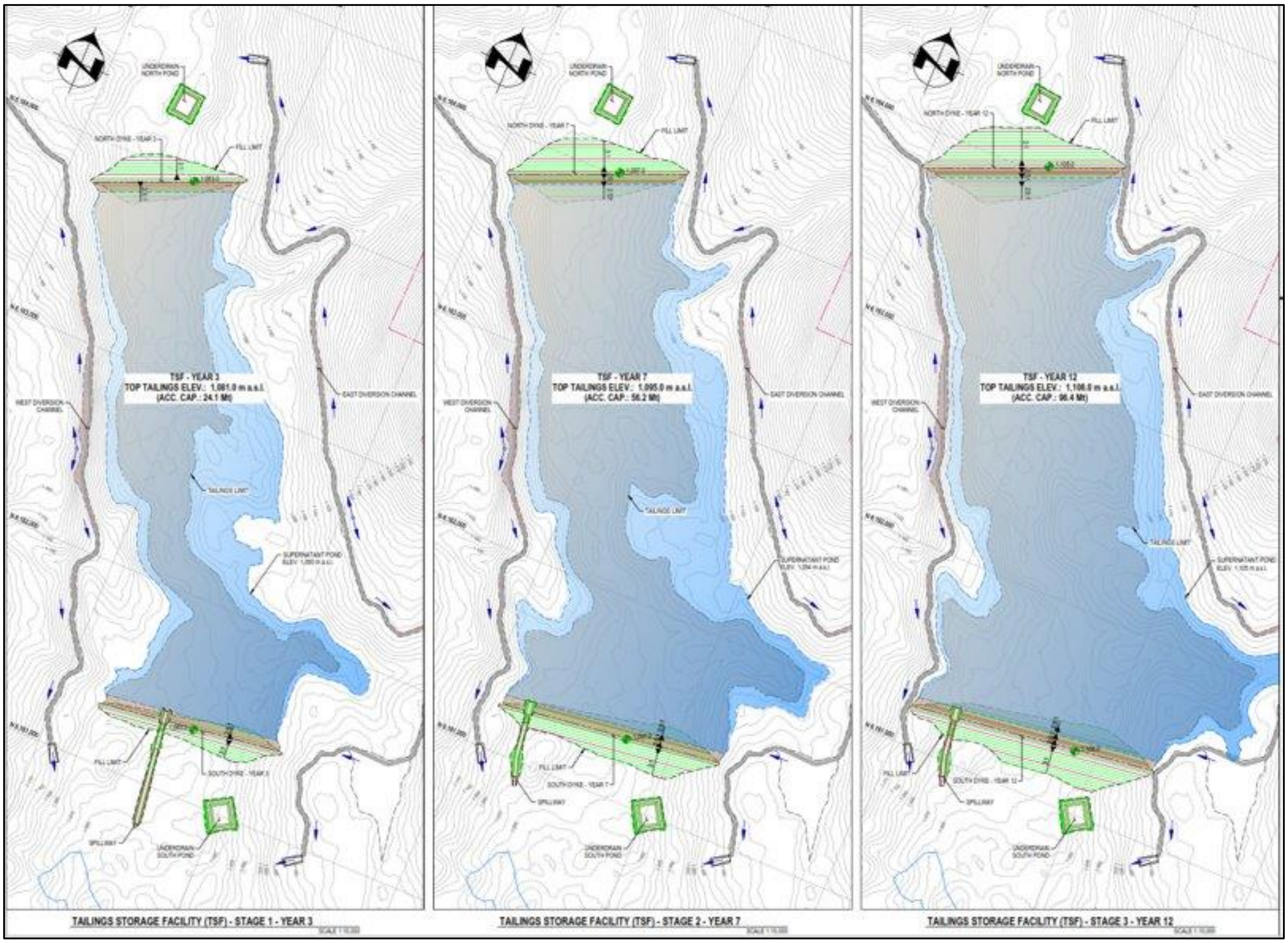
The primary design objectives of the TSF are the secure containment of tailings and the protection of the regional groundwater and surface water during mine operations and after closure. The design of the TSF and accompanying water management facilities has considered the following:

- Staged development of the facility over the LOM.
- Geocomposite liner system (geomembrane liner plus low permeability soil) on the upstream face to limit seepage through the staged embankment.
- Control, collection, and removal of water from the facility for reuse as process water to the maximum practical extent.

The TSF has been designed to accommodate over 96.36 Mt of tailings produced over the life of mine. The proposed TSF will be located in a valley east of the Stardust Deposit and upstream of the process plant site. The site drains both to the northwest and southeast requiring two embankments to contain slurry tailings. Runoff above the facility will diverted around the facility in channels and perimeter access roads, which allows for simple access by the tailings deposition lines and water reclaim system. The TSF will be constructed using a shell of non-acid generating waste rock with an upstream impermeable layer (geomembrane and low permeability soil). The construction of the TSF will utilize downstream construction methodology along with being built in multiple phases to ensure safety and long-term containment of the tailings. The facility is designed in accordance with Canadian Dam Association guidelines (2019) and Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (2016).

The layouts of the three stages of the TSF are shown below in Figure 18-3.

Figure 18-3: Tailings Storage Facility Stage Layout



Source: Ausenco, 2022

18.3.8.1 Topography and Drainage

The proposed TSF site is in a valley located northwest of the processing plant. In general, the site and surrounding area has a mountainous topography with some bedrock exposed or near ground surface in upland and hill areas, and alluvial soils in lowlands and valleys.

At this time, near surface groundwater has not been thoroughly investigated, but the bottom of the valley is wide with a stream running through the facility. Aerial imagery and site visit does not exhibit evidence of significant storm event flows.

18.3.8.2 Facility Design

Two waste rock dams will be impounding the tailings in a shallow valley running north-south. The dams will be developed using concepts that will provide safe and stable dams. The basin will not be lined assuming geological containment can be proven with future geotechnical investigations. The upstream slope of the north and south embankments will be keyed-in to competent bedrock by removing alluvial soils from the valley. The embankment will be lined with 2-mm LLDPE geomembrane liner underlain by a five (5) metre clay liner and a five (5) m filter layer to contain the tailings solids and fluids. Tailings will be transported to the TSF at around 31% solids (by weight) through a slurry pipeline.

The TSF footprint will be cleared and grubbed for foundation preparation and embankments construction. Basin preparation will include removal of overburden material from low points within the topography and placement over any rock outcrops. Overburden materials will be removed beneath the embankment foundations prior to fill placement.

The TSF will be constructed using ROM non-ARD rock generated from open pit mining operations. Rock will be transported by mine haul trucks to a designated staging area east of the main embankment. During construction, rock will be transported by contractor from the staging area to the embankment location(s) and placed as engineered fill in controlled and compacted lifts. The north and south dams of the projected TSF have a crest elevation of 1108 m.a.s.l, a dam height of 68 m and 58 m respectively, crest width of 10 m, downstream slope of 3H:1V and an upstream slope of 2.5H:1V. The embankment construction method will be downstream raise construction.

A drainage layer and a blanket drain will be installed in the upstream slope and underneath the embankment to capture any possible seepage through the dam and maintain a drained downstream zone for stability. Within the foundation blanket drain, a series of 300 and 100 mm PCPE pipes will be installed to collect any seepage and convey it to the seepage collection pond at the toe of the embankment. Seepage water collected in the pond and excess supernatant water collected in the impoundment will be pumped back to the process plant for use in process.

18.3.8.3 Tailings Storage Facility Stability

Sections through the highest portion of both embankments were selected as critical sections for slope stability analysis. Stability was assessed using the limit-equilibrium modelling software SLIDE v6.0, (Rocscience, 2021). Analyses were undertaken for both static and pseudostatic (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. It was determined that the tailings embankment is designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event.

18.3.8.4 TSF Surface Water Management

During operations, permanent storm water diversion channels adjacent to the perimeter facility road will be constructed to convey runoff around the proposed TSF ultimate footprint. Permanent stormwater diversion channels will remain in place during the life of the TSF and into long-term closure. Stormwater diversion channels will be constructed at a minimum 1% grade. Channels are sized to a minimum depth of 50 cm, a minimum width of 2.5 m, and are lined with 30 cm of riprap. Any precipitation that runs off downslope of the diversion channels will report to the impoundment area. Diversion channels will discharge non-contact water into natural drainages and lakes.

18.3.9 Power and Electrical

The HV transmission line from the grid will be connected by distribution line to a 230kV/25kV substation on site. The substation will distribute power to various areas of the project including the process plant, administration building, and the four mining areas. Four distribution lines will be constructed at the project site to provide stepped-down power to the site administration and process facilities, the Kwanika Central pit, the Kwanika Block Cave, the Stardust pit, and the Kwanika South pit. A 25-kV line will be stepped down to 4.16 kV before distribution to the process plant.

18.3.10 Site Water Management

This section discusses site-wide water management, the design of water management structures, hydrology, and water balance. Major drainage paths within the study area were delineated through GIS analysis of LiDAR elevation data with a 5 m contour resolution.

18.3.10.1 Climate and Hydrology

The climate stations close to the project site and with sufficient minimum data history are Smithers A, Quick, Fort St James and Germansen Landing (Figure 18-5). Table 18-2 briefly describes their geographical location relative to the site and their data history period. The stations' climate normal are presented in Table 18-3 to Table 18-4. The IDF curve obtained from Environment Canada is present in Appendix A. Table 18-2 summarizes GERMANSEN LANDING station storm events of the various return.

Table 18-2: Climate stations close to the Kwanika site

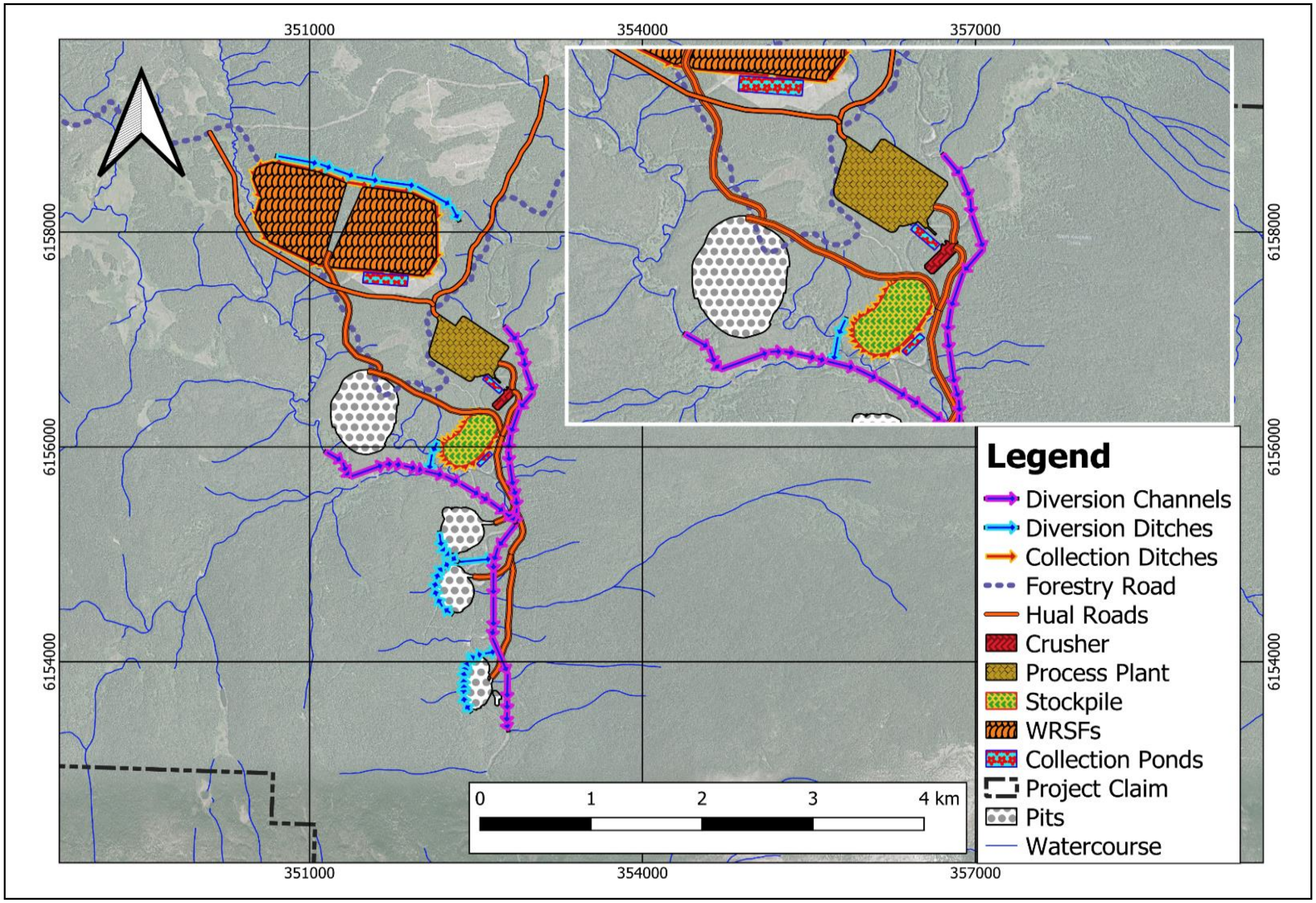
Station Name	Station ID	Distance to site (Km)	Elevation (m)	Lat	Lon	First Year	Last Year
Smithers	1077500	142	521	54°49' N	127°11' W	1971	2002
Quick	1076638	146	533	54°37' N	126°54' W	1963	1994
Fort St James	1092970	139	691	54°28' N	124°17' W	1976	2005
Germansen Landing	1183090	47	766	55°47' N	124°42' W	1964	2005

18.3.10.2 Water Management Structures

This section summarizes a list of proposed water management structures for the Kwanika mine site. The major structures are as follows:

- Diversion Channels and Ditches – diversion channels and ditches are required to divert clean runoff away from the facilities and to minimize the amount of contact runoff to be collected and managed. The design criterion for the diversion ditches was the conveyance of 1:100-year peak flow without overflow.
- Collection Ditches – collection ditches collect contact runoff from the WRSF, Stockpile, Processing Plant, and Crusher Area. The design criterion for collection ditches was the conveyance of 1:100-year peak flow without overflow.
- Collection Ponds – collection ponds were proposed to store contact runoff from the collection ditches. The collection ponds' design criteria were to store 1:100-year 24-hour flood with a minimum freeboard of 0.5 m. The stored contact water should be either treated and released to the environment or reused for process purposes. Figure 18-4 location and arrangement of mine water management facilities.

Figure 18-4: Location of Mine Water Management Facilities



Note: Figure prepared by Ausenco, 2022.

18.3.10.2.1 Conceptual Design and Quantity Estimates

Ditches and ponds were sized using estimated peak flow rates and flood volumes from the Rational method and frequency analysis results. Collection ditches were designed trapezoidal of 2.5:1 (H:V) side slopes with dimensions shown in Table 18-3. Similarly, diversion channels and ditches were designed with a 3:1 and 2.5:1 (H:V) side slope.

Table 18-3: Material Take Off (MTO), Riprap, and Liner Area Estimates for Different Water Management Facilities

Item	Excavation Volume (m^3)	Fill Volume (m^3)	Riprap (m^3)
Diversion channel	339,832	51,200	43,080
Diversion ditch	8,964	0	NA
Collection ditch	10,962	0	NA
Collection pond	104,481	0	NA
Total	**464,239**	**51,200**	**43,080**

18.3.10.3 Site-wide Water Balance

A preliminary site-wide water balance analysis was performed for the Kwanika mine site and is summarized below.

In this analysis, a comparison between water requirements, and available water from the collection system was made to identify the site-wide water balance. This analysis has been made for average, wet and dry climate conditions at the site. The following water components were considered in this calculation:

- Surface runoff from precipitation on WRSF, stockpile, process plant area and pits,
- Evaporation from ponds and pits,
- Process water requirement,
- Tailing Storage Facility reclaim capacity.

As shown in Table 18-4 there is a net annual water deficit of around 232, 202, and 273 m^3/h for average, wet and dry climate scenarios. Any additional water required for the Project will be sourced from Kwanika Creek. It should be noted that groundwater modelling was not conducted at the time of this report, and pit dewatering values are calculated based on precipitation only. Therefore, groundwater input must be added in the next phase of the project. Figure 18-5 shows the flow diagram across the site for average, wet and dry climate scenarios. Note that the existing water in the final product is not shown in this figure.

The water management plan for the project was developed to collect contact runoff/seepage from any facilities and any clean catchment runoff to be diverted away from the facilities.

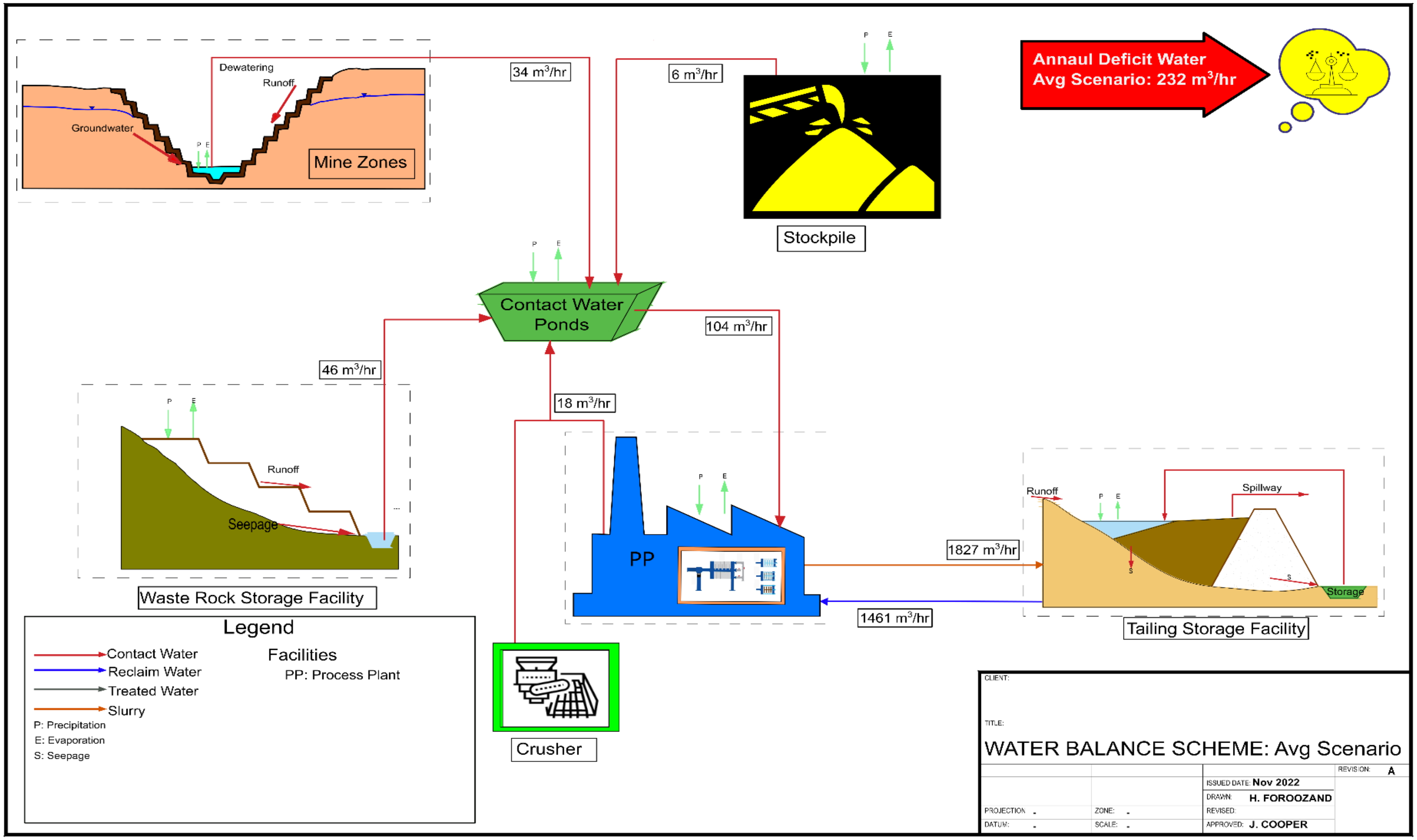
Collection ditches designed to convey contact runoff, were considered to collect, and convey the contact runoff to collection ponds. Only one diversion ditch was designed for the Project. Corresponding excavation volumes were estimated.

Table 18-4: Site-wide Water Balance (m^3/h) – Average Condition

Water Component (m^3/h)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Process Plant water demand	1852	1852	1852	1852	1852	1852	1852	1852	1852	1852	1852	1852	1852
Raw Water Makeup	336	336	336	336	336	336	336	336	336	336	336	336	336
Precipitation Contact Water on Pits													
Precipitation	2	30	33	64	30	45	39	31	36	37	29	33	34
TSF Reclaim Water													
Reclaim Water from Tailings Storage Facility	1461	1461	1461	1461	1461	1461	1461	1461	1461	1461	1461	1461	1461
Contact Water from Net Precipitation and Evaporation													
Process Plant Area	1	14	16	30	18	25	23	18	19	17	14	16	18
Waste Rock Pile	2	37	41	79	47	66	60	48	50	46	35	41	46
Low-Grade Stockpile	0	5	5	10	6	8	8	6	6	6	5	5	6
Pond Direct Precipitation	0	3	3	6	4	5	5	4	4	3	3	3	3
Pond Evaporation	0	0	0	0	7	7	7	6	3	0	0	0	3
Water Deficits/Excess (-/+) in Average Conditions	-331	-248	-237	-147	-239	-194	-210	-234	-224	-227	-251	-238	-232

*Note: The Pit dewatering values are calculated based on precipitation only. Groundwater input must be added in the next phase.

Figure 18-5: Annual Average Water Balance: Average Condition



Note: Figure prepared by Ausenco, 2022.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The concentrate will be trucked from the project site to Mackenzie, where it will be loaded in lots onto bulk material carriers and then transferred onto railcars of the CN Railway to port storage facilities at Vancouver Wharves in North Vancouver. The concentrate will subsequently be transported by sea to clients. Concentrates will be sold into the general market to North American, European, or Asian smelters and refineries.

NorthWest Copper or its consultants have conducted no market study on the sale of copper concentrate. Therefore, the market terms for this study are based on the terms proposed by NorthWest Copper as per their discussion with Ausenco and recently published terms from other similar studies. The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses to support this report.

19.2 Commodity Price Projections

For this technical report, the metal prices presented below in Table 19-1 were used for financial modelling. The metal prices are long-term forecasts over three years provided by an analyst consensus long-term forecast and as agreed by NorthWest Copper.

Table 19-1: Price Projections

Metal	Commodity Unit	Unit Price (US$)
Copper	Pound (lb.)	3.63
Gold	Troy ounce (oz.)	1,650
Silver	Troy ounce (oz.)	21.50

19.3 Contracts

There are currently no sales contracts or refining agreements in place for the project.

The metal payables used in the marketing study are given below in Table 19-2. A summary of the treatment, refining, and transportation costs is provided in Table 19-3 and Table 19-4.

There are no known deleterious elements that could significantly affect a potential future economic extraction. The QP is of the opinion that the information presented here is suitable for use in cashflow analyses to support this assessment.

Table 19-2: Metals Payables

Metal	Unit	Concentrate
Copper	%	96.0%
Less Deductible	%	0
Silver	%	70%
Less Deductible	g/t	0
Gold	%	97.5%
Less Deductible	g/t	0

Table 19-3: Transportation and Treatment Cost

Concept	Value	Unit
Transportation Cost	$ 100.00	per wmt
Treatment Charge	$ 75.00	per dmt

Table 19-4: Refining Charge

Refining Charge	Value	Unit
Copper	$ 0.075	per lb
Gold	$ 5.00	per oz
Silver	$ 0.40	per oz

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This chapter provides an overview of the setting of the Kwanika-Stardust project. It outlines existing biological and physical baseline conditions, proposed new baseline studies to support future permitting applications, existing permits, and future regulatory and permitting requirements including required management plans for water, site environmental monitoring, and waste disposal. In addition, this chapter also discusses socio-economic baseline conditions, the status of community consultation and engagement, and conceptual mine closure and reclamation planning for the project. Recommendations are also provided in the event that the decision is made to progress the project through the prefeasibility study, feasibility study, environmental assessment and permitting phases.

Kwanika-Stardust project involves the development of the Kwanika and the nearby Stardust copper-gold deposits located around 140 km northwest from Fort St. James, and 40 km east of Takla landing British Columbia, Canada. The site is accessible by forest service roads and Tsayta Lake Road and contains several kilometres of excavated trails. The deposits will be developed by surface and underground mining methods. Planned infrastructure includes four open pits (Kwanika Central Pit and the three Kwanika South Pits), as well as two underground operations (Kwanika Central and Stardust), water diversion channels that reroute stream reaches of West Kwanika and Kwanika Creeks, waste rock storage areas, mineralized material storage areas, process plant, roads, powerline and other ancillary supporting infrastructure as discussed and illustrated in Chapter 18. The Kwanika-Stardust project is situated on amongst a group of 61 unpatented mineral claims covering an area of 25,928 ha. The surface rights over the property are owned by the Crown and administered by the Government of B.C., although a detailed review of other potential rights such as placer, timber, water, grazing, trapping, outfitting, etc. has not been undertaken at this time. The property is located within the traditional territory of Takla.

The project site is within a broad valley bordered by mountains of the Omineca Mountain range. Presently, the area is characterized by wilderness, forestry and mineral exploration land use. Aside from the settlements along Takla Lake, the nearest community is Takla Landing, and the majority of roads are used for community access, logging and mining/mineral exploration. The site neighbours Kwanika Creek and West Kwanika Creek, which are tributaries flowing south to the larger Nation River. The proposed project site is contained entirely within the Kwanika Creek subwatershed.

20.2 Environmental and Social Setting

A number of limited field and screening environmental baseline studies and reports were completed in 2018 and 2019 to support a PFS which was underway at the time as well as future potential EA and permitting efforts. The programs involved the collection of baseline data within the proposed project footprint area (as of 2018) and commenced the process of identifying potential environmental constraints and opportunities related to the proposed development of the project, including engineering designs and management plans for the construction, operation, and closure phases of the project. The reports also outlined recommended next steps for the programs. No additional environmental reports were available for the period 2020 to the present.

The 2018-2019 environmental programs covered a range of VECs and involved the following activities:

- Hydrometric and climatic monitoring (Palmer 2019)

- Surface Water quality monitoring (Palmer 2019a)
- Hydrogeological monitoring and testing (Klohn 2019)
- Fish and fish habitat monitoring (Palmer 2019b)
- Soils, Vegetation and wildlife monitoring (Falkirk 2018)
- Socio-economic and cultural baseline studies (Falkirk 2019)

A list of the available reports that were reviewed to support the completion of this section are provided in Section 27 of this report.

In addition to the above studies, a screening level tailings and waste rock facilities alternatives assessment was completed that included environmental criteria as part of the screening methodology and ratings (Klohn 2018). A preliminary geochemistry study was commissioned (Klohn 2019a) that assessed the potential for metal leaching and acid generation from tailings, mineralized material, and waste rock materials.

From a study area perspective, the baseline environmental studies were focused mainly on the areas potentially impacted by the Kwanika deposits and and little information is available for the Stardust deposit area where underground development is proposed. In addition, there have been no baseline studies completed to date on air quality, noise, greenhouse gases and climate change, and groundwater quality.

A summary of the available environmental, social and community studies and factors potentially affecting the project are provided in the following sections.

20.2.1 Hydrology and Climate

The hydrological regime of the project region is snowmelt dominated. The general area is characterized by high flows in the late spring due to snow melt and low flows during the winter months. Flows decrease through the drier summer months, with some rebound in discharges during the autumn months as a result of fall storms and increased precipitation frequency.

The Kwanika-Stardust project area is within the Kwanika Creek subwatershed, which drains 467 km^2. Elevations within the subwatershed range from around 900 to 1990 metres above sea level. Most of the tributaries to Kwanika Creek within the subwatershed originate in the surrounding mountains at higher elevations and increase in stream order as they flow downslope towards Kwanika Creek. The majority of flow in the watershed occurs within the wide valleys between mountaintops, draining into the Nation River at its southernmost point. The proposed mine facilities for the project are along West Kwanika Creek, and upstream of Kwanika Creek.

Two stream gauges were installed in Kwanika Creek upstream and downstream of the proposed project site, one stream gauge was installed on West Kwanika Creek downstream of the project site, and one stream gauge was installed on South Kwanika Creek, a tributary to West Kwanika Creek, upstream of the project site. Stream gauges were installed in May 2018 and visited in July and September for manual monitoring and maintenance. Discharges were measured upwards of 5 m^3/s in May with discharges of less than 1.0 m^3/s measured in September at all stations.

A meteorological station was installed in May 2018 within the project area, and additional sensors were added in July 2018. Data at the climate station were downloaded in September 2018.

There is no record of further hydrological or climate monitoring since 2018.

Long-term monitoring of the project study area will be required if the project advances through FS, EA, and permitting to further characterize the hydrological conditions and develop a water balance model and long-term life of mine water management plan. Section 26.10 provides recommendations for the hydrological studies that will support the advancement of the project through the PFS stage.

20.2.2 Surface Water Quality

During the 2018 field season, five water quality stations were chosen to gain a better understanding of current water quality in West Kwanika Creek, South Kwanika Creek, and Kwanika Creek where potential mine infrastructure was planned. The monitoring array included one upstream reference site and four sites downstream of proposed Kwanika site infrastructure.

Water quality samples were collected monthly at each site between June and September 2018. Water quality results identified four exceedances to water quality criteria (BCWQG) for aluminum, copper and zinc in June, and no exceedances were reported for the July, August and September sampling events at any sites.

Long-term water quality monitoring efforts should focus on areas that will be potentially affected by mine infrastructure based on current infrastructure plans (refer to Chapter 18) and should meet the requirements of an Environmental Assessment application as outlined in Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators, v.2 (ENV, 2016). That guidance document stipulates the requirement for 18 to 24 months of monthly water quality data to support EA applications. Section 26.10 provides recommendations for the surface water quality studies that will support the advancement of the project through the PFS stage.

20.2.3 Hydrogeology

Klohn Crippen Berger completed a hydrogeological investigation in 2018 to preliminarily characterize the site for the purpose of providing estimates of dewatering rates and volumes. A total of eight boreholes were advanced as part of the investigation in the vicinity of Kwanika Central deposit with packer testing completed using a single packer configuration. Six of the boreholes were instrumented with a combination of monitoring wells or vibrating wire piezometres, and a short duration constant head pump test completed on one borehole. Single well in-situ response tests were completed at two well locations.

The results of the investigation provided the basis for the development of a very preliminary hydrogeological conceptual model applicable for the vicinity of the Kwanika Central deposit. The developed model indicated that unconsolidated overburden comprising varying thicknesses of up to 24 m thick overlies a fractured and weathered bedrock zone of up to around 20 m thickness, below which lies relatively sound bedrock that hosts discrete fracture zones. The overburden and shallow weathered/fractured bedrock zone exhibit hydraulic conductivities up to two orders of magnitude greater than the deeper bedrock. However, there are likely permeable fault and fracture zones in the deeper bedrock that have not been characterized or accounted for at this time.

The hydrogeological characterization work completed to date is limited. Most efforts for the collection of hydrogeological testing and monitoring data have been in the vicinity of Kwanika Central deposit; there is little to no data collected for other areas where infrastructure is planned. In addition, groundwater quality was not characterized as part of the 2018 study. Future groundwater monitoring should focus on areas that will be potentially affected by mine infrastructure based on current infrastructure plans (refer to Chapter 18). If the project advances through the FS, EA and permitting, groundwater monitoring and sampling data will need to be adequate to support EA (as outlined in applicable BC guidance document (ENV 2016) for baseline studies) and to support the development of an integrated numerical 3-D groundwater model and a long-term life of mine water management plan. Section 26.10 provides recommendations for hydrogeological studies that will support the advancement of the project through the PFS stage.

20.2.4 Fish and Fish Habitat

Palmer Environmental Consulting Group completed a fish and fish habitat study during the 2018 field season. The objective was to begin the collection of environmental baseline data related to fish community within the proposed project footprint area (as of 2018) to allow the identification of potential environmental constraints and opportunities related to development of the project though the PFS level of design. Specifically, the scope of the fish and fish habitat program involved the identification of species and life history stages of fish in the project area and associated habitat requirements. The main watercourses that may potentially be impacted by the project include the Kwanika and West Kwanika Creeks.

The fish community within streams were sampled at seven locations using a combination of backpack electrofishing and minnow trapping. Six locations along West Kwanika Creek, three sites upstream and three sites downstream of Central Kwanika Pit were sampled and a further sampling location was along South Kwanika Creek. Rainbow trout and Burbot were captured throughout the study area and were healthy according to their relative condition.

Based on a review of available provincial and federal data that spans several decades, other fish species have been observed near the study area including Bull trout and Dolly Varden which are provincially Blue listed, and Yellow listed, respectively (B.C. Conservation Data Centre, 2018). These species were not captured or observed during the 2018 field program.

To establish a better understanding of fish community and habitat baseline conditions within the project site and to support future permitting and approvals, further sampling and assessments are recommended. Section 26.10 provides recommendations for fish and fish habitat studies that will support the advancement of the project through the PFS stage. In the long-term, the baseline program for fish and fish habitat for the project should be designed to meet the requirements which will support the submission of an Environmental Assessment application, (as outlined in ENV 2016). Fish community and fish habitat should include other aquatic resources such as benthic invertebrates and periphyton). ENV requires baseline data collection to occur for an absolute minimum of 12 months (with 24 or more months preferred). Additional data would allow for a better understanding of fish habitat and fish community within the project site to: 1) characterize resources at risk and determine possible impact; 2) predict the significance of impacts and the effectiveness of proposed mitigation; 3) establish thresholds for indicators of ecosystem health; and 4) facilitate the design of monitoring programs. This work will be integral to supporting applications for authorizations under the Fisheries Act, including the requirement for a Fisheries Compensation Plan.

20.2.5 Soils, Vegetation and Wildlife Monitoring

Falkirk Environmental Consultants completed a preliminary field assessment of soils, surficial materials, terrain hazard limitations, vegetation and wildlife during September 2018. A total of 45 plots were established for the collection of soils, vegetation and wildlife data. Plots were completed in the Central Kwanika area and in several of the TSF alternatives areas. The work was summarized in a preliminary draft memo.

Terrain and Soils: No serious limitations related to soil quality and quantity that would limit the proposed development and future reclamation efforts. Soil acidity (pH) was reported as very good in regard to soil nutrient availability during future reclamation efforts.

Vegetation and Ecosystem: This work was mainly focused on TSF options developed by Klohn (2018) which are no longer being considered. The areas surveyed included extensive wetland complexes and older riparian forests with one probable ecological community at risk found in the centre of the wetland complex adjacent to West Kwanika creek. No plant species at risk were found; however, it was noted that it was not the optimum time of year for observation. The potential impacts related to TSF siting in terms of valued components included wetlands, riparian areas, and old growth forests. Impacts to wetlands could trigger the requirement for a Wetland Compensation Plan once the project reaches the EA stage.

Wildlife and Wildlife Habitat: Some project areas overlap with good caribou and moose winter range as well as, spring black and grizzly bear habitat. The wetland complexes are also good habitat for songbirds in the spring and foraging areas for bats in the summer given the extensive wetlands dominated by willows and sedges. The Kwanika Central pit area is very close to the West Kwanika creek and its adjacent riparian forests which may be old growth forest (greater than 145 years old). In that case, some consideration regarding the need for an appropriate buffer to the creek may need to be considered to mitigate impacts to habitat.

Additional surveys will need to be completed related to the areas of terrain/soils, vegetation/ecosystem and wildlife/wildlife habitat for the mine infrastructure presented in Chapter 18. Section 26.10 provides recommendations for soils, vegetation and wildlife studies that will support the advancement of the project through the PFS stage.

20.2.6 Geochemistry

Klohn Krippen Berger Ltd. completed a Waste Rock Characterization study in 2018/2019 for the purposes of understanding the potential for ML/ARD from the project mineralized material and waste rock. These data could be used to define waste rock materials that could potentially be used for construction materials (for TSF) and to allow the development of source terms for a future water quality prediction model. The above work would help to define requirements for studies that would support future environmental assessments and permitting efforts. The Klohn (2019) geochemical report was to be considered a preliminary report of geochemical test results for 103 rock samples collected from the 2006 to 2018 exploration drillholes.

Samples were selected from over 50,000 m of exploration drill core based on providing geological, geochemical and spatial representation within and near the boundaries of the Kwanika Central open pit and underground workings. Static geochemical test work included whole rock analysis, multi-element analysis and ABA methods.

Initial results of the static test work indicated that most of the samples (>50%) represent materials classified as NPAG with 40% of the samples classified preliminarily as PAG or uncertain. A preliminary spatial assessment suggested that discrete zones of PAG and NPAG material are not precisely defined with clear boundaries and classifications are variable across lithologies, depth and mineralized material grades.

The geochemical results do not include all current deposits/pits being considered for development and further work is required for those areas. Additional sample selection and analyses have been recommended in Sections 26.8 and 26.10 to help support and advance the project through the PFS stage.

20.2.7 Socio-Economic, Cultural Baseline Studies and Community Engagement

The project is located within the traditional territory of Takla Nation (Takla). As of 2019, the traditional territory of Takla is in North-Central British Columbia and totalled around 27,250 km^2 including 17 reserves totalling 809 hectares. The closest community to the project is Takla Landing located around 40 km west of the project. Smaller Takla settlements are located at Manson Creek, Bulkley House, Germansen Landing and other areas. Many community members also live outside of the territory. Takla also keeps an administrative office in Prince George. Other First Nations potentially affected by the project include the Nak'azdli Whut'en First Nation and the McLeod Lake Indian Band (MLIB). The MLIB's traditional territories overlap or border those of Takla at or near the project; and include the downstream receiving waters including Tchentlo Lake, Nation River and Williston Lake.

In 2019, Falkirk Resource Consultants Ltd. completed a technical report (Preliminary Socio-Economic, Cultural Baseline Studies and Community Engagement). This report provided a summary of available socio-economic and cultural baseline information for the project and was prepared in part to fulfill preliminary baseline requirements of the environmental, First Nations and socio-economic aspects of the project. Specifically, the report provides preliminary baseline considerations based primarily on desktop-based sources on the following subject areas: population, demographics, education, economy and employment, and infrastructure and civil services.

Community engagement activities of Kwanika Copper Corporation and its consultants were mainly focused on developing environmental monitoring, employment and contracting opportunities in collaboration with Takla, as outlined in the Exploration Agreement, dated September 4th, 2020 between the Kwanika Copper Corporation and Takla.

Traditional Land Use: The known traditional land and resource use within asserted traditional territories include hunting, fishing, plant gathering, habitation, gathering places, sacred sites, trails and travel ways, and trapping. Within the 2018 exploration agreement with Takla. it is understood that Takla's interests regarding traditional land use areas must be addressed through Takla Lands and Resources Department who are responsible for facilitating meetings for impacted Takla Nation families. Several seasonal cabins and campsites are reported to be within the local study area of the project site. Pre-Environmental Assessment (EA) and during the project permitting period additional engagement activities will be required to consult Takla members on the potential impact and cumulative effects of mine development within the Kwanika Creek habitation sites along with any other sites identified. In addition, there are gathering places, sacred sites, and trails and travel ways near or within the project area. The potential interactions of these traditional use sites with the project will need to be addressed during the EA stage of the project. There are also four registered traplines within the project area.

Cultural Heritage: An AOA was conducted by Archer CRM (2008). Results show no known archeological sites are in direct conflict with the proposed development area, at the time. Culturally modified trees (pre-AD 1846) or other protected cultural remains may be present inside proposed development boundaries. A documented aboriginal and probable historic pack train trail (Kwanika Trail) is located in this study area. Overall, archaeological potential is rated moderate to high for the project site and an Archeological Impact Assessment (AIA) will be required at the appropriate time as the project advances into the permitting phase and full extend of the disturbed footprint of the project has been identified.

Exploration Agreements: Takla have been active in seeking employment, business and collaboration opportunities in resource development, so long as development is responsible. The Takla Nation Lands and Stewardship Department is tasked with ensuring resource development, within their traditional territory, is performed in a sustainable, responsible way that benefits Takla's; and follows a stewardship and cumulative effects approach. Hence, the Takla Nation views its relationship with resource developers as a partnership; rather than a proponent/stakeholder relationship.

NorthWest Copper and its predecessor Serengeti have worked closely with the Takla on the Kwanika project. On September 14th, 2020, a new Exploration Agreement was announced between Serengeti (now Northwest Copper) and Takla . The new Exploration Agreement replaced an expired agreement and is valid through to September 14th, 2025. The agreement respects Aboriginal title, rights, and interests, and continues to recognize Takla's stewardship role in environmental and wildlife management and monitoring and traditional land use and knowledge.

On the Stardust project, NorthWest Copper and its predecessor Sun Metals worked closely with the Takla Nation. On August 19, 2020, a new Exploration Agreement was announced between Sun Metals and Takla. The new Exploration

Agreement replaced an expired agreement and was valid through to December 31, 2021. NorthWest Copper and Takla agreed to work using the terms of the previous agreement for the 2022 field season. NorthWest Copper is working with Takla and hopes to have a new exploration agreement in 2023 and the future. The previous agreement respects Aboriginal title, rights, and interests, and continues to recognize Takla's stewardship role in environmental and wildlife monitoring.

Community Engagement: Community engagement activities during 2018 and 2019 included numerous meetings, community updates, employment and training opportunities; and sponsorship of the Takla Nation career fair held in Takla Landing in March 2019. During a meeting in January 2019 KCC met with Takla and outlined their commitment to work with the Takla Nation, including:

- being accountable neighbours of the Takla Nation
- working with Takla Lands and Stewardship Dept. to plan next steps forward
- learning from Takla Nation expertise in regard to navigating the overall process of developing the project
- utilizing Takla Nation expertise to address project related risks and opportunities and to increase the overall value of the project.

During 2020 and 2021, engagement and communication with Takla continued through the implementation of agreements, procurement of services, and employment however engagement in communities was limited due to COVID concerns and restrictions. Following the formation of NorthWest Copper in March 2021, the company continued to engage with Takla through virtual and in person meetings to discuss the Kwanika-Stardust exploration programs, agreement implementation and updating, and contracting and employment opportunities. In 2022, NorthWest Copper also worked collaboratively with Takla to update Wildlife Management and Mitigation Plans (WMMP), conduct an Archaeological Overview Assessment, and refine the Archaeological Chance Find Procedures. The company also worked with Takla to improve its communications and reporting to the Nation by providing monthly activity updates, end of year reporting, agreement implementation financials and uploading information to the Takla Referrals Portal.

Section 26.10 provides recommendations for cultural baseline studies and community engagement efforts that will support the advancement of the project through the PFS stage.

20.3 Permitting

This section summarizes the existing permits in place for the project and the federal and provincial legislation and associated permits, licenses and approvals that will apply or potentially will apply to the construction and operations of the project, as currently proposed.

20.3.1 Existing Permits

20.3.1.1 Kwanika

NorthWest Copper has a Mineral Exploration Permit (No. MX-13-113) issued by the B.C. Ministry of Energy and Mines and Low Carbon Innovation authorizing mineral exploration for the Kwanika project on Claims 501733, 514432, 514433, 502953, 505271, 514455, 546554, and 546553. The permit is valid until August 19th 2027, with the option to renew at the discretion of the B.C. Ministry of Energy and Mines and Low Carbon Innovation. A small reclamation bond has been submitted to the crown to cover the cost of the minor disturbances that are predicted to occur during exploration and for the eventual decommissioning of the existing infrastructure and camp facilities.

20.3.1.2 Stardust

NorthWest Copper has an exploration permit issued by the B.C. Ministry of Energy and Mines and Low Carbon Innovation authorizing mineral exploration for the Stardust project. The permit is good until December 31, 2023, with the option to extend at the discretion of the B.C. Ministry of Energy and Mines and Low Carbon Innovation. The permit authorized work on claims 505166, 514104, 514105, 514106, 504109, 514111, 514114, 514115, 514117, 514119, 514120, 533018, 545320, 545321, 545682, 545684, 505688, 692403, 692424 and 692443. A small reclamation bond has been submitted to the crown to cover the cost of the minor disturbances that are predicted to occur during exploration and for the eventual decommissioning of the existing infrastructure and facilities on site.

20.3.2 Anticipated Permits

A summary of the potential federal and provincial major and minor permits required for the project is provided in Table 20-1.

20.3.2.1 Anticipated Federal Environmental Approvals

The major Federal legislation and associated authorizations related to the anticipated for the project include an Impact Assessment, issued under the Impact Assessment Act (IAA); and a Fisheries Act Authorization, issued under the Fisheries Act.

20.3.2.1.1 Impact Assessment

A new mine project can be designated in accordance with the Physical Activities Regulations, SOR/2019-285 of the IAA. Section 2(18) identifies the thresholds for assessment under the IAA related to the construction, operation, decommissioning, and abandonment of mines and metal mines; if a federal environmental assessment is required for a mining project, the assessment is undertaken by the Impact Assessment Agency of Canada (IAAC). The Physical Activities Regulation lists the mine project designation threshold at:

- a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with a production capacity of 5 000 t/day or more

In consideration that the project will have a production capacity of greater than 5.000 t/day, it will be a designated project in section 2 of the Impact Assessment Act.

Table 20-1: Provincial and Federal environmental approval requirements for the Kwanika-Stardust Project

Legislation	Issuing Agency	Authorization	Purpose
Provincial			
B.C. EAA	B.C. Environmental Assessment Office (B.C. EAO)	B.C. Environmental Assessment Certificate (B.C. EAC)	To assess the potential environmental and socio-economic impacts of the project prior to permitting.
Mines Act (MA)	B.C. Ministry of Energy, Mines and Low Carbon Innovation (EMLI)	Mines Act Permit M-29	Authorization for construction and mining activities.
Environmental Management Act (EMA)	B.C. Ministry of Environment and Climate Change Strategy (ENV)	Environmental Management Act (EMA) PE-00261	Authorization for effluent discharges.
		Environmental Management Act (EMA) PA-105340	Authorization for air discharges.
Drinking Water Protection Act	Interior Health Authority (IHA)	Drinking Water Permits	Drinking water systems.
Forest Act and Forest and Range Practices Act	Ministry of Forests, Lands, and Natural Resources Operations Region (FLNROR)	Occupant License to Cut (OLTC)	Cutting and removal of crown timber from crown land.
Heritage Conservation Act (HCA)	Ministry of Forests (FOR)	Heritage Inspection Permit S 12.2	To assess the archaeological significance of the land and to recover information which otherwise might be lost as a result of site alteration of destruction.
		Site Alteration Permit (SAP) as needed S 12.4	Authorizes the removal of residual archaeological deposits if sites are identified during the inspection and/or investigation.
Land Act	Ministry of Forests (FOR)	Fee Simple Sale ROW update License of Occupation	The Land Act (1996) is the primary article of legislation that is used by the government to convey land to the public for community, industrial, and business use. If a mining project requires the use of Crown land for mining infrastructure in an area not covered under mineral tenure, a Land Act permit will be required.
Mineral Tenure Act	Ministry of EMLI	Mineral Claim Acquisition	Subsurface rights to minerals in a defined unit, up to 10,000 tonnes per year per unit.
	Ministry of EMLI	Mining Lease changes	Conversion of mineral claim to a mining lease is necessary before production can exceed above limits
Water Sustainability Act (WSA)	Ministry of Forest (FOR)	Change Approval S11	Changes in and about a stream are carried out under change approvals and notifications. These approvals authorize work in and around streams.
		New Water License S10	Water licences allow licensees to divert, store, and use specific quantities of water for one or more water use purposes. A water licence may also authorize works related to the diversion and use of the water.
Wildlife Act	Ministry of Forests (FOR)	Authorization Permits for general wildlife (relocation)	If work occurs within identified wildlife areas. A blanket permit applies across the entire project.
Federal			
Impact Assessment Act (IAA)	Impact Assessment Agency of Canada (IAAC)	Impact Assessment Decision	For projects designated as reviewable under the Physical Activities Regulation.
Fisheries Act	Fisheries and Oceans Canada (DFO)	Section 35 Authorization	Where works may cause the harmful alteration, disruption, or destruction of fish habitat.
		Scientific License	License required to harvest fish for experimental, scientific, educational or public display purposes.
	Metal and Diamond Mining Effluent Regulations, SOR/2002-222	Schedule 2 Amendment Authorization to deposit an effluent that contains a deleterious substance	Authorization for a tailings impoundment area in "waters frequented by fish". The definition of tailings in the case of the MDMER involves all mine related waste including mine rock and untreated effluent. Applies to mines that exceed an effluent flow rate of 50 m^3 per day, based on effluent deposited from all the final discharge points of the mine, and that deposit a deleterious substance in any water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water. At this time, it is not known whether a Schedule 2 amendment will be required for the project.
Explosives Act	Ministry of Forests (FOR)	Manufacturing/Storage License	Required for the Manufacturing and Storage of explosives
Migratory Birds Convention Act	Canadian Wildlife Services	Damage or Danger Permit	Required for all activities with the potential to damage or danger migratory birds listed in the Migratory Bird Convention Act, 1994.
Canadian Navigable Waters Act (CNWA)	Transport Canada (TC)	Notice of works to the Minister under CNWA (Major and Minor Works in scheduled and unscheduled waters) Navigation Protection Program	Under CNWA, owners of works propose to construct, place, alter, rebuild, remove or decommission works that are in, on under, through or across any navigable waters may be required to apply to TC, for scheduled waterways, or go through public resolution for unscheduled waters.
Species at Risk Act (SARA)	Environment and Climate Change Canada (ECCC)	Species at Risk Permit	Required if the project crosses critical habitat, as designated under SARA. Requires regulatory consultation with ECCC.

20.3.2.1.2 Substitution and Coordination

When a project falls under both provincial and federal environmental assessment responsibility, there is an agreement in place between B.C. and Canada which enables the two governments to carry out a single, cooperative environmental assessment while retaining their respective decision-making powers. Provincial and federal ministers make independent decisions on whether to issue an Environmental Assessment Certificate (EAC) from a single report.

20.3.2.1.3 Fisheries Act

In effect since June 2019, Canada's modernized Fisheries Act, RSC 1985, c. F-14 provides protection for all fish and fish habitats. Where works may cause the harmful alteration, disruption, or destruction of fish habitat, authorization from Fisheries and Oceans Canada (DFO) under Section 35 of the Fisheries Act may be required. If the proposed mine infrastructure associated with a proposed project, will impact fish-bearing water, then a Fisheries Authorization and Fish Habitat Compensation Plan may be required.

Mining projects in B.C. may also require authorizations from ECCC (Government of Canada 2022) under Section 36 of the Fisheries Act, which prohibits the deposition of deleterious substances into water frequented by fish. If the deposition of a deleterious substance into waters frequented by fish is due to the proposed establishment of a tailings impoundment area or other mine waste storage facility, an authorization from ECCC, in the form of a Schedule 2 (SOR/2002-222) amendment to the MDMER under Section 36 of the Fisheries Act, may be required. At this time it is likely/unlikely that a Schedule 2 amendment will be required by the project.

The project as envisioned in this report will require a Fisheries Authorization and Fish Habitat Compensation Plan. A Schedule 2 amendment to the MDMER may also be required subject to further fish and fish habitat surveys required for areas where mine waste will be stored (waste rock, tailings, mineralized material, and untreated contact water / mine effluent).

20.3.2.2 Anticipated Provincial Environmental Approvals

The major provincial legislation and associated authorizations anticipated for the project include the following: a B.C. MA Permit issued under the MA; Effluent and Air Emissions Permit issued under the B.C. EMA; and B.C. EAC, issued under the EA.

20.3.2.2.1 Mines Act (MA)

All mines in B.C. must hold a permit issued by the Chief Permitting Officer (CPO), under the MA and in accordance with Part 10 of the Health, Safety and Reclamation Code for Mines in British Columbia (Government of B.C. 2017). Permitting under the MA is applicable for all on-site mining activities and considers detailed designs for all project components and phases of mine life, including construction, operation, reclamation, and closure. An amendment application must be made if a proponent wishes to change any component of their mines act permit unless the changes meet the criteria of a deemed authorization or a departure from approval.

20.3.2.2.2 B.C. Environmental Management Act (EMA)

The EMA is the primary piece of legislation that enables the Province of B.C. to regulate the introduction of waste to the receiving environment. The Waste Discharge Regulation of EMA prescribes those discharges of wastes from mining activities to the environment require authorization under the EMA. The EMA is under the mandate of the B.C. ENV.

According to the Ministry of EMLI, an EMA permit to authorize the ongoing discharge of waste for a mining project is required for:

- Effluent discharges (e.g., tailings pond supernatant, mine-influenced runoff and sewage)
- Air emissions (e.g., refuse incinerator emissions, emissions from large power-generating plants, emissions from milling processes, etc.)
- Solid wastes (e.g., mill tailings, water treatment plant sludge, municipal and industrial refuse, etc.) (Government of B.C. 2020)

An EMA permit will set the terms and conditions for the waste discharge, with the goal of ensuring the protection of human health and the environment. The terms and conditions of an EMA permit may include limiting the quantity and quality of waste contaminants and monitoring the discharge and the receiving environment.

20.3.2.2.3 Joint Mines Act/Environmental Management Act Application

MA and EMA permits are required for all new mines in B.C. to allow for mining activities and associated discharges to the environment. Although these permits fall under the jurisdiction of the EMLI and ENV, complex applications allow for an integrated permitting process that will enable proponents to apply for a coordinated authorization review process. Complex projects are referred to the coordinated authorization process, including a mining project that is an extension, expansion or re-start requiring multiple authorizations.

There are no statutory timelines for the coordinated authorization process based on varying project conditions. However, typical service timelines for major processes range from six to twelve months from application submission to decision (this does not include pre-application) (Ministry of Energy, Mines and Low Carbon Innovation 2022).

20.3.2.2.4 B.C. EAA

Large projects have the potential for significant environmental, economic, social, cultural and health effects. In B.C., when a proposed project meets or exceeds criteria thresholds in the Reviewable Project Regulations, it must undergo an EA. EAs are managed by the EAO, which administers the process and authorizations legislated under the BCEAA.

For mining projects, the thresholds in the Reviewable Project Regulations (BCR 243/2019) are based on the volume of production and area of new land disturbance as follows:

- A new mine facility that, during operations, will have a production capacity of ≥ 75 000 tonnes/year of mineralized material.

In consideration that the project will have a production capacity of greater than ≥ 75 000 tonnes/year, it will be a designated as a reviewable project under the Reviewable Project Regulation and subject to EA.

20.4 Environmental Management and Monitoring Plans

As the project progresses through the PFS and EA/permitting stage a number of environmental management and monitoring plans will be required for the purpose of guiding the development and operation of the project and mitigating and limiting environmental impacts. These plans will be complementary to the engineered designs that will be required for the storage of tailings, waste rock, mineralized material, and conveyance/storage (refer to Chapter 18 of this report). series of mine and environmental management and monitoring. For the purposes of the Application for an EAC (Application), management plans are described at a conceptual level for aspects of the project where potential effects to valued components (VCs) were identified during the EA. A preliminary list of the plans that will need to be developed are provided in Table 20.2.

Table 20-2: List of Anticipated Environmental Management and Monitoring Plans for EA and Permitting

• Sustainability Management System Access Management Plan	• Access Management Plan
• Groundwater Contingency and Monitoring Plan	• Fish and Aquatic Effects Monitoring Plan
• Air Quality Management and Monitoring Plan	• Surface Erosion and Sediment Control Plan
• Emergency Response Plan	• Environmental Emergency, Spill, and Hazardous Materials Plan
• Hazardous Materials Management Plan	• Mine Waste, Tailings, and ML/ARD Management Plan
• Heritage Management Plan	• Reclamation and Closure Plan
• Occupational Health and Safety Plan	• Soil Handling Management Plan
• Surface Water Management and Monitoring Plan	• Ecosystems Management Plan
• Vegetation Management and Monitoring Plan	• Wetland Monitoring Plan
• Water Treatment Plan	• Wildlife Management and Monitoring Plan
• Invasive Plant Management and Monitoring Plan	• Waste Management Plan
• Stakeholder and Indigenous Nations Communication Plan	

20.5 Other Potential Environmental Concerns

The historic Bralorne Takla Mercury Mine is located within the property boundaries. This historic mine site is under the jurisdiction of the CCSP. A full remediation and cleanup program was completed on this site through CCSP in 2018. At this point, only ongoing monitoring through CCSP and their contractors is required. NorthWest Copper is not involved with or responsible for any of the ongoing monitoring programs.

There are three provincial parks within 30 km of the project including:

- Nation Lakes Provincial Park located around 10 km south and downstream of the project and consists of a land area of 19,398 hectares
- The Omineca Provincial Park and Protected Area is located around 20 km north and 25 km northeast of the project. The protected area includes 3,138 hectares protecting spring calving areas for the blue listed Northern Woodland Caribou in the Wolverine Range; important moose winter range and waterfowl habitat along the Omineca Rivers; goat habitat in the South Omineca and Germansen Lake areas; as well as important wolverine habitat

- Mt. Blanchet Provincial Park is around 30 km southeast of the project and comprises 24,774 ha on Takla Lake. Mount Blanchet Park protects valuable wildlife habitat including spring range for ungulates on the south-facing slopes along the northwest arm of Takla Lake. The park includes significant caribou over-wintering and calving areas, and alpine habitat suitable to mountain goat and grizzly bear. Sockeye salmon spawn in several of the creeks.

20.6 Conceptual Mine Closure and Reclamation Plan

Under the BC Mines Act, anyone who engages in mining exploration work or mining operations determined by regulation must submit a reclamation plan. A conceptual reclamation and closure plan and a closure security estimate will need to be developed to support the submission of an EA report to the province or to the federal agency. The reclamation security will need to be posted to BC government prior to the commencement of construction the construction phase.

The current Conceptual Closure and Reclamation Plan for the project includes the following measures:

- Partial backfilling of open pits with waste rock, and flooding of the remaining open pit, and in the case of the Kwanika Central Pit, the underlying block cave mine, likely achieved by breaching the diversion dam and channel
- The mineralized material stockpile will be reclaimed, once depleted
- The mine portals will be decommissioned, plugged and backfilled
- The plant and infrastructure pad will be dismantled, removed, and re-contoured and revegetated
- The tailings dam will be vegetated to establish an erosion resistant surface
- The tailings beach will be capped with soil and vegetated
- Water treatment will be continued until the TSF water quality meets discharge criteria
- Once TSF water quality meets discharge criteria, water treatment will be stopped, diversions will be decommissioned, and the TFF will be allowed to discharge naturally via a closure spillway
- At closure, PAG rock will be managed by: rehandling into the pit to keep it permanently submerged in the pit lake or capping it with low permeability glacial till to reduce seepage and oxygen infiltration. NPAG waste rock stored on the surface will be capped with soil and revegetated.

For recommendations section move to recommendations chapter. The closure and reclamation costs are discussed in section 21.2.12.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated cists that can be used to assess the preliminary economics of the Kwanika-Stardust project. The estimates are based on an open pit and underground mining operation, as well as the construction of a process plant, tailings storage facility, and infrastructure, as well as Owner's costs and provisions. The processing plant nameplate capacity is 22,000 t/d (8.03 Mt/a), with a life of mine of 11.8 years.

All capital and operational cost estimates are presented in Canadian dollars (C$), with no escalation or exchange rate variations factored in. An exchange rate of 0.77 (CAD/USD) has been applied as necessary.

The capital cost estimate conforms to Class 5 guidelines for a preliminary economic assessment level estimate with a +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q4 2022 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

21.2 Capital Costs

21.2.1 Capital Cost Estimate Summary

The total initial capital cost for the Kwanika-Stardust project is C$567.9 million, the LOM sustaining cost including financing is C$282.5 million, and the LOM growth capital cost is C$493.3 million. The capital cost summary is presented below in Table 21-1.

Table 21-1: Summary of Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Mining	1000	65.8	151.4	393.3	610.5
Process Plant	2000	198.0	0.0	0.0	198.0
Additional Process Facilities	3000	6.4	5.6	0.0	12.0
On-site Infrastructure	4000	21.6	4.9	0.0	26.5
Off-site Infrastructure	5000	82.5	78.5	0.0	161.0
Total Directs		**374.3**	**240.4**	**393.3**	**1008.0**
Project Preliminaries	6000	28.4	2.1	0.0	30.5
Project Delivery	7000	50.4	2.1	0.0	52.5
Owner's Costs	8000	33.7	27.3	100.0	161.0
Provisions	9000	81.1	10.5	0.0	91.6
Total Indirect		**193.6**	**42.0**	**100.0**	**335.6**
Project Total		**567.9**	**282.4**	**493.3**	**1343.6**

*Numbers may not add up to rounding

The capital cost for the Project is split into initial capital, sustaining capital, and growth capital costs. The initial capital is any project development cost incurred during the pre-production years. Sustaining capital is the capital incurred to support production from the project. Growth capital is the capital incurred for the development of Kwanika block cave mine and Stardust Mine during the production period.

21.2.2 Capital Cost Estimate Responsibilities

The capital cost estimate was developed in accordance with the responsibility breakdown presented in Table 21-2.

Table 21-2: Capital Cost Responsibility by WBS

WBS	Description	Ausenco	Mining Plus
1000	Mining		
1100	Geology and Mine Design		Y
1200	Mine Infrastructure and Services	Y	Y
1300	Surface Mining		Y
1400	Underground Mining		Y
2000	Process Plant		
2100	Crushing	Y	
2200	Stockpile and Reclaim	Y	
2300	Grinding	Y	
2400	Flotation & Regrind	Y	
2500	Concentrate Handling	Y	
2700	Tailings Thickening and Pumping	Y	
2800	Reagents	Y	
2900	Process Plant Services and Common	Y	
3000	Additional Process Facilities		
3100	Mobile Equipment	Y	
3200	Water Treatment Plant	Y	
4000	On-Site Infrastructure		
4100	Bulk Earthworks	Y	
4200	Power Station	Y	
4300	MV Power Switchyard and Power Distribution	Y	
4400	Fuel Storage	Y	
4500	Sewage	Y	
4600	Infrastructure Buildings	Y	
5000	Off-Site Infrastructure		
5100	Main Access Road	Y	
5200	Water Supply	Y	
5300	Power Supply HV	Y	
5400	Tailings Storage Facility	Y	
5500	Permanent Camp	Y	
5600	Pipeline	Y	
6000	Project Preliminaries	Y	
7000	Project Delivery	Y	
8000	Owner's Costs	Y	
9000	Provisions	Y	Y

21.2.3 Basis of Estimate

The capital cost estimate was developed in Canadian dollars in Q4 2022 and was based on data from projects and research in Ausenco's internal database and knowledge gained from similar operations. The capital cost estimate conforms to Association for the Advancement of Cost Engineering International (AACE International) requirements for a PEA-level estimate with +50%/-30% accuracy. An exchange rate of 0.77 is applied for CAD:USD conversion.

The data for the estimates has been derived from a variety of sources, including the following:

- mining schedule
- conceptual engineering design by Ausenco and Mining Plus
- major mechanical equipment costs are based on vendor quotations, first principles, and Ausenco's database of historical projects
- material take-offs (MTOs) for concrete, steel, electrical, instrumentation, in-plant piping, and platework were factored by benchmarking against similar projects with equivalent technologies and unit operations
- topographical information considered
- engineering design at the level of a preliminary economic assessment
- cost escalation to 2022 when historical pricing is considered.

21.2.3.1 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- land acquisitions
- sales taxes

21.2.4 Direct Costs – Mining (WBS 1000)

21.2.4.1 Kwanika Central Block Cave Capital Costs

Capital costs for the Kwanika Central Block Cave mine were estimated in the 2019 Internal Report. The estimate was based on first principles of materials and quantities and includes an 18% contractor's overhead and profit markup. Where designs were incomplete, quantities were factored or assumed based on similar installations. Quotes were obtained in 2019 for owner's production mobile equipment, ground support bolts, ventilation ducting, utilities piping, and explosives. Capital costs have been factored for this study to account for inflation; and where applicable, adjustments to throughput and footprint size (Table 21-11).

21.2.4.2 Stardust Underground Capital Costs

The Stardust underground capital cost encompasses box cut and pads for portal and ventilation shaft, haul road, dewatering, ventilation, electrical, and compressed air. Underground Equipment has not been included in initial capital because it will be contractor based. Equipment to be purchased by the owner are support equipment for the owner, surface haul trucks, and grader.

21.2.4.3 Underground Mines Growth Costs

Separate from the mining initial and sustaining capital costs are the mining growth costs. These growth costs represent the expansion capital required to develop the underground mines before production.

21.2.4.4 Open Pit Mines Capital Costs

The open pit capital cost encompasses 10 Mt pre-striping mining cost from Central Pit in year -1. Since the open pits are planned to be contractor based, no major equipment purchasing is included in the initial capital cost (Table 21-12). Also, no sustaining capital cost is included in the open pit mining cost estimation. The only equipment purchase cost for open pit mining will be ancillary equipment for the owner. Total mining capital cost including initial capital, sustaining capital, financing costs, and growth capital are presented in Table 21-3.

Table 21-3: Total Mining Initial, Sustaining, and Growth Capital Costs – Open Pit and Underground Mines

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Geology and Mine Design	1100	-	-	-	-
Mine Infrastructure and Services	1200	7.3	4.9	17.3	29.5
Surface Mining	1300	30.2	-	-	30.2
Underground Mining	1400	28.3	146.5	376.0	550.8
Total Direct		**65.8**	**151.4**	**393.3**	**610.5**

21.2.5 Direct Costs – Process Plant (WBS 2000)

The definition of process equipment requirements was based on process flowsheets and process design criteria, as defined in Section 17. All major equipment was sized based on the process design criteria to derive a mechanical equipment list. Mechanical scopes of work were developed, and major equipment were sent for budgetary pricing to equipment suppliers. For mechanical equipment costs, 40% of the value was sourced from budgetary quotes; the remainder was sourced by benchmarking against other recent North American flotation concentrator mining projects and studies. Similarly, the major electrical equipment was sized based on the project's equipment list and the equipment were sourced by benchmarking against other recent North American flotation concentrator mining projects and studies.

In support of the major installation construction contracts, engineering for the process plant was completed to a PEA-level of definition. Bulk material quantities were derived for earthworks and priced from other benchmark projects. All other quantities for concrete, steel, piping, cable etc. were factored and priced.

Process plant costs are summarized in Table 21-4 and described in the following sections. Direct costs include all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

Table 21-4: Process Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Crushing	2100	12.0	-	-	12.0
Stockpile and Reclaim	2200	10.0	-	-	10.0
Grinding	2300	116.5	-	-	116.5
Flotation & Regrind	2400	34.2	-	-	34.2
Concentrate Handling	2500	6.3	-	-	6.3
Tailings Thickening and Pumping	2700	4.9	-	-	4.9
Reagents	2800	8.7	-	-	8.7
Process Plant Services	2900	5.3	-	-	5.3
Total Direct		**198.0**	**-**	**-**	**198.0**

21.2.6 Direct Costs - Additional Process Facilities (WBS 3000)

The breakdown of the additional process facilities capital costs is presented below in Table 21-5. These process facilities include a water treatment plant and the sustaining capital costs for financing processing plant mobile equipment.

Table 21-5: Additional Process Facilities Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Mobile Equipment	3100	1.1	5.6	-	6.7
Water Treatment Plant	3200	5.2	-	-	5.2
Total Direct		**6.4**	**5.6**	-	**12.0**

21.2.7 Direct Costs - On-Site Infrastructure (WBS 4000)

In support of the major installation construction contracts, engineering for the project was completed to a PEA-level of definition. Bulk material quantities were derived for earthworks including the entire project site, process plant, TSF and water management structures, and priced from other benchmark projects. The power requirements for the project were estimated based on the electrical equipment list developed for the process plant, power demand for the mining operations and allowances for other site requirements such as pit dewatering, lighting etc. The cost of the substation and cable routing was estimated based on benchmarked North American projects.

The buildings required for the operation were sized and costed based on benchmark projects with similar weather and snow conditions located in Canada.

The breakdown of the on-site infrastructure capital costs is shown in Table 21-6.

The on-site infrastructure covers the cost of the site earthworks, site-wide electrical distribution, fuel storage, sewers, and various infrastructure buildings.

Table 21-6: On-Site Infrastructure Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Bulk Earthworks	4100	10.5	3.5	-	14.0
Power Station	4200	4.4	-	-	4.4
MV Power Switchyard and Power Distribution	4300	1.5	1.5	-	3.0
Fuel Storage	4400	0.8	-	-	0.8
Sewage	4500	0.6	-	-	0.6
Infrastructure Buildings	4600	3.8	-	-	3.8
Total Direct		**21.6**	**4.9**	-	**26.6**

21.2.8 Direct Costs - Off-Site Infrastructure (WBS 5000)

The breakdown of the costs for the off-site infrastructure planned for the project is shown in Table 21-7.

The off-site infrastructure costs include building the main access road, water supply, high voltage power supply, tailings storage facility, permanent camp, and pipeline.

The permanent camp will be financed over 7 years which includes 2 years of pre-production and 5 years of postproduction period. A down payment of 20% of the total cost will be made during the pre-production period and the balance of payments including interest payments will be made from Y1 to Y5. The camp will be financed at an annual interest rate of 7%.

Table 21-7: Off-Site Infrastructure Capital Costs

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Main Access Road	5100	5.1	-	-	5.1
Water Supply	5200	0.5	-	-	0.5
Power Supply HV	5300	23.6	-	-	23.6
Tailings Storage Facility	5400	31.7	37.5	-	69.2
Permanent Camp	5500	8.4	41.0	-	49.4
Pipeline	5600	13.2	-	-	13.2
Total Direct		**82.5**	**78.5**	**-**	**161.0**

21.2.9 Indirect Capital Costs

Indirect capital costs are calculated as a percentage of the direct costs. The indirect capital costs are summarized in Table 21-8 and described below.

Table 21-8: Indirect Capital Costs Summary

WBS Description	WBS	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Growth Capital Cost (C$M)	Total Cost (C$M)
		LOM	LOM	LOM	
Project Preliminaries	6000	28.4	2.1	-	30.6
Project Delivery	7000	50.4	2.1	-	52.5
Owner's Costs	8000	33.7	127.3	-	160.9
Provisions	9000	81.1	10.5	-	91.5
Total Indirects		**193.6**	**142.0**	**-**	**335.6**

21.2.9.1 Project Preliminaries

Indirect costs under this category are required during project delivery to enable and support construction activities. These costs have been estimated based on Ausenco's historical project costs for projects of a similar nature. The initial capital cost of these project preliminaries is 10% of the total direct costs. The sustaining capital costs associated with the project preliminaries is 5% of the total direct capital costs for the MV power switchyard and distribution, bulk earthworks, tailings storage facility, and the haul roads to the Kwanika South Pit.

21.2.9.2 Project Delivery

The project delivery costs, which are estimated at C$50.4 million, have been based on Ausenco's historical project costs of a similar nature and are calculated to be 18% of total direct costs. The sustain capital is 5% of total directs costs for the MV power switchyard and distribution, bulk earthworks, tailings storage facility, and the haul roads to the Kwanika South Pit.

21.2.9.3 Owner's Costs

The Owner's costs are estimated as 5% of total direct costs and are calculated to be C$33.7 million Owner's costs include such things as project staffing and miscellaneous expenses, pre-production labour, home office project management, home office finance, legal costs, and insurance. The sustaining capital costs are estimated from the mining sustaining capital costs. These costs are calculated to be C$27.3 million. The growth capital component of the Owner's costs total C$100.0 million.

21.2.9.4 Provisions (Contingency)

The provisions represent the contingency costs, or variances between the estimated and the actual costs for materials and equipment. The amount of contingency varies according to the terms of the contract and the demands of the client. The estimate for capital costs must have a provision to offset the risk from uncertainty because there were uncertainties when the estimated was created.

The contingency estimate will not allow for the following:

- abnormal weather conditions
- changes to market conditions affecting the cost of labour or materials

- changes of scope within the general production and operating parameters
- effects of industrial disputations
- financial modelling
- technical engineering refinement
- estimate inaccuracy.

The initial capital cost of the provisions is estimated at C$81.1 million. The contingency is calculated as 21% of direct capital costs and 10% of indirect costs plus mining contingency which is estimated at $3.9M, 4.9% of the initial mining capital. The sustaining capital cost of the contingency, which is estimated at C$10.5 is 10% of the direct capital costs of mine infrastructure and services, the permanent camp, process mobile equipment, MV power switchyard and distribution, bulk earthworks, tailings storage facility, and the haul roads to the Kwanika South Pit.

21.2.10 Salvage Costs

The salvage cost for the project is estimated at $2.5 million at the end of mine life which is the cost recovered by selling the process plant equipment.

21.2.11 Sustaining Capital and Growth Capital

21.2.11.1 Overview

The life of mine sustaining cost for the project is estimated at $282.4 million which includes $240.4 million in direct costs and $42 million in indirect costs.

The life of mine growth capital for the project is estimated at $493.3 million which includes $393.3 million in direct costs and $100 million in indirect costs (Owner's costs).

21.2.11.2 Mining

The costs of ongoing mine operation such as pit dewatering throughout the LOM, mobile equipment loan payments and pit expansion costs, and underground development costs are included in the sustaining costs of mining. The LOM mining sustaining capital is estimated at $151.4 million.

The growth capital for the project includes the cost of developing Kwanika underground mine and Stardust mine. The total growth capital is estimated at $493.3 million.

21.2.11.3 Additional Process Facilities and On-site Infrastructure

The sustaining cost under additional process facilities includes the financing cost of process mobile equipment which is estimated at $5.6 million. The 20% down payment for the mobile equipment will be made during the construction period and the balance of payments will be made over 5 years at an annual interest rate of 7% p.a.

The sustaining cost of the on-site infrastructure includes the cost of roads built for Stardust mines and the medium voltage electrical costs associated with Stardust mine. The total estimated LOM sustain cost of on-site infrastructure is $4.9 million.

21.2.11.4 Off-site Infrastructure

The off-site sustaining costs include the cost of TSF expansion and the financing of permanent camp. The 20% of the down payment for the camp will incur during the construction period and the remaining 80% of the cost including interest payments at 7% p.a. will be made over 5 years. The estimated LOM off-site infrastructure sustaining costs is $78.5 million.

21.2.12 Closure and Reclamation Planning

The closure cost for the project is estimated at C$42 million.

21.3 Operating Costs

21.3.1 Operating Cost Estimate Summary

The costs considered operating costs are those related to mining, processing, tailings handling, maintenance, power, and general and administrative activities.

According to the Association for the Advancement of Cost Engineering International (AACE International) requirements for a PEA study, the estimate has an accuracy of +50%/-30% due to the approach employed to create the capital estimate and the conceptual level of engineering definition. A summary of the operating costs is presented below in Table 21-9.

Table 21-9: Operating Cost Summary

Cost Area	Total ($M)	C$/t Milled	% of Total
Mining	1,207.8	12.63	54.8
Processing	776.9	8.13	35.3
G&A	218.1	2.28	9.9
Total	**2,202.9**	**23.04**	**100**

21.3.2 Basis of Estimate

A key assumptions were made to estimate the operating costs for the project:

- Cost estimates are based in Q4 2022.
- Costs are expressed in Canadian Dollars (C$).
- Where applicable, an exchange rate of US$ 0.77 per C$1.00 was used.
- Power cost of C$0.0674 per kilowatt-hour (kWh) was assumed.
- A diesel cost of C$1.23 per litre was assumed based on long-term consensus price.
- A throughput of 22,000 t/d or 8.030 Mt/a was used for the processing plant.
- Processing plant availabilities and operating costs were as per the design criteria shown in Section 17.
- Plant crusher availability is assumed to be 75%, while the availability for the rest of the processing plant is assumed to be 92%.

- ROM and concentrate grades, and recoveries are based on metallurgical testwork results described in Section 13.
- Material and equipment are purchased as new.
- Reagent consumption rates are based on metallurgical testwork results and in-house benchmarks.
- Grinding media consumption rates are based on mineral material characteristics as described in Section 13.

21.3.3 Mine Operating Costs

21.3.3.1 Kwanika Central Block Cave Operating Costs

Kwanika Central Block Cave operating costs were estimated from first principles for the 2019 Internal Report 15,000 t/d throughput option at 7.01 $/t mining cost and 2.30 $/t G&A. Operating costs include direct labour and equipment, materials handling, water, ventilation, power, maintenance, and indirect administrative costs. Operating costs have been factored for this study to account for inflation and throughput to 20,000 t/d. Furthermore, a deduction of 3.15 M$ per annum was applied to the G&A cost to account for the shared overhead burden as the block cave is operated concurrently with the other mines. This study is based on operating unit costs of 8.32 $/t mining and 2.30 $/t G&A.

21.3.3.2 Stardust Underground Operating Costs

Stardust Operating cost was estimated on benchmark-based figures within North American mines. Due to the nature of Stardust being a base and precious metal mine benchmarks were established for both groups of mines. 83.00 $/t mined was established based on the average mining cost between base and precious metal mines. Crush and Haul was based on benchmark data and a calculation of Stardust's haulage needs to the Kwanika Mill. The cost is based upon rehandling material from the Stardust mineralized material stockpile and hauling to the Kwanika Mill for processing. A separate set of equipment will be required to maintain the haul road and reduce cycle time for underground materials handling. Table 21-10 shows the assumptions and benchmark averages used for the Stardust operating cost.

Table 21-10: Stardust Operating Cost Benchmark

Benchmark Mine Costs	Overall Cost	Mine	Crush and Haul
Overall Average	$92.84	$83.11	$9.73
Precious Metal Average	$118.99	$113.41	$5.58
Base Metal Average	$68.25	$57.14	$11.11
Stardust Underground	$90.31	$83.00	$7.31

21.3.3.3 Open Pit Mines Operating Costs

The open pit cost is estimated based on the unit mining costs presented in Table 21-12. The reference contractor mining cost from the 2017 PEA includes the responsibility for all mining areas including direct mining, mine maintenance, and dewatering activities. The costs from the 2017 PEA are escalated to the year 2022; see Table 21-11 for cost escalation rates. This cost is assumed as hard rock mining cost. For overburden mining cost a factor of 0.75 is applied to reference mining cost to deduct the drilling, loading and blasting costs.

Mine technical services, such as geology, engineering, and management will be the owner's responsibility and are captured in the general mine expense area. In this regard, an extra G&A cost is added to the escalated contractor mining cost. Also, the reclaim cost from the stockpile is considered as $0.75/t. Total operating mining cost is presented in Table 21-13.

Table 21-11: Cost Escalation Table

Year	(Chem Eng Plant Cost Index)	from 2017	from 2018	from 2019	from 2020	from 2021	from 2022
2017	568	100%					
2018	603	106%	100%				
2019	619	109%	103%	100%			
2020	638	112%	106%	103%	100%		
2021	702	124%	116%	113%	110%	100%	
2022	772	136%	128%	125%	121%	110%	100%

Table 21-12: The open pit mining cost assumptions

Open pit Mining Costs	Unit	2017 PEA	Escalated cost in 2022***	G&A	Open pit Mining Cost
Reference unit cost for hard rock	CAD/t	2	2.72	0.37*	3.09
Additional cost per bench	CAD/t/bench	0.05	0.07	0.01	0.08
Overburden mining cost	CAD/t				2.32**
Support equipment	CAD	2,500,000	3,400,000		3,400,000

Notes: *Assuming 35 persons with an average of 250,000 CAD annual income.
** Deduction of drill and blast cost (0.75 of hard rock mining cost).
*** By a factor of 1.36.

Table 21-13: Total Mining operating costs – Open Pit and Underground Mines

Operating Cost															
Production year	Unit	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Kwanika Central Pit	C$M			-60129	-68361	-73423	-41223								
Kwanika Block Cave	C$M					-1788	-35256	-77523	-77518	-77519	-74536	-54296	-40120	-24554	-4569
Kwanika South Pit	C$M											-34024	-37394	-42053	-36360
Stardust	C$M						-54896	-67051	-68129	-68319	-65465	-22326			
Stockpile	C$M							-96	-89	-88	-318	-285	-5	-7	-57

21.3.4 Process Operating Costs

The process operating cost estimate is based on a 22,000 t/d mill consisting of grinding, rougher flotation and concentrate regrind, cleaner flotation, concentrate dewatering, and tailings handling. The operating cost estimates are summarized below in Table 21-14.

Table 21-14: Summary of Process Plant Operating Costs

Cost Centre	Total Cost (C$M/a)	Unit Cost (C$/t)	% of Total
Reagents & Consumables	33.1	4.12	39.6%
Maintenance Consumables	2.5	0.32	3.0%
Power	15.0	1.87	18.0%
Labour	18.2	2.26	21.8%
Mobile Equipment	0.8	0.10	0.9%
Total (LOM) – Plant Feed	**83.6**	**10.41**	**100%**

21.3.4.1 Reagents and Consumables

Reagents, grinding media, and various consumables are required to process the mineralized material from the Kwanika and Stardust deposits. The consumption rates of each of the consumable items are based on the metallurgical testwork outlined in Section 13 and based on the planned process plant throughput of 22,000 t/d. The total costs of the reagents and consumables by area are shown below in Table 21-15. A detailed breakdown of each reagent and consumable is presented in Table 21-15 and Table 21-16.

Table 21-15: Reagents and Consumables Cost Summary

Cost Centre	Total Cost (C$M/a)	Unit Cost (C$/t)
Primary Crushing	0.6	0.08
Grinding	22.8	2.84
Rougher Flotation	4.4	0.54
Concentrate Regrind	0.5	0.07
Cleaner Flotation	4.7	0.59
Total	**33.1**	**4.12**

Table 21-16: Reagents and Consumables Cost Breakdown

	Item	Consumption Rate (t/a)		Unit Cost (C$)		Cost (C$M/a)
Primary Crushing	Crusher liners - mantle	2	set/y	171,579	$/set	0.3
	Crusher liners - concaves	1	set/y	291,278	$/set	0.3
	Subtotal					**0.6**
Grinding	SAG mill media	2,112	t/y	1,979	$/t	4.2
	SAG mill liners	1,205	t/y	1,979	$/t	2.4
	Ball mill media	6,729	t/y	1,979	$/t	13.3
	Ball mill liners	1,124	t/y	1,979	$/t	2.2
	Lime to milling	1,606	t/y	423	$/t	0.7
	Subtotal					**22.8**
Rougher Flotation	Lime to roughers	3,292	t/y	423	$/t	1.4
	A208 to rougher	112	t/y	6,228	$/t	0.7
	3418A to rougher	80	t/y	17,618	$/t	1.4
	MIBC to rougher	217	t/y	3,992	$/t	0.9
	Subtotal					**4.4**
Concentrate Regrind	Regrind mill ceramic media	153	t/y	2,987	$/t	0.5
	Regrind mill liner	1.0	set/y	90,909	$/set	0.1
	Subtotal					**0.5**
Cleaner Flotation	Lime to cleaners	7,388	t/y	423	$/t	3.1
	3418A to cleaner	52	t/y	17,618	$/t	0.9
	MIBC to cleaner	169	t/y	3,992	$/t	0.7
	Flocculant to concentrate thickener	2	t/y	5,195	$/t	0.0
	Subtotal	-	-	-	-	**4.7**
	TOTAL	-	-	**-**	-	**33.1**

21.3.4.2 Maintenance Operating Costs

The maintenance costs were calculated by multiplying the total installed mechanical capital cost of each area by a factor from 3% to 5%. A summary of the maintenance costs is provided below in Table 21-17.

Table 21-17: Maintenance Consumables

WBS	Plant Area	Mechanical Equipment Cost (C$ M)	Factor (%)	Total Cost (C$M/a)	Unit Cost (C$/t)
2100	Primary Crushing	4.4	5.0%	0.2	0.03
2200	Stockpile and Reclaim	5.3	3.0%	0.2	0.02
2300	Grinding Circuit	54.6	3.0%	1.6	0.20
2400	Flotation and Regrind	13.0	3.0%	0.4	0.05
2500	Concentrate Handling	2.5	3.0%	0.1	0.01
2700	Tailings Pumping	1.9	3.0%	0.1	0.01
2900	Plant Services	0.1	3.0%	0.0	0.00
Maintenance Total Operating Costs		**81.8**	**3.1%**	**2.5**	**0.32**

21.3.4.3 Power Costs

Power operating costs are calculated from an estimate of annual power consumption using a unit cost of C$0.067/kWh. The annual power consumption for the processing plant is based on the average utilization of each motor on the electrical load list for the process plant. Table 21-18 gives a summary of installed electrical power, usage, and costs for each area.

Table 21-18: Power Operating Cost Summary

WBS	Plant Area	Installed Power (kW)	Operating Power (kW)	Energy Consumption (MWh/a)	Total Power Cost (C$M/a)	Unit Cost (C$/t)
2100	Primary Crushing	547	383	2,517	0.2	0.02
2200	Stockpile and Reclaim	443	310	2,499	0.2	0.02
2300	Grinding Circuit	28,445	19,911	160,468	10.8	1.35
2400	Rougher Flotation	1,440	1,008	8,126	0.5	0.07
2400	Concentrate Regrind	2,265	1,586	12,780	0.9	0.11
2400	Cleaner Flotation	912	6368	5,143	0.3	0.04
2500	Concentrate Thickening	18	12	100	0.0	0.001
2500	Concentrate Filtration	174	122	982	0.1	0.01
2800	Reagents	105	74	593	0.0	0.005
2900	Water Systems	366	256	2,065	0.1	0.02
2900	Plant Air Systems	820	574	4,626	0.3	0.04
2700	Tailings	1,492	1,044	8,417	0.6	0.07
5000	On-site and Off-site Infrastructure	2,311	1,618	13,037	0.9	0.11
2900	Tailings Pond	313	219	1,767	0.1	0.01
Electrical Total Operating Costs		**39,651**	**27,756**	**223,120**	**$ 15.0**	**1.87**

21.3.4.4 Labour Costs

Labour costs for the process plant were developed using benchmarks from similar projects, with salaries and hourly wages based on similar Canadian projects and expected local industrial rates. These labour costs do not include the camp stuff such as catering and housekeeping. Table 21-19 gives a summary of the process plant labour.

Table 21-19: Process Plant Labour Cost Summary

Category	Number of Employees	Total Labour Cost (C$M/a)	Unit Cost (C$/t)
General and Administration	23	4.2	0.52
Mill Staff	14	2.8	0.35
Mill Operators	26	4.6	0.57
Met Lab & Quality Control	14	2.1	0.26
Plant Maintenance	27	4.6	0.57
Labour Total Operating Costs	**104**	**18.2**	**2.26**

21.3.4.5 Vehicles

Light and mobile equipment is needed for G&A activities, process plant operations, and maintenance. The costs for fuel and service of these vehicles are given below in Table 21-20.

Table 21-20: Vehicle Operating Cost Breakdown

Vehicle Type	Used by/for	Fuel Cost (C$/a)	Spares Cost (C$/a)	Total Cost (C$/a)	Unit Cost (C$/t)
SUV	General Manager	8,059	2,579	$10,637	0.001
4x4 Crew Cab	Admin/HR Manager	8,059	2,579	$10,637	0.001
4x4 Crew Cab	Environmental	16,118	5,157	$21,275	0.003
4x4 Crew Cab	Social Community Coordinator	16,118	5,157	$21,275	0.003
4x4 Crew Cab	Procurement/Warehouse	32,236	10,314	$42,550	0.005
4x4 Crew Cab	HSEC Manager	16,118	5,157	$21,275	0.003
4x4 Crew Cab	Health & Safety	16,118	5,157	$21,275	0.003
Fire Engine	Health & Safety	5,373	1,842	$7,214	0.001
Ambulance	Health & Safety	3,582	1,289	$4,871	0.001
3t All-Terrain Forklift	Warehouse	42,981	15,471	$58,452	0.007
Grader, snow	Operations	26,863	15,471	$42,334	0.005
4x4 Crew Cab	Plant Manager	5,756	1,579	$7,335	0.001
4x4 Crew Cab	Chemistry/Met Lab	13,432	3,684	$17,115	0.002
4x4 Crew Cab	Ore Handling Crusher Operations	13,432	3,684	$17,115	0.002
4x4 Crew Cab	General Process	13,432	3,684	$17,115	0.002
Boom Lift	Operations	71,635	12,279	$83,914	0.010
Bobcat	Operations	20,147	6,139	$26,287	0.003
Front-End Loader	Operations	80,590	18,418	$99,008	0.012
Mobile Rock Breaker	Operations	20,147	4,604	$24,752	0.003
4x4 Crew Cab	Mechanical/Electrical Supervisor	26,863	7,367	$34,230	0.004
4x4 Crew Cab	Predictive/Instrumentation	26,863	7,367	$34,230	0.004
4x4 Crew Cab	General Maintenance	26,863	7,367	$34,230	0.004
Yard Crane	Maintenance	21,491	4,604	$26,095	0.003
Flat Bed Truck, 3.5t with crane	Maintenance	21,491	4,604	$26,095	0.003
Flat Bed Truck, 10t with crane	Maintenance	21,491	5,525	$27,016	0.003
Forklift, 2.5t	Maintenance	35,818	6,139	$41,957	0.005
Hydraulic Crane, 35t	Maintenance	53,726	19,339	$73,065	0.009
Vehicle Total Operating Costs		**637,937**	**171,085**	**$809,022**	**0.101**

21.3.5 General and Administrative Operating Costs

The general and administrative operating costs cover the expenses of the operating departments. A summary of the G&A costs is presented in Table 21-21.

General and administrative (G&A) costs were developed using Ausenco's in-house data on existing Canadian operations. The costs were estimated based on the following items:

- human resources (including recruiting, training, and community relations)
- infrastructure power (HVAC and administrative buildings)
- site administration, maintenance, and security (including office equipment, garbage disposal)
- assets operation (including non-operation-related vehicles) • health and safety (including personal protective equipment, hospital service cost) • environmental (including sampling and DSTF operation)
- IT and telecommunications (including hardware and support services)
- contract services (including insurance, sanitation, licence fees, and legal fees).

Table 21-21: G&A Cost Summary

G&A Expense	Total Cost (C$M/a)	Unit Cost ($/t)
General Administration Costs	1.4	0.17
Personnel Transportation	2.9	0.36
Catering and Housekeeping	9.1	1.00
Laboratory Costs	0.3	0.04
Insurance	1.3	0.16
Total	**14.0**	**1.74**

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

For the following economic analysis, it is necessary to mention that under Canadian securities law, the outcomes of the economic assessments mentioned in this section constitute forward-looking information. The preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized. The results of this economic analysis rely on inputs that could vary considerably from these predicated here due to known and unknowable risks, uncertainties, and other factors. The following is a list of forward-looking information:

- mineral resource estimates
- expected commodity prices and exchange rates
- the planned mine production plan
- estimated mining and process recovery rates
- expectations as to mining dilution and capability to mine in areas earlier exploited using mining methods as predicted the timing and amount of projected future production
- sustaining costs and proposed operating costs
- assumptions as to closure costs and closure requirements
- assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include the following:

- changes to costs of production from what is assumed
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- accidents, labour disputes, and other risks of the mining industry
- geotechnical or hydrogeological considerations during mining differing from previous assumptions
- failure of mining methods 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 license to operate
- changes to interest rates
- changes to tax rates.

22.2 Methodologies Used

The project has been evaluated using a discounted cashflow (DCF) analysis with a 7.0% discount rate. Cash inflows are generated from projections of annual revenue. Capital outlays, such as pre-production, operating costs, taxes, and royalties, make up the cash outflows. The annual cashflow predictions are produced by subtracting cash outflows from the inflows.

Cashflows are assumed to occur at the end of each period. Tax calculations involve complex variables that can only be precisely determined during operations; thus, the actual post-tax outcomes may differ from those estimated. A sensitivity analysis was conducted to determine the effects of changes in commodity price, discount rate, head grade, total operating cost, and initial capital cost variations. Section 21 of this report describes the project's capital and operating cost estimates. This economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

22.3.1 Assumptions

The economic analysis was conducted on the base case assumptions of US$3.63/lb of copper, US$1,650/oz of gold, US$21.5/oz of silver. The forecasts used are meant to reflect the average metals price expectation over the life of the project. The effects of price escalation or inflation were not considered in the economic analysis. There is a possibility that the forecast may differ from the predictions and that the price of the commodity may change.

The economic analysis also used the following assumptions:

- The construction period will be 2 years
- The mine life is 11.9 years
- Cost estimates are in constant Q4 2022 dollars with no inflation or escalation factors considered
- Results are based on 100% ownership with revenue from copper concentrate production
- There are no royalties that are applicable to the property
- Initial capital costs are funded with equity. There are financing costs associated with mine infrastructure and services, mobile equipment, and the permanent camp
- All cashflows are discounted to the start of the construction period using a full-period discounting convention
- All metal products will be sold in the same year that they are produced
- Project revenue will be derived from the sale of copper concentrate
- Currently, there are no contractual refining arrangements.

22.3.2 Taxes

The Kwanika-Stardust project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model used in this economic analysis was created by NorthWest Copper with assistance from third-party taxation experts where necessary.

At the effective date of this report, the project is assumed to be subject to the following tax regime:

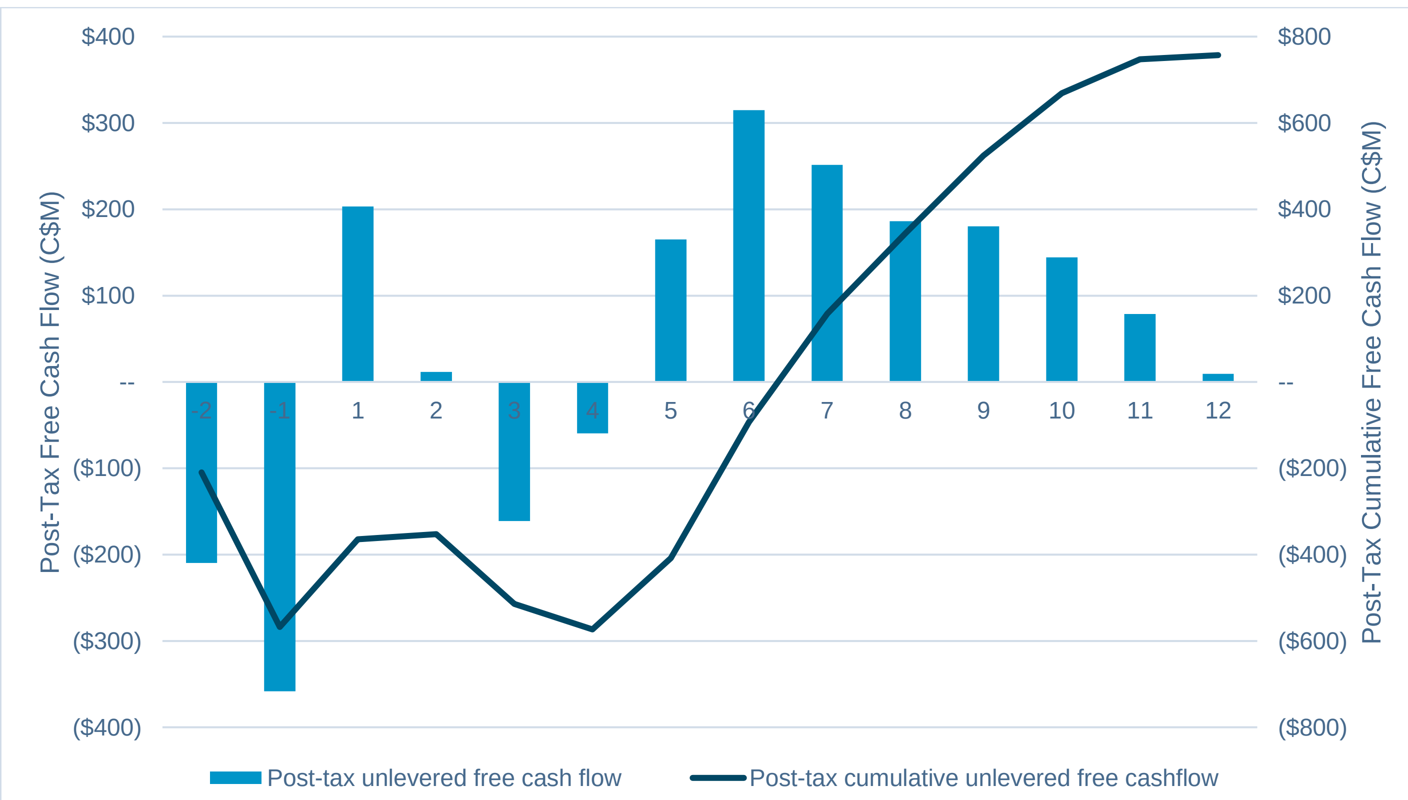
- Canadian federal income tax of 15%
- British Columbia provincial income tax of 12%
- British Columbia mineral tax of 2%

Opening tax pools including Canadian Exploration Expenses (CEE) and Canadian Development Expense (CDE) acquired on the acquisition of the project are included as applicable deductions in the tax model. The taxes in the model are calculated at a high level to provide a general idea of potential taxes and are anticipated to change as the economics of the project change. The total tax payments are estimated to be C$438.1 million over the life of mine.

22.4 Economic Analysis

The economic analysis utilized an assumed 7% discount rate. At a 7% discount rate, the pre-tax NPV is C$440.1 million, the IRR is 17.1%, and the payback period is 5.99 years. The post-tax NPV discounted at 7% is C$215 million, the IRR is 12.7%, and the payback period is 6.37 years. A summary of the project's economics is shown graphically in Figure 22-1 and listed in Table 22-1. This economic analysis was performed on an annual cashflow basis, the results of which is shown in Table 22-2.

Figure 22-1: Project Economics



Note: Figure prepared by Ausenco, 2022

Table 22-1: Economic Analysis Summary

Description	Unit	LOM Total / Avg.
General		
Copper Price	US$/lb	$3.63
Gold Price	US$/oz	$1,650
Silver Price	US$/oz	21.5
Exchange Rate	CAD:USD	0.77
Mine Life	Years	11.9
Total Mineralized Material Processed	kt	95,607
Total Waste	kt	86,926
Strip Ratio – Kwanika Central OP	waste tonnes:ore tonnes	1.9
Strip Ratio – Kwanika South OP	waste tonnes:ore tonnes	1.7
Production		
Average Feed Grade, Cu	%	0.39
Average Feed Grade, Au	g/t	0.39
Average Feed Grade, Ag	g/t	2.21
Average Open pit Mill Recovery Rate, Cu	%	84.3
Average Open pit Mill Recovery Rate, Au	%	60.0
Average Open pit Mill Recovery Rate, Ag	%	57.8
Average Underground Mill Recovery Rate, Cu	%	89.7
Average Underground Mill Recovery Rate, Au	%	71.4
Average Underground Mill Recovery Rate, Ag	%	70.3
Total Payable Copper	mlbs	694
Total Payable Gold	koz	803
Total Payable Silver	koz	3,204
Total Payable Copper Equivalent	mlbs	1,078
Operating Costs		
Mining Cost	C$/t Mined	$6.62
Mining Cost	C$/t Milled	$12.63
Processing Cost	C$/t Milled	$8.13
G&A Cost	C$/t Milled	$2.28
Refining and Transport Cost	C$/lb of Cu Eq.	$0.27
Cash Cost	C$/lb of Cu Eq.	$2.6
All-in Sustaining Costs	C$/lb of Cu Eq.	$1.5
Capital Costs		
Initial Capital	C$M	$567.9
Sustaining Capital	C$M	$282.5
Growth Capital	C$M	$493.3
Closure Costs	C$M	$41.9
Salvage Costs	C$M	($2.5)
Financials		
Pre-Tax NPV (7%)	C$M	$440.1
Pre-Tax IRR	%	17.1%
Pre-Tax Payback (Years)	Years	5.99
Post-Tax NPV (7%)	C$M	$215.0
Post-Tax IRR	%	12.7%
Post-Tax Payback (Years)	Years	6.37

Table 22-2: Cashflow Statement on an Annualized Basis

	Units	Total/Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Total Revenue	**C$M**	**$5,068**	**--**	**--**	**$425**	**$328**	**$286**	**$280**	**$539**	**$715**	**$620**	**$522**	**$493**	**$401**	**$279**	**$180**
Operating Costs	C$M	($2,203)	--	--	($144)	($152)	($159)	($215)	($228)	($229)	($230)	($224)	($195)	($161)	($150)	($117)
Treatment & Refining Charges, Penalties and Transportation	C$M	($287)	--	--	($24)	($20)	($18)	($16)	($29)	($35)	($31)	($28)	($28)	($26)	($19)	($13)
Royalties	C$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
EBITDA	**C$M**	**$2,578**	**--**	**--**	**$258**	**$156**	**$109**	**$49**	**$281**	**$451**	**$359**	**$270**	**$270**	**$214**	**$110**	**$50**
Initial Capex (incl. Capitalized Opex)	C$M	($568)	($210)	($358)	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	C$M	($776)	--	--	($49)	($141)	($268)	($107)	($110)	($55)	($27)	($6)	($6)	($2)	($2)	($0)
Closure Costs	C$M	($42)	--	--	--	--	--	--	--	--	--	--	--	--	--	($42)
Salvage Value	C$M	$2	--	--	--	--	--	--	--	--	--	--	--	--	--	$2
Pre-Tax Unlevered Free Cash Flow	**C$M**	**$1,195**	**($210)**	**($358)**	**$209**	**$15**	**($159)**	**($58)**	**$171**	**$396**	**$332**	**$265**	**$264**	**$212**	**$108**	**$10**
Post-Tax Unlevered Free Cash Flow	**C$M**	**$757**	**($210)**	**($358)**	**$203**	**$12**	**($161)**	**($59)**	**$165**	**$315**	**$252**	**$186**	**$180**	**$144**	**$79**	**$9**
Mining																
Mineralized Material Mined	kt	95,607	--	--	8,031	8,030	8,203	9,123	7,902	7,911	7,913	7,607	7,650	8,023	8,021	7,193
Waste	kt	86,926	--	10,000	13,168	13,520	12,967	5,559	--	--	--	--	10,663	8,753	7,291	5,004
Mined Resource Grades																
Copper	wt%	0.39%	--	--	0.40%	0.33%	0.30%	0.26%	0.46%	0.55%	0.50%	0.46%	0.47%	0.42%	0.32%	0.25%
Gold	g/t	0.39	--	--	0.4	0.3	0.2	0.2	0.5	0.8	0.7	0.5	0.4	0.3	0.2	0.1
Silver	g/t	2.21	--	--	1.6	1.1	0.9	1.9	3.4	4.1	3.3	3.2	2.4	1.9	1.6	1.3
Copper Equivalent	wt%	0.6%	--	--	0.6%	0.5%	0.4%	0.4%	0.8%	1.0%	0.9%	0.8%	0.7%	0.6%	0.4%	0.3%
Milling and Concentrate Production																
Resource Milled	kt	95,607	--	--	8,031	8,030	8,035	8,033	8,030	8,030	8,030	8,030	8,030	8,030	8,030	7,268
Concentrate Production																
Cu Concentrate Produced	kt	1,312.0	--	--	110	90	81	74	132	158	143	129	129	118	87	61
Revenue																
Copper Revenue	C$M	$3,263	--	--	$274	$223	$202	$185	$328	$393	$355	$320	$321	$294	$216	$152
Gold Revenue	C$M	$1,715	--	--	$146	$101	$82	$88	$198	$307	$253	$192	$164	$101	$58	$24
Silver Revenue	C$M	$89	--	--	$5	$3	$2	$7	$12	$15	$12	$11	$8	$6	$5	$3
Total Revenue	**C$M**	**$5,068**	**--**	**--**	**$425**	**$328**	**$286**	**$280**	**$539**	**$715**	**$620**	**$522**	**$493**	**$401**	**$279**	**$180**
Operating Costs																
Mining	C$M	$1,208	--	--	$60.1	$68.4	$75.2	$131.4	$144.7	$145.7	$145.9	$140.3	$110.9	$77.5	$66.6	$41.0
Processing	C$M	$777	--	--	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$65.3	$59.1
G&A	C$M	$218	--	--	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$18.3	$16.6
Total Mine site Operating Costs	**C$M**	**$2,203**	**--**	**--**	**$144**	**$152**	**$159**	**$215**	**$228**	**$229**	**$230**	**$224**	**$195**	**$161**	**$150**	**$117**
Treatment & Refining Charges, Penalties and Transportation																
Treatment & Refining Charges	C$M	$156	--	--	$13	$11	$10	$9	$16	$19	$17	$15	$15	$14	$10	$7
Transportation	C$M	$131	--	--	$11	$9	$8	$7	$13	$16	$14	$13	$13	$12	$9	$6
Total Treatment & Refining Charges, Penalties and Transportation	**C$M**	**$287**	**--**	**--**	**$24**	**$20**	**$18**	**$16**	**$29**	**$35**	**$31**	**$28**	**$28**	**$26**	**$19**	**$13**
All-in Sustaining Costs	**US$/lb CuEq**	**$2.37**	**--**	**--**	**$1.85**	**$3.47**	**$5.65**	**$4.39**	**$2.48**	**$1.62**	**$1.69**	**$1.79**	**$1.69**	**$1.71**	**$2.23**	**$3.42**
Capital Expenditures																
Initial Capital Cost																
Mining	C$M	$66	$9	$57	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	C$M	$198	$79	$119	--	--	--	--	--	--	--	--	--	--	--	--
Additional Process Facilities	C$M	$6	$3	$4	--	--	--	--	--	--	--	--	--	--	--	--
On-Site Infrastructure	C$M	$22	$9	$13	--	--	--	--	--	--	--	--	--	--	--	--
Off-Site Infrastructure	C$M	$83	$33	$50	--	--	--	--	--	--	--	--	--	--	--	--
Preliminaries	C$M	$28	$11	$17	--	--	--	--	--	--	--	--	--	--	--	--
Delivery	C$M	$50	$20	$30	--	--	--	--	--	--	--	--	--	--	--	--

	Units	Total/Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Owner's Costs	C$M	$34	$13	$20	--	--	--	--	--	--	--	--	--	--	--	--
Provisions	C$M	$81	$32	$49	--	--	--	--	--	--	--	--	--	--	--	--
Total Initial Capital	**C$M**	**$568**	**$210**	**$358**												
Sustaining Capital Cost																
Mining Sustaining Capital	C$M	$149	--	--	--	--	$2	--	$81	$47	$7	$4	$4	$2	$2	--
Mine Office Financing	C$M	$2	--	--	$0	$0	$0	$0	$0	--	--	--	--	--	--	--
Mobile Equipment Financing	C$M	$6	--	--	$1	$1	$1	$1	$1	--	--	--	--	--	--	--
On-Site Infrastructure	C$M	$5	--	--	--	$0	$0	--	$2	--	--	$1	$2	--	--	--
Off-Site Infrastructure	C$M	$38	--	--	--	--	$21	--	--	--	$16	--	--	--	--	--
Camp Financing	C$M	$41	--	--	$8	$8	$8	$8	$8	--	--	--	--	--	--	--
Total Indirects	C$M	$42	--	--	$1	$1	$7	$1	$18	$9	$4	$1	$1	$0	$0	$0
Total Sustaining Capital	**C$M**	**$282**	**--**	**--**	**$11**	**$11**	**$41**	**$11**	**$110**	**$55**	**$27**	**$6**	**$6**	**$2**	**$2**	**$0**
Growth Capital Cost																
Mine Infrastructure and Services	C$M	$17	--	--	$1	$5	$11	$0	--	--	--	--	--	--	--	--
Underground Mining	C$M	$376	--	--	$25	$97	$174	$80	--	--	--	--	--	--	--	--
Total Indirects	C$M	$100	--	--	$12	$28	$43	$17	--	--	--	--	--	--	--	--
Total Growth Capital	**C$M**	**$493**	**--**	**--**	**$38**	**$130**	**$228**	**$97**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Closure Cost																
Closure Cost	C$M	$42	--	--	--	--	--	--	--	--	--	--	--	--	--	$42
Total Closure Cost	**C$M**	**$42**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**$42**
Salvage Value																
Salvage Value	C$M	($2)	--	--	--	--	--	--	--	--	--	--	--	--	--	($2)
Total Salvage Value	**C$M**	**($2)**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**($2)**
Total Capital Expenditures Including Salvage Value	**C$M**	**$1,383**	**$210**	**$358**	**$49**	**$141**	**$268**	**$107**	**$110**	**$55**	**$27**	**$6**	**$6**	**$2**	**$2**	**$40**

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the project using the following variables: operating cost, capital cost, commodity price, head grade, recovery, mining cost, and foreign exchange rate. The pre-tax sensitivity analysis is shown in Table 22-3 and the post-tax sensitivity analysis is shown in Table 22-4.

The pre-tax sensitivities are shown graphically below in Figure 22-1 and Figure 22-2. The pre-tax economics are most sensitive to commodity price and foreign exchange. The post-tax sensitivities are presented in Figure 22-3 and Figure 22-4. The post-tax NPV are most sensitive to commodity price and foreign exchange, while the post-tax IRR is most sensitive to commodity price and foreign exchange.

Table 22-3: Pre-Tax Sensitivity Analysis (NPV and IRR) to Discount Rate, Capex, OPEX, Head Grade and FX Rate

		Commodity Price							Commodity Price				
Discount Rate		-20%	-10%	0%	10%	20%	Discount Rate		-20%	-10%	0%	10%	20%
	3.0%	$4	$401	$799	$1,196	$1,593		3.0%	3.1%	10.6%	17.1%	23.0%	28.6%
	5.0%	($82)	$259	$600	$940	$1,281		5.0%	3.1%	10.6%	17.1%	23.0%	28.6%
	7.0%	($148)	$146	$440	$734	$1,028		8.0%	3.1%	10.6%	17.1%	23.0%	28.6%
	10.0%	($219)	$19	$257	$495	$733		10.0%	3.1%	10.6%	17.1%	23.0%	28.6%
	12.0%	($253)	($44)	$164	$372	$581		12.0%	3.1%	10.6%	17.1%	23.0%	28.6%

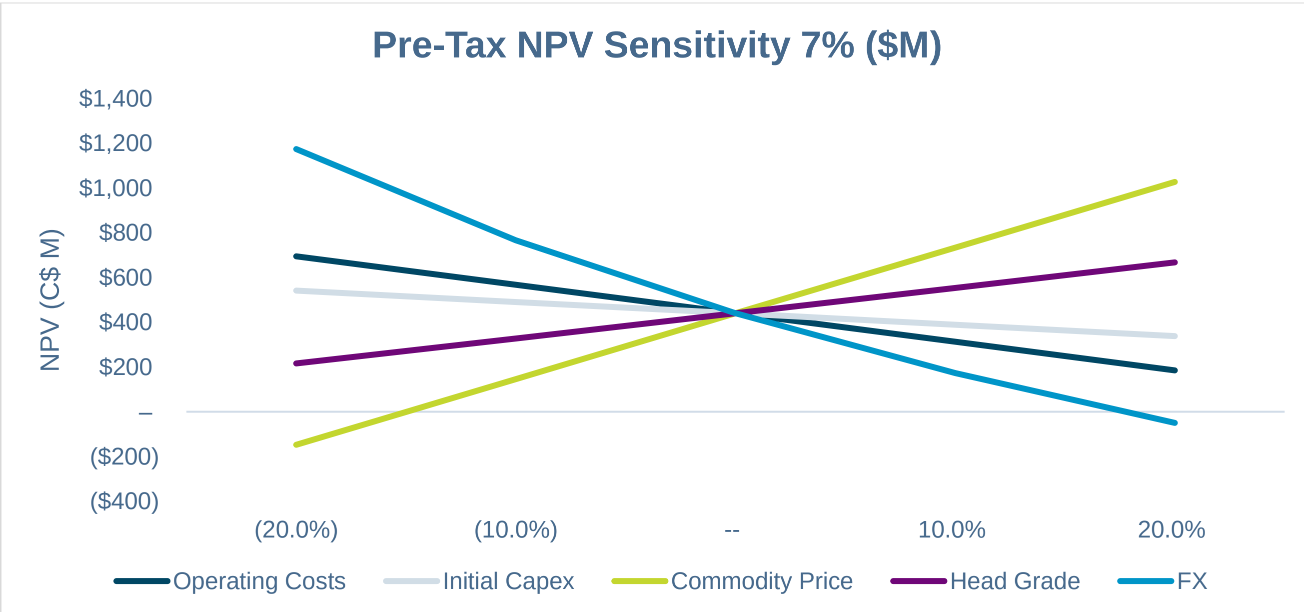
		Commodity Price							Commodity Price				
Initial Capital Cost		-20%	-10%	0%	10%	20%	Initial Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	($46)	$248	$542	$836	$1,130		(20%)	5.6%	13.9%	21.2%	28.1%	34.6%
	(10%)	($97)	$197	$491	$785	$1,079		(10%)	4.3%	12.1%	19.0%	25.3%	31.3%
	--	($148)	$146	$440	$734	$1,028		--	3.1%	10.6%	17.1%	23.0%	28.6%
	10%	($199)	$95	$389	$683	$977		10%	2.0%	9.2%	15.4%	21.0%	26.3%
	20%	($249)	$44	$338	$632	$926		20%	1.0%	8.0%	13.9%	19.3%	24.2%

		Commodity Price							Commodity Price				
Operating Capital Cost		-20%	-10%	0%	10%	20%	Operating Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	$107	$401	$695	$989	$1,283		(20%)	9.6%	16.2%	22.2%	27.9%	33.2%
	(10%)	($20)	$274	$568	$862	$1,155		(10%)	6.5%	13.5%	19.7%	25.5%	30.9%
	--	($148)	$146	$440	$734	$1,028		--	3.1%	10.6%	17.1%	23.0%	28.6%
	10%	($275)	$19	$313	$606	$900		10%	0.0%	7.5%	14.3%	20.5%	26.2%
	20%	($403)	($109)	$185	$479	$773		20%	0.0%	4.1%	11.5%	17.9%	23.8%

		Commodity Price							Commodity Price				
Head Grade		-20%	-10%	0%	10%	20%	Head Grade		-20%	-10%	0%	10%	20%
	(20%)	($324)	($54)	$216	$486	$756		(20%)	0.0%	5.7%	11.7%	17.0%	21.8%
	(10%)	($236)	$46	$328	$609	$891		(10%)	0.8%	8.1%	14.3%	19.9%	25.0%
	--	($148)	$146	$440	$734	$1,028		--	3.1%	10.6%	17.1%	23.0%	28.6%
	10%	($58)	$248	$554	$860	$1,166		10%	5.4%	13.2%	20.0%	26.4%	32.5%
	20%	$32	$350	$668	$986	$1,304		20%	7.9%	15.9%	23.2%	30.1%	36.7%

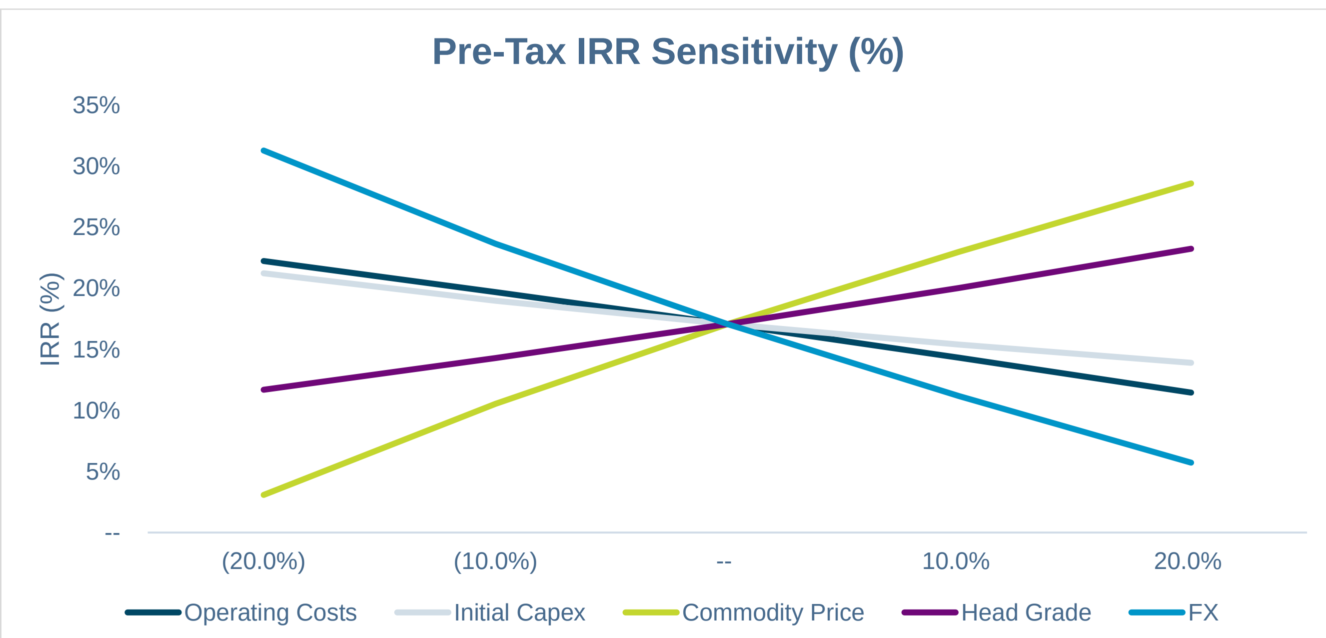
		Commodity Price							Commodity Price				
FX		-20%	-10%	0%	10%	20%	FX		-20%	-10%	0%	10%	20%
	(20%)	$440	$807	$1,175	$1,542	$1,910		(20%)	17.1%	24.4%	31.3%	37.8%	44.1%
	(10%)	$114	$440	$767	$1,093	$1,420		(10%)	9.8%	17.1%	23.6%	29.8%	35.7%
	--	($148)	$146	$440	$734	$1,028		--	3.1%	10.6%	17.1%	23.0%	28.6%
	10%	($361)	($94)	$173	$440	$707		10%	0.0%	4.5%	11.2%	17.1%	22.5%
	20%	($540)	($295)	($50)	$195	$440		20%	0.0%	0.0%	5.7%	11.7%	17.1%

Figure 22-2: Pre-Tax NPV Sensitivity Chart



Source: Ausenco, 2022

Figure 22-3: Pre-Tax IRR Sensitivity Chart



Source: Ausenco, 2022

Table 22-4: Post-Tax Sensitivity Analysis (NPV and IRR) to Discount Rate, Capex, OPEX, Head Grade and FX Rate

		Commodity Price				
Discount Rate		-20%	-10%	0%	10%	20%
	3.0%	($53)	$213	$472	$728	$982
	5.0%	($129)	$104	$329	$550	$770
	7.0%	($187)	$18	$215	$408	$598
	10.0%	($249)	($78)	$85	$244	$399
	12.0%	($278)	($126)	$19	$159	$297

		Commodity Price				
Discount Rate		-20%	-10%	0%	10%	20%
	3.0%	1.9%	7.5%	12.7%	17.5%	22.0%
	5.0%	1.9%	7.5%	12.7%	17.5%	22.0%
	8.0%	1.9%	7.5%	12.7%	17.5%	22.0%
	10.0%	1.9%	7.5%	12.7%	17.5%	22.0%
	12.0%	1.9%	7.5%	12.7%	17.5%	22.0%

		Commodity Price				
Initial Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	($103)	$97	$292	$483	$673
	(10%)	($144)	$58	$254	$445	$636
	--	($187)	$18	$215	$408	$598
	10%	($231)	($22)	$176	$370	$561
	20%	($275)	($62)	$136	$332	$524

		Commodity Price				
Initial Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	3.7%	10.0%	15.9%	21.4%	26.7%
	(10%)	2.8%	8.7%	14.2%	19.3%	24.2%
	--	1.9%	7.5%	12.7%	17.5%	22.0%
	10%	1.1%	6.4%	11.3%	15.9%	20.2%
	20%	0.3%	5.5%	10.2%	14.5%	18.6%

		Commodity Price				
Operating Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	($8)	$189	$383	$573	$764
	(10%)	($95)	$104	$300	$491	$681
	--	($187)	$18	$215	$408	$598
	10%	($286)	($69)	$130	$325	$516
	20%	($408)	($159)	$44	$241	$433

		Commodity Price				
Operating Capital Cost		-20%	-10%	0%	10%	20%
	(20%)	6.8%	12.0%	16.9%	21.4%	25.8%
	(10%)	4.4%	9.8%	14.8%	19.5%	23.9%
	--	1.9%	7.5%	12.7%	17.5%	22.0%
	10%	0.0%	5.1%	10.4%	15.5%	20.1%
	20%	0.0%	2.6%	8.2%	13.3%	18.1%

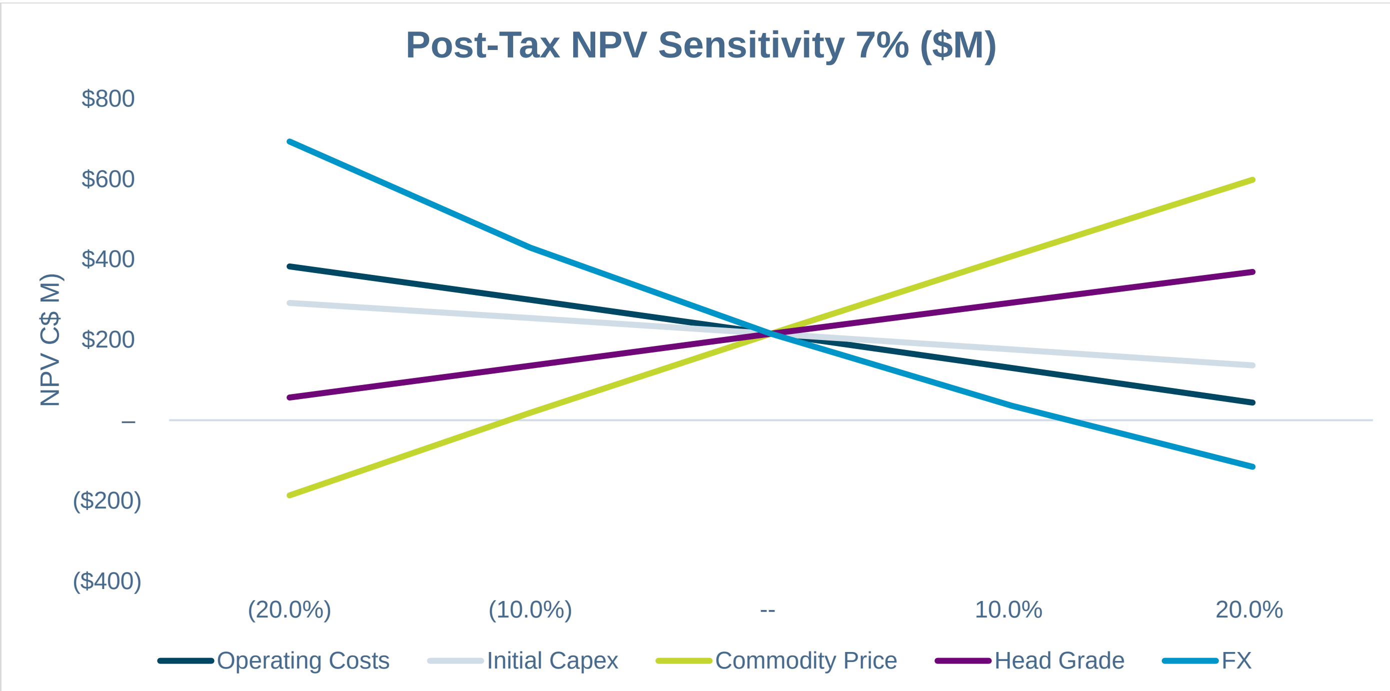
		Commodity Price				
Head Grade		-20%	-10%	0%	10%	20%
	(20%)	($331)	($127)	$56	$236	$415
	(10%)	($258)	($54)	$136	$323	$508
	--	($187)	$18	$215	$408	$598
	10%	($118)	$90	$293	$491	$690
	20%	($53)	$161	$369	$576	$782

		Commodity Price				
Head Grade		-20%	-10%	0%	10%	20%
	(20%)	0.0%	3.8%	8.4%	12.6%	16.6%
	(10%)	0.1%	5.6%	10.4%	15.0%	19.2%
	--	1.9%	7.5%	12.7%	17.5%	22.0%
	10%	3.6%	9.5%	15.0%	20.1%	25.1%
	20%	5.4%	11.7%	17.5%	23.0%	28.4%

		Commodity Price				
FX		-20%	-10%	0%	10%	20%
	(20%)	$215	$456	$694	$931	$1,168
	(10%)	($4)	$215	$429	$641	$852
	--	($187)	$18	$215	$408	$598
	10%	($370)	($147)	$36	$215	$390
	20%	($541)	($305)	($116)	$51	$215

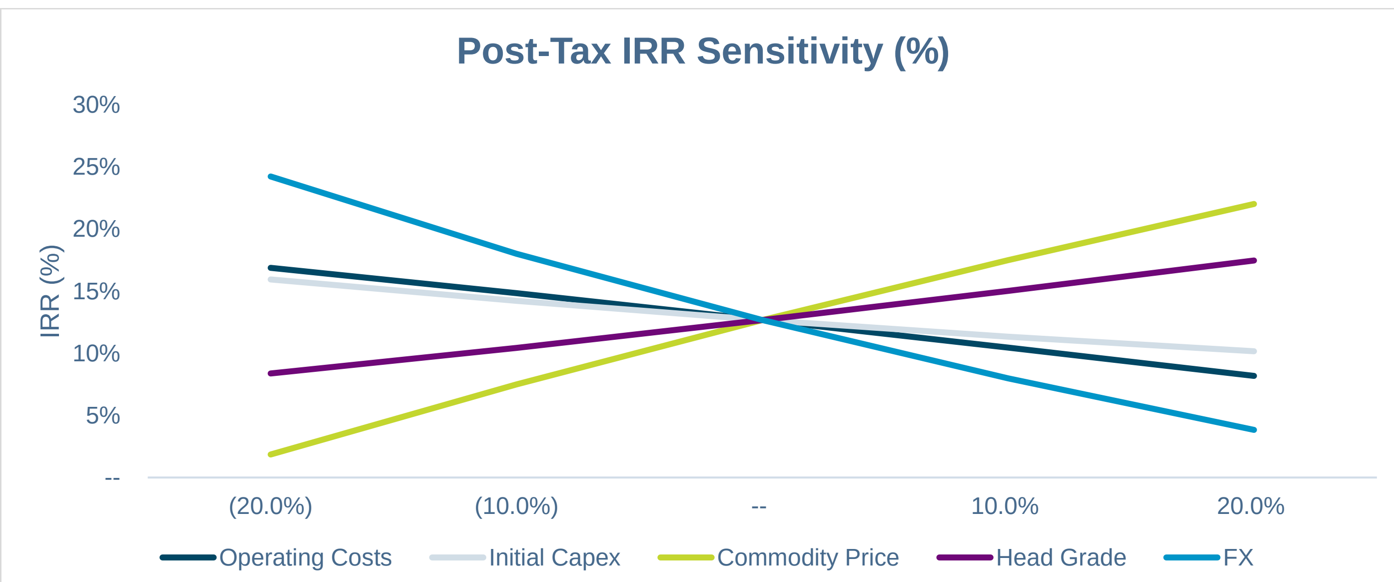
		Commodity Price				
FX		-20%	-10%	0%	10%	20%
	(20%)	12.7%	18.6%	24.2%	29.7%	34.9%
	(10%)	6.9%	12.7%	18.0%	23.0%	27.9%
	--	1.9%	7.5%	12.7%	17.5%	22.0%
	10%	0.0%	3.0%	8.0%	12.7%	17.1%
	20%	0.0%	0.0%	3.8%	8.4%	12.7%

Figure 22-4: Post-Tax NPV Sensitivity Chart



Source: Ausenco, 2022

Figure 22-5: Post-Tax IRR Sensitivity Chart



Source: Ausenco, 2022

22.6 Comments on Economic Analysis

The economic analysis was performed with an assumed discount rate of 7%. The pre-tax NPV discounted at 7% is C$440.1 million; the IRR is 17.1%, and the payback period is 5.99 years. On a post-tax basis, the NPV discounted at 7% is C$215.0 million, the IRR is 12.7%, and the payback period is 6.37 years.

23 ADJACENT PROPERTIES

The Quesnel Trough is the host to several other porphyry copper ± gold mines and significant deposits. These deposits include: the Mount Polley Mine, the former Kemess Mine and its related infrastructure located north of Kwanika, and the Mount Milligan Mine located around 85 km south of Kwanika.

Figure 23-1: Map showing the Kwanika project location and other mines in the region

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Schedule

The project execution schedule addresses the overall project (objectives, scope, strategies, and roles and responsibilities) and provides a high-level plan for the development and implementation of the project. The schedule covers the plan for studies, site works, engineering, procurement, construction, start-up and commissioning of the project.

NorthWest Copper 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.

24.1.1 Execution Strategy

The execution schedule activities include the following:

- bridging phase
- studies and site work
- permitting
- engineering procurement construction management (EPCM)
- constraints and limitations

24.1.2 Studies and Site Work

24.1.2.1 Bridging Phase

During the bridging phase prior to the commencement of the prefeasibility study, the focus will be on obtaining the necessary information to successfully execute the prefeasibility phase and environmental impact assessment works.

This information will be obtained through the following:

- trade-off studies
- infill drilling
- condemnation drilling
- geotechnical drilling
- metallurgical test program
- geochemical characterization
- hydrogeological characterization
- hydrology analysis
- environmental characterization.

24.1.2.2 Prefeasibility Study

During the prefeasibility study, the engineering process will evaluate the technical and economic viability of a mining project using the updated resource model based on infill drilling (validate reserve). Construction and operation of the mine, plant, and associated infrastructure may not take place until environmental permits have been received. The work performed during the prefeasibility phase will therefore be aligned to support the provincial and federal EIA procedures.

24.1.2.3 Feasibility Study

A comprehensive technical and economic feasibility study will be developed to serve as the basis to decide whether to proceed with the mine plans. Activities during this phase will include a comprehensive analysis of the mineral deposit, updated budget pricing for all major equipment, development of design and drawings to support MTO, definition of all supporting infrastructure, development of a detailed execution schedule and plan, and development of a detailed capital cost estimate.

24.1.2.4 Engineering Procurement Construction Management (EPCM)

The EPCM contractor will provide a complete and fully functional process plant and other on-site infrastructure as defined in Section 18 by performing the services below.

- The engineering and design required for the construction of the facilities will be completed. Design for construction will include all engineering disciplines such as civil, structural, architectural, mechanical, piping, electrical, instrumentation and control.
- All materials, goods, and services will be procured to construct and commission the process plant. This includes the procurement of commissioning spare parts at the time of equipment procurement.
- Logistics management, warehousing and preservation of all procured materials and goods will be provided prior to issue to construction contractors.
- All work within the defined scope will be managed in accordance with the project execution plan and all other project plans, to achieve the project schedule and budget.

- A project controls system will be implemented to adequately monitor and report on project progress, including adherence to or deviation from the schedule and the budget. Monthly Project Progress reports will be provided to thoroughly explain project progress.
- Engineering and supervisory support will be provided for the process plant from start-up through to final completion.

The construction effort will follow a strategy of working on numerous work fronts concurrently. Work fronts to include optimal sequencing to be determined for early works activities such as tree clearing and grubbing, access roads and site roads and excavation. The site layout allows sufficient room for laydowns within the processing site following site preparation.

The contractor will commence with infrastructure development, pre-stripping, overburden removal and storage, laydown areas, haul roads, access roads, deep undergrounds, fire protection, water and sewer services, buried permanent and temporary electrical services, substation and electrical distribution, piling and then work out of the ground. As civil work completes the concrete foundations, piers and pedestals will be placed.

24.1.3 Constraints and Limitations

No allowance for inclement weather or other time-delay for risk events has been included; however, care should be taken to avoid critical tasks for non-winter months (such as concrete pours and welding), and to avoid bird nesting windows for tree clearing activities.

Due to potential issues associated with the ground conditions in wet areas, bulk earthworks is advised to be undertaken during the winter months when the ground is frozen. Construction activities will be sequenced to consider the northern weather conditions and outdoor construction windows.

A construction schedule for the project is shown in Table 24-1.

Table 24-1: Construction Schedule for The Kwanika-Stardust Project

Task Name	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Bulk Earthworks																								
Detailed Earthworks																								
Concrete Works																								
Construction																								
Commissioning																								

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

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

The Kwanika deposit is located in North-Central British Columbia, in the Omineca Mining Division, around 140 km northwest (around 200 km by road) of Fort St. James (Figure 4-1). The project area is on NTS map sheets 93N06 and 93N11, at latitude 55.53º N and longitude 125.35º W.

NorthWest Copper owns a 100% interest in the project, which is situated amongst a group of 59 unpatented mineral claims covering an area of 24,152.04 ha. The property is not subject to any royalties or other outstanding liabilities.

The property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

NorthWest Copper, indirectly through its wholly owned subsidiary Tsayta Resources Corp. (Tsayta), owns a 100% interest in the Stardust property. The Stardust property encompasses 26 mineral claims covering 12,932.39 ha. There are no title encumbrances, surface rights issues or legal access obligations that must be met in order for NorthWest Copper to retain the property. The Stardust property is not subject to any royalty terms, back-in rights, payments or any other agreements or encumbrances.

There are no other known factors or issues that materially affect the project other than normal risks faced by mining projects in the province of British Colombia in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors. Mining Plus is not aware of any known legal or title issues that would materially affect the project's potential economic viability.

25.3 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The Kwanika resource estimate is based on 166 drillholes for the Central Zone and 62 drillholes for the South Zone. The Stardust resource estimate is based on results from 186 diamond drillholes. The adequacy of sample preparation, security and analytical procedures are sufficiently reliable to support a mineral resource estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices at the time of collection.

25.4 Mineral Processing and Metallurgical Testwork

The metallurgical testwork conducted on Kwanika and Stardust samples has shown that both deposits are amenable to conventional sulphide flotation. Subsequent cleaner flotation can produce a saleable concentrate with acceptable cleaner circuit recoveries.

The key conclusions of the testwork conducted include:

- The composites tested were of moderate hardness with respect to grinding in a SAG mill, returning Axb values averaging about 55. The composites ranged from moderately hard to hard with respect to ball milling, as Bond ball mill Wi index values of 16.2 to 22.4 kWh/tonne were measured.
- The deposits host relatively fine-grained copper that benefits from a relatively fine primary grind size and regrind of rougher flotation concentrate.
- A primary grind sizing of 100 µm P_{80} may provide an acceptable degree of liberation for bulk rougher flotation.
- Cleaner flotation on reground rougher flotation concentrate can produce a saleable copper concentrate.
- Open pit LOM average recoveries for copper and gold are expected to be 84.3% and 60.0% respectively.
- Underground LOM average recoveries for copper and gold are expected to be 89.7% and 71.4% respectively.

25.5 Mineral Resource Estimates

25.5.1 Kwanika

Mining Plus has prepared the Mineral Resource estimate for the Kwanika property. The following observations and conclusions were drawn:

- The geology and mineralization at the Kwanika property are sufficiently well understood to develop 3D models and support estimation of mineral resources.
- Sample preparation, security and analytical procedures are adequate to support mineral resource estimation. Sample preparation, analysis, and security are generally performed in accordance with exploration best practices.
- The database used for the Kwanika resource estimate comprises data from exploration drilling conducted between 2006 and 2021. Drilling on the Central Zone totalled 76,156 m in 166 holes. A total of 29,431 core samples were submitted to the lab for analysis. Drilling on the South Zone totalled 19,099 m in 62 holes. A total of 8,490 core samples were submitted to the lab for analysis.
- Significant mineralization exists outside of the constraining pit shell and underground block cave shape that is currently not included in the mineral resource. This mineralization currently does not exist in a sufficient quantity or continuity to satisfy reasonable prospects for eventual economic extraction.

Areas of uncertainty that may materially impact the potential economic viability or continued viability of the project include:

- Commodity price assumptions
- Assumptions that all required permits will be forthcoming
- Metallurgical recoveries
- Mining and process cost assumptions
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

There are no other known factors or issues that materially affect the Kwanika property other than normal risks faced by mining projects in the province of British Colombia in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors. Mining Plus is not aware of any known legal or title issues that would materially affect the property potential economic viability.

25.5.2 Stardust

Geosim has prepared a Mineral Resource estimate for the Stardust Property CCS zone. The following observations and conclusions were drawn:

- The Canyon Creek zone is a skarn-hosted mineral occurrence hosted by Permian Cache Creek group sedimentary rocks in proximity to the Glover stock. The presently defined mineralized zones extend around 1,200 m along strike and 1,000 m downdip.
- The sample preparation, security and analytical procedures are sufficiently reliable to support an Indicated and Inferred mineral resource estimation and that sample preparation, analysis, and security were generally performed in accordance with exploration best practices at the time of collection.
- The resource estimate is based on analytical data from 206 drillholes representing 80,700 m of drilling. Fifty-eight of these holes (38,329 m) were completed in the most recent drill programs carried out in 2018, 2019 and 2020. Block grade estimation is based on samples from 186 of these drillholes.
- Statistical analysis of gold grade distribution indicates that cutting or capping of high grades is warranted.
- There is significant potential for expanding the current resource and for discovering additional mineral deposits on the property and extensions to known mineral showings.

Areas of uncertainty that may materially impact the property potential economic viability or continued viability include:

- commodity price assumptions;
- assumptions that all required permits will be forthcoming;
- metallurgical recoveries;
- mining and process cost assumptions; and
- ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

There are no other known factors or issues that materially affect the property other than normal risks faced by mining projects in the province of British Colombia in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors. Geosim is not aware of any known legal or title issues that would materially affect the property potential economic viability.

25.6 Mine Plan

25.6.1 Geotechnical Considerations

The Kwanika geotechnical dataset is considered adequate for conceptual PEA-level designs for both open pit and underground. Preliminary geotechnical investigations have provided valuable guidance on ground support strategies and, caveability/subsidence assessments. Additional work is required to upgrade the geotechnical understanding of the rockmass to complete a PFS study.

The Kwanika Central and South open pit designs are based on the current geotechnical dataset to provide suitable guidance for pit slope angles for PEA-level planning. Collection of additional geotechnical and hydrogeological information is recommended to optimize the existing designs, which are conservative in nature.

The Stardust underground geotechnical dataset is limited, thereby requiring referential estimates to be made to guide stope sizing parameters and ground support strategies. The development of rock-type specific geotechnical information for the Stardust deposit is a high priority target prior to advancing to a PFS-level study.

25.6.2 Kwanika Underground Mine

The Kwanika Central Block Cave mine produces 44 Mt mill feed with an average NSR value of 56.79 $/t. This mine utilizes the block cave mining method and operating cost equal to 10.62 $/t inclusive of mining, transportation, and G&A. This mine commences production in year 3 of the project and is the predominant feed source for years 4 through 9 at a maximum throughput of 20,000 t/d.

25.6.3 Stardust Underground Mine

The Stardust mine consists of one underground mine 7 km away from the Kwanika underground mine. The overall production in the 7-year mine life is 3.1 Mt at 195.41 $/t NSR. The mine will be responsible for a steady state production of 581 kt/a. The selected mining method is sublevel, open stoping with a combination of longitudinal and transverse stopes. The mine schedule is based upon rapid lateral development rates provided by contractor development.

25.6.4 Open Pit Mines

This project includes 2 open pit mines named Kwanika Central Pit and South Pit. Both OP operations are designed as contractor-operated in this PEA.

Kwanika Central Pit produces 29.4 Mt mill feed with an average NSR value of 36.74 $/t and a strip ratio of 1.87. After 1 year of pre-stripping at year -1, this mine is the only source of mill throughput in the production year of 1 to 3 and also the majority of mill feed in Year 4.

Kwanika South Pit produces 19.1 Mt mill feed with an average NSR value of 23.4 $/t and a strip ratio of 1.66. This mine will supplement the Kwanika Block Cave in order to maintain mill feed capacity in years 9 to 12.

25.7 Recovery Methods

The process plant design is based on a throughput of 8 Mt/y, or 22,000 t/d of mineralized material. The LOM average head grades fed to the plant are Cu 0.39%, Au 0.39 g/t and Ag 2.21 g/t. Mined material is crushed, ground, and processed in a flotation plant to produce a copper concentrate. The zinc flotation tailings are deposited in a wet. tailings storage facility.

The process plant flowsheet designs were based on a combination of industry standard practices and testwork results. The flowsheet was developed to optimize recovery while minimizing capital expenditure and life of mine operating costs. The plant uses process methods that are conventional to the industry. The comminution and recovery process are widely used with no significant elements of technological innovation.

25.8 Infrastructure

The Kwanika-Stardust project includes on-site infrastructure such as civil, structural, and earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, power supply, and concentrate transportation. The site infrastructure will include:

- mine facilities, including mining administration offices, a mine fleet truck shop and wash bays, a mine workshop, and a mine water treatment plant
- common facilities, including an entrance/exit gatehouse, a security/medical office, overall site administration building, potable water and fire water distribution systems, compressed air, power generation and distribution facilities, diesel reception and combustion plants, communications area, and sanitation systems
- process facilities, housed in the processing plant, including crushing, grinding and classification, flotation, product regrind, concentrate handling, thickening, dewatering, and filtration, reagent mixing and distribution, assay laboratory and process plant workshop and warehouse
- other infrastructure includes the on-site man-camp, TSF and WRSF.

The project can be reached via an existing FSR between Fort St. James and Tsayta Lake Road. The Tsayta Lake Road itself will be improved along a 26.5-km length to meet operational requirements and allow for the delivery of bulk freight by tractor-trailer units.

A 230-kV transmission line will connect to an on-site substation before being stepped down to 25-kV for distribution to different power requirements across the site.

Freshwater will be sourced from the Kwanika river and wells and pumped via a 404-m insulated pipeline to the process plant where freshwater tanks will be located.

The overall site layout was developed using the following criteria and factors:

- The facilities described above must be located on a site within the Kwanika-Stardust project boundary.
- The location of the process plant must be close to Kwanika open pit and underground mine which is the major source of feed, to reduce haul distance but outside of the 500 m blast radius.
- The location of the WRSF must be close to the open pits to reduce haul distance.
- The location of the primary crushing and ROM stockpile must be close to the Kwanika deposits to reduce haul distance.
- The TSF should be located at a site that takes advantage of sloped natural terrain to adequately drain entrained water and reduce earthworks, concrete, and structural development if possible.
- The arrangement of the administration buildings, mine workshops, processing plant, and additional offices should be optimized for foot and vehicle traffic.

25.9 Environmental, Permitting and Social Considerations

A number of limited field and screening environmental baseline studies and reports were completed in 2018 and 2019. The programs involved the collection of baseline data within the proposed project footprint area (as of 2018) and commenced the process of identifying potential environmental constraints and opportunities related to the proposed development of the project, including engineering designs and management plans for the construction, operation, and closure phases of the project. The reports also outlined recommended next steps for baseline data collection.

From a study area perspective, the baseline environmental studies were focused mainly on the areas potentially impacted by the Kwanika deposits and little information is available for the Stardust deposit area where underground development is proposed. In addition, there have been no baseline studies completed to date on air quality, noise, greenhouse gases and climate change, and groundwater quality, Ongoing and expanded baseline studies will be required to support the project through the feasibility and EA/permitting phases of the project. The results of baseline studies will be used to minimize impact of the project on valued ecosystem components and to optimize the location and operation of project infrastructure. Baseline study recommendations for the purpose of advancing the project to the PFS stage are provided in Section 26.11.

In terms of water management, the main consideration for the project is related to changes to the flow regime of Kwanika and West Kwanika Creeks which will require diversion around open pits and loss of fish habitat which will require fisheries authorization and habitat compensation measures. The project as envisioned in this report will require a Fisheries Authorization and Fish Habitat Compensation Plan. A Schedule 2 amendment to the MDMER may also be required subject to further fish and fish habitat surveys required for areas where mine waste will be stored (waste rock, tailings, mineralized material, and untreated contact water/mine effluent).

There are traditional land use activities undertaken by Takla and others First Nations near the project area. The potential interactions of traditional use sites with the project will need to be addressed during future stages of the project.

As additional baseline data is collected and community engagement efforts proceed, changes to project infrastructure design (and estimated costs) may be required at the PFS and future stages including permitting due to the following:

- fish and fish habitat characteristics for the areas of proposed project disturbance as related to future permitting requirements and risks
- refined understanding of hydrological and hydrogeological conditions as related to water balance
- the quality of mine contact water based on geochemical characterization and predictions
- refined understanding of vegetation/ecosystem and wildlife/wildlife habitat
- traditional land use activities undertaken by Takla and other First Nations near the project area
- locations of archaeological sites and cultural importance to First Nations.

Main risks associated with permitting the project include:

- Lack of cooperation and support of First Nations, and
- Unanticipated impacts to fish and fish habitat that cannot be readily compensated for, resulting in difficulties in receiving Fisheries Authorization and/or MDMER Schedule 2 amendment.

The implementation of the recommendations presented in Section 26.10 will help to quantify, qualify and mitigate these risks.

25.10 Markets and Contracts

The concentrate will be trucked from the project site to Mackenzie, where it will be loaded in lots onto bulk material carriers and then transferred onto railcars of the CN Railway to port storage facilities at Vancouver Wharves in North

Vancouver. The concentrate will subsequently be transported by sea to clients. Concentrates will be sold into the general market to North American, European, or Asian smelters and refineries.

For this technical report, the metal prices presented below in Table 25-1 were used for financial modelling. The metal prices are long-term forecasts over three years provided by financial analysts and as agreed by NorthWest Copper.

Table 25-1: Price Projections

Metal	Commodity Unit	Unit Price (US$)
Copper	Pound (lb.)	3.63
Gold	Troy ounce (oz.)	1,650
Silver	Troy ounce (oz.)	21.50

There are currently no sales contracts or refining agreements in place for the project. The marketing terms used in this study and addressed in this part were based on a preliminary marketing study.

The metal payables used in the marketing study are given below in Table 25-2. A summary of the treatment, refining, and transportation costs is provided in Table 25-3 and Table 25-4.

Table 25-2: Metals Payables

Metal	Unit	Concentrate
Copper	%	96.0%
Less Deductible	%	0
Silver	%	70%
Less Deductible	g/t	0
Gold	%	97.5%
Less Deductible	g/t	0

Table 25-3: Transportation and treatment cost

Concept	Value	Unit
Transportation Cost	$ 100.00	per wmt
Treatment Charge	$ 75.00	per dmt

Table 25-4: Refining Charge

Refining Charge	Value	Unit
Copper	$ 0.075	per lb
Gold	$ 5.00	per oz
Silver	$ 0.40	per oz

There are no known deleterious elements that could significantly affect a potential future economic extraction. The QP is of the opinion that the information presented here is suitable for use in cashflow analyses to support this assessment.

25.11 Capital Cost Estimates

The capital and operational cost estimates provided in this PEA offer expenses that can be utilized to evaluate the Kwanika-Stardust project's preliminary economics. The calculations are based on a combination of open pit and underground mining operation, the development of a Processing Plant, Infrastructure, a Tailings Storage Facility and Management Facility, and the Owner's expenses and provisions.

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with a +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q3 2022 based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

The total initial capital cost for the Kwanika-Stardust project is C$567.9 million, the LOM sustaining cost is $282 million, and the growth capital is $493 million. The capital costs are summarized in Section 21.

25.12 Operating Cost Estimates

The operating cost estimate was developed in Q3 2022 using data from projects, studies, and previous operations from Ausenco's internal database. The operating cost estimate is around +50%/-30% accurate. The estimate covers the TSF, mobile equipment, G&A, and mining and processing. Section 21 includes a summary of the operating expenses.

The unit operating cost per tonne of material milled is $23.04 and the LOM operating cost is $2.2 billion.

25.13 Economic Analysis

The economic study was conducted on the base case assumptions of US$3.63/lb for copper, US$1,650/oz for gold and US$21.50/oz for silver. These metal prices were computed using current economic research and consensus analyst estimations. The forecasts used are meant to reflect the average metals price expectation over the project's life. The effects of price escalation or inflation were not considered. Therefore, the price of a commodity may fluctuate, and there is the potential for variation from the forecast.

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

The economic analysis was performed assuming a 7% discount rate. The pre-tax NPV discounted at 7% is C$440 million; the IRR is 17.1%, and payback period is 6 years. On a post-tax basis, the NPV discounted at 7% is C$215 million, the IRR is 12.7%, and the payback period is 6.4 years. Section 22 includes a summary of the economic analysis.

25.14 Risks and Opportunities

25.14.1 Project Risks

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

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

25.14.1.2 Mineral Resources

- Commodity price falling below the assumed price
- Assumptions that all required permits will be forthcoming
- Actual metallurgical recoveries being lower than assumed recoveries in the resource estimate
- Significant increase in mining and process cost than the current assumptions
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

25.14.1.3 Mine Plan

25.14.1.3.1 Kwanika Underground Mine:

- increased costs materially above what is assumed in this study
- higher rock mass competency forcing a reduction in cave draw rates and productivity
- large seasonal water inflows which could cause mud rushes.

25.14.1.3.2 Stardust Underground Mine

- geotechnical and hydrogeological assumptions
- ability for mining operation to meet rapid development rates.

25.14.1.3.3 Open Pit Mines

Risks to the mine plan include changes to the following factors and assumptions.

- metal prices
- CAD/USD exchange rate
- interpretations of mineralization geometry and continuity in mineralized zones
- geotechnical and hydrogeological assumptions
- ability of the mining operation to meet the annual production rate and anticipated grade
- operating cost assumptions

- dilution, mine operation and process plant recoveries.

25.14.1.4 Metallurgical Testwork

- The majority of the metallurgical testwork was performed on samples which are representative of higher grades than the life of mine average feed grades. There is a risk that the recoveries for the lower grade feed material will not be achieved and will negatively impact the project economics. Further testwork is required to confirm recovery estimate at expected feed grades.

25.14.1.5 Recovery Methods

- Flowsheet development is based on the adjustment of results of historical testwork that was completed at a primary grind size of 75 µm to account for a primary grind size of 100 µm. There is a risk that the increase in primary grind size may decrease recovery more than has been predicted and could negatively impact the project economics presented. Future testwork needs to be performed to confirm the effect of coarser primary grind sizes.
- The process design is based on limited variability testwork, particularly on Stardust. There is a risk that the process design may not achieve the throughput, or the recoveries predicted which could negatively impact project economics. Further testwork is required to manage and mitigate these risks.

25.14.1.6 Geotechnical

- Ground conditions, geological containment, and stability of the proposed TSF area are unknown as a geotechnical program has not been completed.
- There is a possibility for cost increase if the geotechnical or hydrogeological considerations for the TSF are different from the criteria considered in this study impacting the capital, sustaining capital and operating costs of the project.

25.14.1.7 Site Water Management

- Riprap and erosion control requirements may differ materially from what is assumed in this study.
- Subsurface conditions and assumed excavation depths may differ from what has been assumed in this study.
- Diversion channels design and site water balance is based on available data from Environmental Canada and Water Survey of Canada. Data is evolving and may not be representative of future climate conditions.

25.14.1.8 Environment and Permitting

- Canada has a rigorous regulatory process for permitting that involves many stakeholders that could impact the timing and approval of the Project.
- Unanticipated impacts to fish and fish habitat that cannot be readily compensated for, resulting in difficulties in receiving Fisheries Authorization and/or MDMER Schedule 2 amendment.
- The implementation of the recommendations presented in Section 26.10 will help to quantify, qualify and mitigate these risks.

25.14.1.9 Logistics

- The concentrate transportation cost in the PEA is based on projects in the nearby region which may vary and impact the project economics.
- The study considers transportation of final copper concentrate to the Port of Vancouver Wharves. No studies have been conducted or discussions have been held to confirm adequate capacity at the port to handle the product from the project.

25.14.1.10 Contracts

- There are no contracts established with any equipment suppliers, power or fuel suppliers, and marketing companies. Equipment quotes were received for the mining fleet and major process equipment, however, the prices are subject to vary at the time of project construction and execution.

25.14.1.11 Marketing Terms

- The marketing terms considered in this PEA are based on projects with similar commodities. No marketing study was completed, or any discussions were held with the marketing companies in determining the marketing terms.

25.14.1.12 Financial Analysis

- The economics analysis has not considered the risk of the project to metal price fluctuation, inflation or other unexpected events such as COVID that can significantly impact the economics and schedule.

25.14.1.13 General

- In addition to the risks and uncertainties mentioned above, the property is subject to the typical external risks that apply to all mineral exploration projects, such as changes in copper, silver, and gold prices, and volatility of supply and demand economics which can affect the availability of investment capital as well as changes in government regulations, community engagement and general environmental concerns. The authors are unaware of any unusual risk factors, other than the ones mentioned above and risks normally associated with mineral exploration that might affect future exploration work and potential development of the property.

25.14.2 Project Opportunities

25.14.2.1 Exploration

- There is significant potential at Stardust for expanding the current resource and for discovering additional mineral deposits on the property and extensions to known mineral showings.

25.14.2.2 Metallurgical Testwork and Recovery Methods

- Preconcentration (ore sorting) has the potential to improve project economics and decrease tailings volume
- Further metallurgical test work on mineralized material from Kwanika-Stardust to optimize metallurgical recoveries.

25.14.2.3 Mine Plan

25.14.2.3.1 Kwanika Underground Mine

- Evaluate various cave initiation locations and cave face advance geometries to bring forward grade.

25.14.2.3.2 Stardust Underground Mine

- Operating costs being lower than expected based on synergies between operations.

25.14.2.3.3 Open Pit Mines

- The mine plan is partly based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserve, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The following opportunities have been identified as they relate to underground and open pit mining:

- Improving the financial outcome for Kwanika Central by optimizing the pit limit considering the Block Cave mining.
- Reducing waste rock haulage costs ($/tonne) by dumping waste in depleted South Pits rather than hauling it to the waste dump.

25.15 Conclusions

The PEA completed on the Kwanika-Stardust project demonstrates positive economics with combination of open pit and underground mining, a 22,000 t/d Cu Flotation Plant and a conventional wet TSF. Based on the assumptions and parameters presented in this Report, the PEA shows positive economics (i.e., $215 M post-tax $NPV_{7\%}$ and 12.7% post-tax IRR). The PEA supports a decision to carry out additional detailed studies to progress the project further into detailed assessment.

The risks stated in Section 25.14.1 may have an impact on the project economic outcomes which have not been considered in this PEA.

26 RECOMMENDATIONS

26.1 Overall Recommendations

The Kwanika-Stardust project is technically viable and demonstrates positive economics, as shown by the results presented in this Technical Report.

It is recommended to continue developing the project through prefeasibility study.

Table 26-1 summarizes the estimated cost for the recommended future work on the Kwanika-Stardust project.

Table 26-1: Cost Summary for the Recommended Future Work

Recommendation	Cost (CAD$)
Project Management	200,000
Mineral Resource (Including drilling)	6,000,000
Geotechnical & Hydrological Studies	2,300,000
Mine Engineering	450,000
Metallurgical Testwork	620,000
Process and Infrastructure Engineering	500,000
Geotechnical Studies (Infrastructure)	840,000
Topography	100,000
Geochemical Assessment	110,000
Water Management Studies	100,000
Environmental Studies and Permitting	300,000
Total	**11,520,000**

26.2 Mineral Resource Estimate

26.2.1 Kwanika Central and Kwanika South

The results of the recent exploration work clearly demonstrate that additional exploration is warranted. Drilling should continue at both Kwanika Central and Kwanika South to potentially extend and upgrade the mineral resources. Estimated cost for this task is $2,500,000.

26.2.2 Stardust

The results of the recent exploration programs clearly demonstrate that additional exploration is warranted. The program should continue to focus on expanding the CCS zone as well as testing for additional skarn lenses along the siliciclastic sedimentary – carbonate contact. Infill drilling should be carried out to upgrade Inferred resources to Measured or Indicated categories. Advanced metallurgical testing should also be carried out. Specific recommendations for a first phase program include:

- Resource expansion drilling to potentially expand the mineral resources within the CCS Zone

- Infill drilling to potentially upgrade inferred mineral resources to measured or indicated
- Further metallurgical testing including comminution testing, locked cycle tests on main rock types, variability testing and detailed concentrate analysis to identify any potential deleterious elements that might impact marketability of the final concentrates.

The cost to implement the above recommendations is estimated to be $3,500,000.

26.3 Geotechnical and Hydrogeologic Studies

26.3.1 Kwanika

A review of the available geotechnical dataset in support of the Kwanika underground block cave mine plan indicates a need for further geotechnical and hydrogeologic information to advance to a PFS-level study. Specific recommendations include:

- Expand the rock mass classification and geotechnical domains to encompass the projected mine development area
- Complete drilling investigations for the proposed portal and decline route including sterilization drilling
- Deepen drilling investigations to investigate opportunities to deepen or split production elevations
- Complete in-situ stress measurements
- Update the structural interpretations and model facilitated by acoustic televiewer surveys on existing and future drillholes
- Complete hydrogeological modelling to increase confidence in water inflows from the various geological units during mine development and production.

The cost to implement the above recommendations is estimated to be $1,100,000.

26.3.2 Stardust

A review of the available geotechnical dataset in support of the Stardust underground mine plan indicates a need for further geotechnical and hydrogeologic information to advance to a PFS-level study. Specific recommendations include:

- Complete geotechnical drilling investigations, guided by the location of major ramps, capital infrastructure and production areas to collect rock mass classification data, structural data and samples for rock material property testing. Structural information collection should be augmented with the application of geophysical acoustic televiewer surveys
- Complete Matthew's Stability graph stope size optimization studies as well as Equivalent Linear Overbreak Slough (ELOS) assessments to refine production scheduling and stope dilution estimations
- Develop geotechnical domains based on and updated structural model and geologic data
- Complete in-situ stress measurements
- Complete empirical and numerical modelling analyses to refine pillar sizing estimates for the mine plan

- Complete hydrogeologic investigations to evaluate the impact of water flows on the development and production areas.

The cost to implement the above recommendations is estimated to be $1,200,000.

26.4 Mine Engineering

The following work is recommended:

- A study to better understand and quantify the pit internal dilution
- A material handling trade-off study should be conducted to determine the optimal plant location and optimal waste storage locations based on haul route cycle time analysis for the Kwanika deposits
- Study mine automation such as automated truck hauling and mucking between shifts
- Geotechnical and hydrogeological study for Kwanika Underground
- Further block cave footprint and production schedule optimization for Kwanika UG in advanced stages of study
- First principles study for Stardust Mining and G&A costs
- Geotechnical and hydrogeological study for Stardust.

The recommended budget for these works is $450,000.

26.5 Metallurgical Testwork

The metallurgical work outlined below is recommended for the next project phase and should be completed on samples originating from ½ drill core.

- Sample selection for future mining studies should reflect mineralization that would be treated throughout the mine life. Variability samples are required to understand the responses of the various mineralized zones
- Feed mineralogy tests
- Additional comminution tests to further expand the comminution database is recommended to develop a robust comminution model and grinding circuit design. This will improve the future analysis of power requirements and equipment selection. The comminution tests would include SMC, RWi, BWi and Ai tests
- Bench scale flotation tests including rougher, cleaner, and locked cycle tests. The testing should investigate the potential to apply a primary grind sizing that is coarser than 75 µm P_{80}
- Bench scale tests for assessment of preconcentration technologies such as coarse particle flotation and sensor sorting
- Low levels of Mo were identified in the samples, expected Mo levels in the resource should be confirmed and appropriately represented in future test samples. If adequate Mo levels are present, bulk concentrate generation and subsequent Cu-Mo separation tests should be arranged.

The estimated cost of the above recommended metallurgical testwork and the cost of testwork management is $620,000.

26.6 Process and Infrastructure Engineering

The estimated cost for process and infrastructure engineering for the PFS is $500,000. Engineering deliverables would include:

- process trade-off studies (comminution, cyanidation options and preconcentration studies)
- flow diagrams (comminution, recovery processes, tails)
- detailed equipment list
- power listing and consumption estimate
- architectural (building sizes) to estimate steel and concrete quantities
- detailed material and water balance
- detailed process design criteria
- GA and elevation drawings (for crushing/overland conveying, comminution, leaching, recovery, reagents)
- electrical single line drawing
- equipment and supply quotations updated, and sources determined
- estimate of equipment and materials freight quantities
- capital cost estimate
- operating cost estimate
- major equipment spares and warehouse inventory cost estimate
- construction manpower estimate
- construction schedule.

The following activities are recommended to support infrastructure design for the PFS phase.

26.7 Site-wide Assessment and Tailings Storage Facility Studies

Due to the conceptual nature of this study and the paucity of information available at the time of writing, assumptions have been made regarding the layout, MTOs, and construction of the proposed TSF. Construction material geotechnical properties are required to perform slope stability analyses and other geotechnical assessments to confirm that the TSF can be built as designed. A tailings deposition plan will be required which may lead to the conceptual staging requiring adjustment to contain the given capacities.

Additional studies and data collection will be required to advance project development beyond the conceptual level. Some, but not necessarily all, of the current data gaps that would need to be addressed in future studies include the following:

- Geological and geotechnical site investigations and laboratory program should be carried out for infrastructure, Process plant, WRSF and TSF, including drilling and in-situ and laboratory testing, to understand subsurface soil and rock characteristics, construction material properties, and existing groundwater levels
- Seepage analysis for the TSF needs to be investigated

- Additional geotechnical testing of the anticipated tailings, waste rock, and other associated construction materials, (e.g., horizontal drain gravel and sand and candidate geomembranes) should be carried out
- Hydrological information should be gathered from site-specific climate studies to detail ponds and channels
- Hydrogeological information from desktop studies and site investigations should be gathered to better understand subsurface flow regimes
- A trade-off study between dry stacking of tailings vs conventional disposal of tailings.

As additional information is obtained, assumptions made in this study can be verified or updated to advance the project to the next level of design. The cost of implementing the above recommendations is estimated at CAD $840,000.

26.8 Water Management Studies

- A detailed site-wide stochastic water balance modelling using GoldSIM (Monte-Carlo) should be completed for the next phase of the study: This would inform of potential needs for water treatment and support future financial analysis. Estimated cost for this item is $50,000
- Packer testing should be conducted to determine pit hydrogeology, hydraulic conductivity and refine pit water inflow estimates
- Further hydrogeological and hydrological characterization are required in the pit areas
- Updating the water management structure design based on the updated site layout. Estimated cost for this item is $50,000.

The cost of carrying out the above work is estimated at CAD $100,000.

26.9 Geochemical Assessment

For proceeding to a PFS-level study, the general level of effort required to establish the ARD/ML risk for a typical project would generally comprise:

- Around 200 – 300 waste rock samples
- Six to 12 tailings samples (if composition different)
- Six to 12 mineralized rock samples
- Several overburden samples
- Range of tests to include:
    - Elemental analysis
    - Acid base accounting
    - Shake flask extraction (short term leach)
    - NAG pH
    - Mineralogy
    - Humidity cell testing (minimum 40 weeks).

The estimated cost for the recommended lab testwork is $80,000.

1. To better assess the ARD/ML risk from tailings, confirmation of the type of tailings streams (i.e. spiral / flotation / cyanidation) and the percentage ratios of each type that will be deposited in the tailings storage facility
2. If available, the results of testing of historical mine wastes and site water quality data should be reviewed as this can provide useful supporting information to aid in assessing the existing geochemistry data.

The estimated cost of assessment is $30,000.

The total cost for geochemical assessment is $110,000.

26.10 Topography

A site-wide LiDAR survey is recommended to define the site topography at higher accuracy. The current topography is based on SRTM which is sufficient for PEA, however, higher definition will be required in the PFS. The estimated cost for this task is $100,000.

26.11 Environmental, Permitting, Social and Community Recommendations

The following recommendations are made regarding future studies and activities related to areas of environment, permitting and community engagement. These studies and activities will be necessary to support the project to the PFS stage and provide a strong basis for future EA preparation and permitting.

26.11.1 Water Resources

- Development and implementation of s hydrological and meteorological monitoring plan for key areas within the study area will be required to further characterize the hydrological conditions and to develop a future water balance model
- Development and implementation of a surface water and groundwater monitoring, sampling, and testing plan and focusing on areas that will be potentially affected by mine infrastructure based on current infrastructure plans (refer to Chapter 18) and should meet the requirements of an Environmental Assessment application (ENV 2016). The surface and groundwater baseline and testing data will need to support the development of a future integrated numerical 3-D groundwater predictive model and overall water balance model for the site
- Hydrogeological testing of monitoring wells to support groundwater inflow estimates for pits and underground workings
- Development of a conceptual groundwater model
- Estimated cost for these tasks is $150,000.

26.11.2 Geochemistry

- The geochemical testing results do not include all current deposits/pits being considered for development and further work is required for those areas, specifically the Kwanika South deposit and the Stardust deposit areas. Additional sample selection and analyses are recommended including for mine waste rock/tailings and mineralized material. Detailed description and costs for the proposed geochemical program is provided in Section 26.8, above.

26.11.3 Fish and Fish Habitat

- Additional fish and fish habitat sampling and assessments are recommended for the areas of proposed project disturbance
- Estimated cost for this task is $50,000.

26.11.4 Terrestrial and Wildlife Monitoring

- Additional surveys will need to be completed related to the areas of vegetation/ecosystem and wildlife/wildlife habitat for the mine infrastructure presented in Chapter 18. The results of those surveys should be used to develop plans that will eliminate or mitigate environmental risk for the purpose of PFS
- Takla and other land users should be closely involved in the development and execution of wildlife baseline studies, especially in relation to traditional and current use of the land for harvesting
- Estimated cost for this task is $40,000.

26.11.5 Socio-Economic, Cultural Baseline Studies and Community Engagement

- Archaeological Overview or Impact Assessment to be completed on locations of proposed project infrastructure.
- It is important that NorthWest Copper carry on with commitments previously made to Takla including:
  - Continuing to meet as a group for follow-up discussion on project plans
  - Developing TOR for defining collaboration process and procedures
  - Working towards defining communications, processes and procedures to guide the project through the next stages.
- Estimated cost for this task is $50,000.

26.11.6 Environmental Constraints Mapping

- To assist in the development of the project at the PFS stage, environmental constraint mapping should be developed and continuously updated, based on the results of historical and future baseline environmental and land use studies. This mapping should be utilized to limit risks at the design stages of the project
- Estimated cost for this task is $10,000.

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